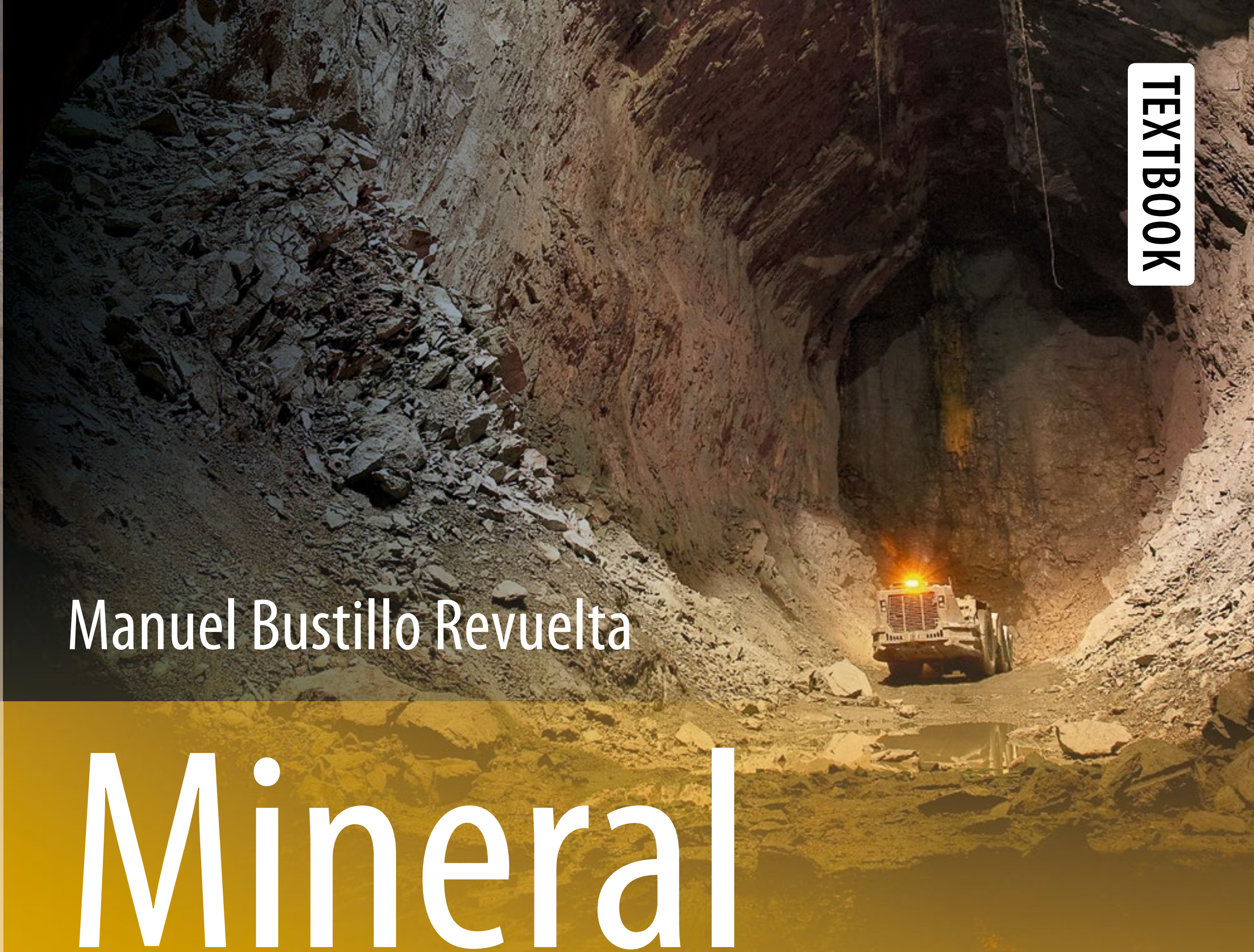

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Manuel Bustillo Revuelta

Mineral Resources

From Exploration to Sustainability Assessment

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This book is especially dedicated to my family:
Marian, Carolina, and Manuel Jorge.
They have been and are the castle and the keep.

Preface

This book represents an invaluable review of the whole subject of mineral exploitation and provides the reader with a comprehensive list of up-to-date references to which he/she can gain more specific information and guidance on the application of some of the methods and techniques described in the book. The author has clearly researched the subject matter of the book thoroughly. In particular, it provides excellent reviews of classification systems and philosophies for ore deposits and of international reporting codes and guidelines for mineral resources and reserves.

A well-illustrated chapter describing the main types of ore deposits provides an excellent basis for the following chapters which lead us systematically through the methods that could be used for their exploration, their modeling and evaluation, and their exploitation. The latter include the mining methods employed in open-pit and underground operations and then the recovery of the valuable minerals/metals from the ores produced. Numerous case histories further enhance these descriptions. The penultimate chapter considers the environmental impact of these operations and the methods that could be employed to minimize this impact and finally site reclamation. The final chapter describes some of the computer software packages available for the production and analysis of assay databases and grade and tonnage models in 2-D or 3-D and the design of mining operations and associated infrastructure.

I believe that this book will prove invaluable not only for undergraduates and postgraduates studying geology and mining geology but also those following courses in mining engineering and mineral processing who would benefit enormously from a solid background in exploration and mining geology and mineral economics. It will also provide a superb reference book for those intending to follow a career in, or are currently working in, the mineral economics or mining finance industry.

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Retired Principal Mining Consultant
Stratford upon Avon, UK

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The Author

Contents

Introduction and General Concepts

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Summary
This chapter explains the concepts and terminology important to mineral resources studies, from exploration to environment and sustainability. Important concepts include mineral resources/reserves classification systems, mining cycle and its main stages, international reporting standards, distribution of mineral resources in the Earth, mineral resources consumption, sustainable development, critical raw materials, mineral resource recycling, trade and markets, and mining as a business, introducing the London Metal Exchange market. A brief history of mining is also described as a starting point for this book.

1.1 Definitions

It is necessary to define some fundamental terms that will provide indispensable background for the entire book. The first term to be explained deals with the title of the book: mineral resources. A mineral resource can be defined broadly as the concentration of material of economic interest in or on the Earth's crust. In this book, it includes solid earth materials such as metals (i.e., copper, gold, iron), industrial minerals (e.g., fluorite, quartz), and rocks (e.g., limestone, sand, gravel). The reason to introduce the word solid is that some fuel resources, mainly oil and gas, are not solid materials, and their mining cycle (see ▶ Sect. 1.3) is completely different from other raw materials cited. This restriction is not valid for fuel resources that are solid ones (e.g., coal, tar sands, and bituminous shales) and whose exploration, evaluation, exploitation, mineral processing, and reclamation stages present similar guidelines that those involved for metals or industrial minerals and rocks. This more restricted view of the term, excluding nonsolid fuel resources, is the most commonly used in the field of mineral resources. World mining also includes many other common terms such as mineral, mineral deposit, ore, gangue, waste, prospect, commodity (fairly similar to mineral raw material), and much more. Some, but not all, used terms are defined because the list cannot be obviously exhaustive, and this book is not a mining dictionary. In this sense, many mining dictionaries can be downloaded from Internet web pages.

A mineral is «an element or chemical compound that is normally crystalline and that has been formed as a result of geological processes» (International Mineralogical Association). A mineral deposit can be defined in different ways, all of them very similar: a concentration of mineral of possible economic interest, a concentration of mineral resources profitable to extract (always in or on the Earth's crust), and many others. It is also necessary to bear in mind that a rock is a naturally formed aggregate of different types of crystals or mineral particles (◘ Fig. 1.1). Sometimes, the rocks can be profitable to extract, usually as industrial rocks (e.g., limestone for cement or granite for ornamental rock). In these cases, the term mineral deposit is usually applied.

Another essential term used in mining is ore (◘ Fig. 1.2). This word is applied solely to describe the material that is extracted for treatment. By definition, mines extract ore (Lane 1988) or solid substances currently recoverable at a profit. It applies to explored and developed deposits of metallic minerals, but the use in other nonmetallic minerals has long been discussed (e.g., Brown 1956). Whatever the case, the economic implication is always present. This economic implication was already established long ago: «technically, it (ore) is an aggregation of ore minerals and gangue from which one or more metals may be extracted at a profit» (Bateman 1950). Alternatively, ore is defined only as a concentration of mineralization, without the economic background, but this concept is not so common. On the contrary, gangue means the valueless mineral particles or crystals within an ore, while waste is the material that must be mined to obtain the ore.

Regarding the ore concept, it should be considered that the ore (or mineralization) has a specific grade, namely, the average concentration of the valuable substance (e.g., gold or tin) in a sample or in a mineral deposit. In general, the grade in metallic ores is expressed as a percentage or as grams per ton, which is equal to ppm (parts per million). The minimum grade in an ore needed to become a profitable extraction is called «cut-off grade» or simply «cutoff» (see ▶ Chap. 4). This concept is essential in all mining projects.

A prospect is a term mainly used in mining exploration. It can be broadly defined as a limited area of ground with a possibility to include a mineral deposit. It commonly receives the name of a geographical location. Finally, there are some terms that are also frequently used in mining, but they are combining words such as mineral occurrence, ore

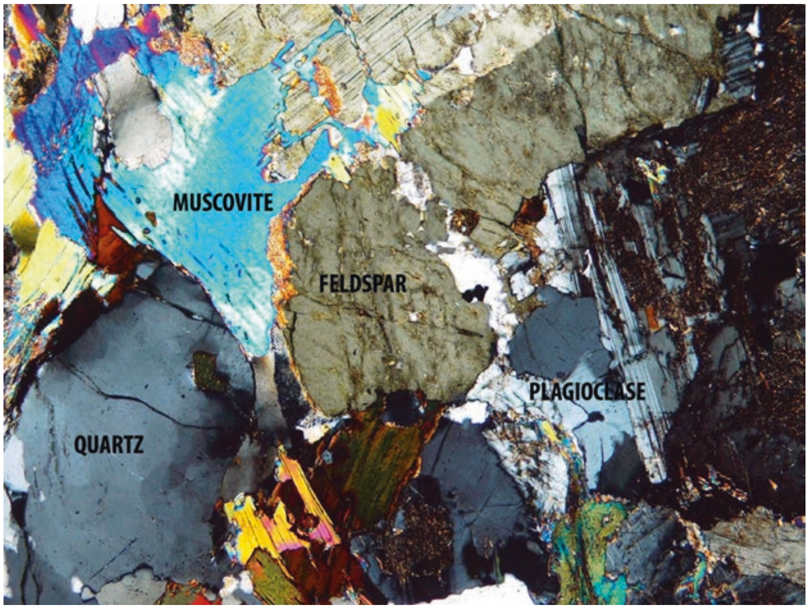


Fig. 1.1 Rock formed by crystals (Image courtesy of Pilar Andonaegui)

Fig. 1.2 Musselwhite ore formed by abundant pyrrhotite, quartz flooding and, rarely, visible gold (Image courtesy of Gold-Corp Inc.)

deposit (more or less similar to mineral deposit), ore reserves, mineral prospect, etc.

1.2 Mineral Resources/Reserves Classification: «McKelvey Box»

A mineral resource classification is used to organize information about raw materials or commodities of economic value. The classification systems can be grouped into three main categories, which are the most accepted by the industry (resource companies), the financial community, and the regulatory bodies: (a) classifications developed by government agencies (e.g., Geological Surveys): these classifications use a combination of both enterprise data and geological studies and are based on the «McKelvey Box» (Box 1.1: McKelvey Box); (b) classifications based on government and industry reporting: this group aims at capturing the full resource base in order to project future production potential for the country; this category includes classifications developed by ad hoc committees (e.g., the NI 43-101 for Canada, the SAMREC Code in South Africa, or the JORC Code – Australia and New Zealand); and (c) international classifications: these are developed at an international level to promote consistency of terminology and definitions; the most notable are the International Reporting Template of CRIRSCO and the United Nations Classification (UNFC). The first group is explained in detail in this section, while the groups (b) and (c) will be described in ▶ Sect. 1.5. A fourth class can also be considered, based on Security Disclosure (Edens and DiMatteo 2007).

Box 1.1

McKelvey Box

The most famous and widely cited resource classification scheme developed by Geological Surveys is the classification published at U.S. Geological Survey Bulletin (1450-A) entitled «Definitions of Mineral Resource Classification Terms used by the U.S. Bureau of Mines and U.S. Geological Survey». It was a joint report by the U.S. Bureau of Mines and U.S. Geological Survey in 1976. It includes the commonly known as «McKelvey Box» of mineral resources and reserves (◘ Fig. 1.3). In 1980, the Geological Survey Circular No. 831 entitled «Principles of a Resource/Reserve Classification for Minerals» revised and extended the previous classification system. The descriptions of both the documents are derived from a seminal work by V.E. McKelvey, director of the U.S. Geological Survey at that time, entitled «Mineral Resource Estimates and Public Policy» (1972). In the document, McKelvey declared «I have been developing over the last several years a system of resource classification and terminology that brings out the classes of resources that need to be taken into account in appraising future supplies». Other systems of classification were developed by the same time (Harris and Skinner 1982; United Nations Secretariat 1979), but the McKelvey Box was quickly converted in the main guide for many governments and markets, and its principles were widely accepted. This system of classification can be applied to a specific mine site, in a region, in a country, or in the world at large.

One of the most important features of this classification was to distinguish between identified deposits that are recoverable with existing technology and at current prices from those that are yet undiscovered or, if known, are not yet within economic research. Thus, the term «reserves» is applied only for identified deposits that could be produced commercially at the time the estimate is made. Before the McKelvey Box, a similar scheme that established the relations among economics, technology, and degree of knowledge to categorize the resources was outlined (Schurr and Netschert 1960).

In the most classical version of the McKelvey Box, known resources are classified from two standpoints: (1) purely geological or physical/chemical characteristics such as grade, quality, tonnage, thickness, and depth of the material in place; and (2) profitability analyses based on costs of extracting and marketing the material in a given economy at a given time, including technology, factor prices, and product price, among others. The former standpoint provides important objective scientific information of the resource and a relatively unchanging foundation upon which the latter more variable economic delineation can be based. According to these two main edges of classification, a resource is a concentration of naturally occurring solid, liquid, or gaseous material in or on the Earth's crust in such form and amount that economic extraction of a commodity from the concentration is currently or potentially feasible.

The other essential topic in the McKelvey Box is the concept of reserves. Mineral reserves in the McKelvey Box are identified resources known to be economically feasible for extraction. This is the portion of a resource that meets specified minimum physical and chemical criteria related to current mining and production practices, including those for grade, quality, thickness, and depth. It is important to note that, in general, «reserves» represents just a tiny fraction of the resources of any mineral or metal.

Obviously, the edges of this classification system are dynamic because the degree of geological assurance of a resource/reserve (increasing form right to left) and the degree of feasibility of recovery (increasing from bottom to top) can change (◘ Fig. 1.4). For instance, an increase in exploration effort switches the geological background of a region; an improvement of the technological recovery of a metal or a decrease in its price may change the economic characterization from subeconomic to economic. Therefore, what today is a noneconomic resource may be an economic resource 5 years later. This can result from increased demand and higher prices, cheapening of the real cost of labor and capital for a given technology, the adoption of newly conceived technology, of all of the foregoing. In this sense, resources can be created by man's economic activities and his scientific and engineering genius. Similarly, they can be destroyed by unfavorable economics, which includes the availability of low-cost foreign supplies and policies of taxation, trade, environmental protection, mineral leasing, among others (Harris 1984).

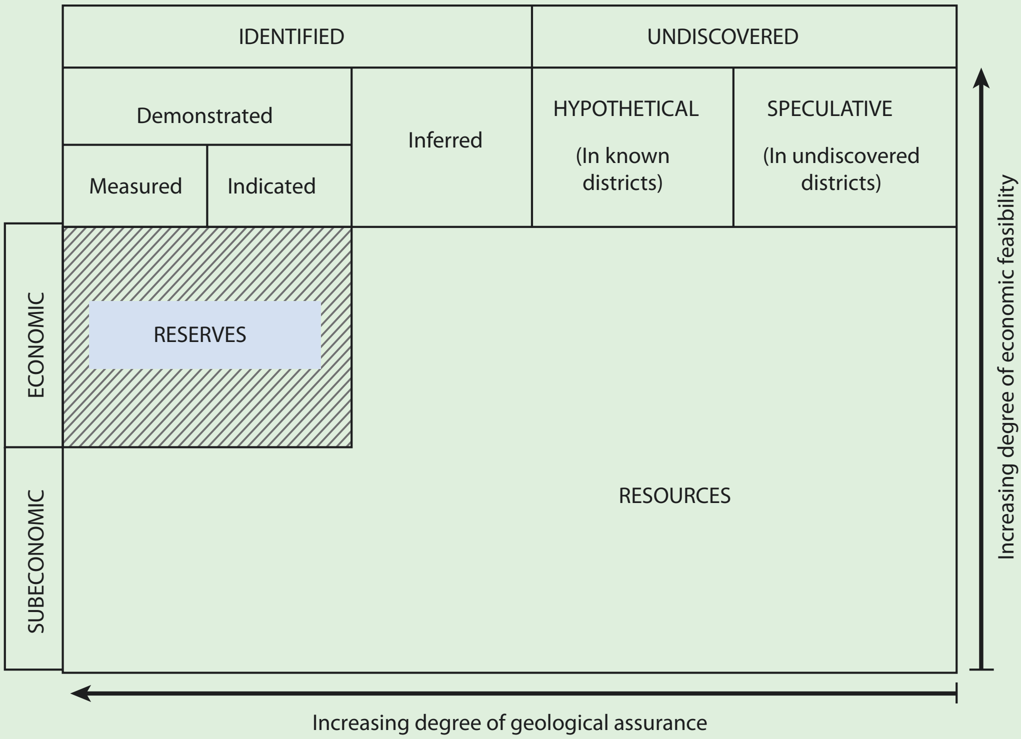


Fig. 1.3 McKelvey Box for classification of mineral resources

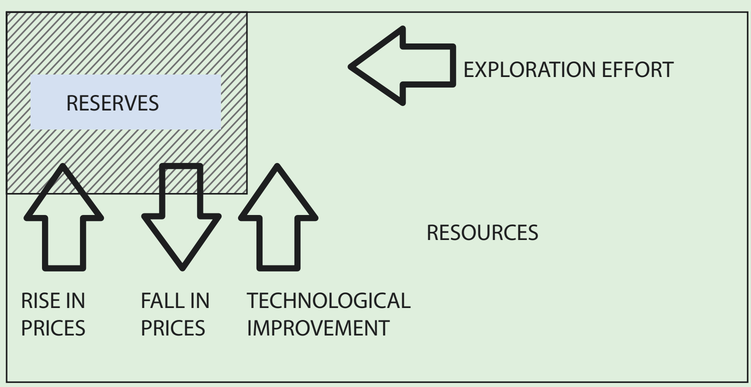


Fig. 1.4 Dynamic edges of McKelvey Box and some changing factors

1.3 A Brief History of Mining

Alongside agriculture, mining represents one of humankind's earliest activities and has played a major role in the development of civilization. Moreover, little has changed in the importance of these industries, and agriculture (including fishing) and mining (including extraction of any natural substances – solid, liquid, and gas) continue to supply all the basic resources used by modern civilization. Mining contributes to economic progress of nations worldwide improving the quality of life of people. A proof of the importance of mineral resources obtained in mines is the denomination of ancient times in the history of civilization such as Stone Age (prior to 4000 BC), Copper Age or Chalcolithic (considered a part of the Bronze Age), Bronze Age (4000–5000 BC), and Iron Age (1500–1780 BC), which means a sequence of ages linked to the complexity of mining. This also reflects the relevance of nonfuel minerals, metals, and materials technology and applications (e.g., the method to obtain an alloy). Over time, each metal discovered led to a range of innovations and applications that provided a marked advantage until they were adopted by competing civilizations or overtaken by other innovations. In fact, most known metals and metalloids were discovered in the last centuries.

The use of minerals has increased over the centuries in both volume and variety to respond to the demands of the society. Thus, present-day society is more dependent on the minerals industry than it was in the past. However, it is necessary to bear in mind that mining is a business, and, as such, it relies on the capital markets to operate. Without the stock market, there would be no exploration industry and without the larger financial institutions that support mine development, there would be no mining industry (Stevens 2010).

1.3.1 Pre-5000 BC

Humans mined hematite in Swaziland (Africa) about 43,000 years ago presumably to prepare the red pigment ochre, which was utilized as a cosmetic for personal adornment and cave paintings. This African mine is probably the oldest underground mine yet discovered. Coeval mines in Europe are believed to be sites where Neanderthals probably extracted flint to obtain weapons and tools. They had learned that certain stone (e.g., flint or obsidian) provided better workability and sharper cutting edges. Many Neolithic flint mines located in France, England, and Spain were developed for these purposes (Box 1.2: Neolithic Flint Mines). Some of the early flint mines consisted of vertical shafts 2 m in diameter and 20 m deep. It is apparent that miners got sufficient geological knowledge to

Box 1.2

Neolithic Flint Mine of Casa Montero

Flint has been a preferred raw material for tool making since the Paleolithic because when struck it splits or fractures in a reasonably predictable manner. This kind of fracture (conchoidal) generates strong, sharp cutting edges and allows the production of a wide variety of desired tool forms. Although seemingly of little relevance to the Neolithic, this fact is critical when assessing Neolithic flint mining. Neolithic flint mining in Europe was not a uniform phenomenon, at least not in terms of labor organization. There was considerable variability in labor intensity, the degree of elaboration of percussion tools, tool standardization, the scale of mining events, and the size of the workforce involved (Capote 2011). Neolithic miners were clearly able to adapt to the physical conditions imposed by the geological settings they encountered.

The Neolithic flint mine of Casa Montero (Madrid, Spain) (Fig. 1.5) was discovered in 2003 while performing the Archaeological Impact Assessment of Madrid's M-50 highway belt. The site, located on a river bluff south-east of Madrid, covers an area of at least 4 ha. The position dominates one of the main regional river basins, the Jarama valley, where some scattered Neolithic sites have been known to exist. The Neolithic flint works contain over 4000 documented vertical shafts, 1 m wide on average, and of up to 9 m deep, dependent on the variable depth and quality of

Fig. 1.5 Neolithic flint works in Casa Montero (Madrid, Spain) (Illustration courtesy of Proyecto Casa Montero, Spanish Research Council—CSIC)

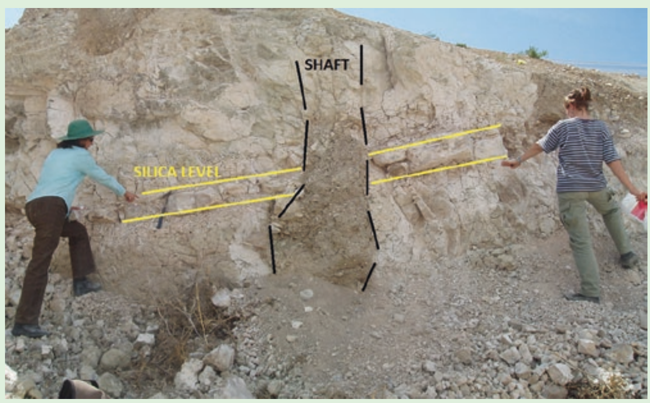
1

the flint seams. The lithic record is exceptional, both in quantity and quality, and includes all phases of the operative chain. A small percentage of these shafts contain chronologically diagnostic items, mainly impressed pottery and bone rings that suggest an Early Neolithic date. This has been confirmed by two radiocarbon datings.

The petrological characteristics and the features of the outcrops, formed by interlayered claystones and flints, may help to understand why intensive Neolithic mining was practiced at Casa Montero: the mine is unique in terms of its compact flint layers and their accessibility. Silica rocks from Casa Montero form nodules arranged in discontinuous beds that may have some lateral continuity. They appear deformed as a result of collapses of the underneath evaporitic episodes. This deformation produced a depression in which most of the shafts are concentrated, and may be particularly related to the horizontal depth variability of shafts throughout the site: as a general pattern, shafts are deeper in the central area of the excavation. Neolithic miners certainly had good knowledge of the geological structure of the area (▪ Fig. 1.6). They stopped excavating the shallow pits whenever they found green clay levels that are stratigraphically located under the opaline episodes. Furthermore, the depth of the shafts adapts to deformations resulting from siliceous episodes. This geological know-how would have been the result of a transmission of local mining knowledge from generation to generation.

Mining shafts offer little size and shape variability. They are mainly simple cylinder-like structures on average 1 m wide and up to 9 or 10 m deep. Their infillings show little differentiation, and few archaeological remains other than an impressive amount of flint. Shafts were dug close together, none superimposed on another, and with just enough distance so as to walk between them while avoiding wall collapses. Depending on the size of the flint nodule, and its position inside the shaft, nodules would have been extracted whole or quartered and extracted in large flakes. Those nodules small enough to be manipulated would have been directly removed from flint seams, while the bigger ones would have been fractured (Capote et al. 2006).

The soil extracted during the excavation of shafts was either dumped into other nearby shafts along with the remaining waste from flaking, or left aside and finally dumped back in. This of course depends on whether miners opened more than one shaft at a time. In any case, it seems that shafts were filled almost immediately after being excavated. The mine seems to have developed as the result of reiterative, short-term, seasonal mining expeditions. The shafts rarely cut into any previous extraction pits, suggesting that these more than 4000 Neolithic shafts are probably the result of several centuries of mining. The total mining intensity would have been about 13 shafts per year, considering a time-span of 300 years for the whole period of activity.



▪ **Fig. 1.6** Shaft cutting a silica level (Image courtesy of María Ángeles Bustillo)

prospect the presence of flint nodules overlain by sand and gravel. The raw materials were hauled from the bottom of vertical shafts to surface by one or two men loading in leather bags or wicker baskets. The first problem for early miners was probably to break the rock because their crude tools were made of bone, wood, and stone. They soon devised a revolutionary technique called fire setting, heating first the rock to expand it and then doused it with cold water to contract and break it.

However, the first mineral used by the humans was probably sodium chloride (salt or common salt), used to preserve foods since many microorganisms cannot live in a salty environment. An example of the relevance of salt is that the exploitation of this mineral was a privilege of the kings during many centuries. The Salt March of Gandhi in 1930 against the British monopoly of salt testifies the importance of the mineral even in very recent times. It is outstanding to note that part of the dating of this basic mining goes back beyond the time where the modern man appeared, probably between 50,000 and 100,000 years ago. It is thought that some of the mining activity was carried out by Neanderthal man. Such people were generally considered to be prehuman and thus incapable of any sort of sophisticated labor such as mining, as it was suspected that they would have been unable to instigate the planning necessary to develop a mine successfully (Coulson 2012).

Regarding the evolution of technology, crucible, which is a procedure essential in metallurgy, was probably discovered in Anatolia (actually Turkey) where tin deposits permitted alloying of this metal with copper to obtain bronze. Presumably, the crucible technology spread later to Middle East, Egypt, Persia, etc., providing the material for the metalworkers. Learning more on this subject is problematic because dating of ancient mining sites is often difficult due to relative lack of pottery and other datable material. The use of similar tools and techniques through long periods of history is an additional factor against easy dating of mining sites.

1.3.2 Egyptians

The earliest records of organized mining are those of the Egyptians. They mined different minerals (i.e., malachite or turquoise) and gold in northern Sudan and Israel; Nubian gold mines about 4000 years old were famous at that time. Moreover, Egyptian military expeditions, with mining and quarrying teams, looking for stone and copper in Sinai and in the Egypt's Eastern Desert were constant over the years. Mining in the Egyptian and early Roman periods was carried out by prisoners of war and criminals, being working conditions terrible. In a fragment of the book «Agatharchides's on the Erythraean Sea» (Agatharchides was a Greek historian and geographer who lived about 100–150 BC), this author describes the ancient mode of working the Egyptian gold mines: «The Kings of Egypt compelled many poor people, together with their wives and children, to labor in the gold mines, wherein they underwent more suffering than can well be imagined. The hard rocks of the gold mountains being cleft by heating them with burning wood, the workers then apply their iron implements … These are young men, under 30 years of age, strong and vigorous, who pound the broken fragments in iron mortars … and women, three on each side, work at it until it is reduced to a fine powder. These poor women are entirely naked … the excavations are of great extent, and reach down to the sea cost».

Gold was among the first metals mined in the antiquity. The reason is that it commonly occurs in its native form, not combined with other elements. Particularly noteworthy are the gold items in the tomb of Tutankhamun, a young pharaoh who ruled Egypt in the fourteenth century BC. Examples of the utilization of gold in jewelry can be located worldwide, being the metal par excellence of the human civilization. From Egypt and Mesopotamia, the knowledge of metals spread across Europe, and the copper-based cultures were replaced by cultures using bronze, about 1500 BC. This change produced improvement in weapons quality. Both production and trade in copper and bronze were important features of the Near East and Mediterranean societies during the third to first millennia BC (Jones 2007).

With the coming of Iron Age, mining took a step forward. In earlier mining stages, stone implements were the main tools for digging and breaking rock, but they were not robust enough, thus prone to become quickly unusable. Bronze was too valuable and too soft to be a realistic substitute in the making of such heavy-duty tools, but the emergence of iron introduced an altogether tougher and more durable metal, ideal for tool making. These developments proved of material help in the advancement of civilization, particularly in the hands of Romans (Coulson 2012).

1.3.3 Roman Empire

The Romans followed the Greeks as leaders of the then known world and were undoubtedly the best miners in the ancient times. They used hydraulic mining methods on a large scale in their gold mines, being the «ruina montium» method the most common system to obtain the gold included in alluvial deposits, as described by Pliny the Elder in 77 AD. (Box 1.3: Las Médulas Roman Gold Mine).

The Romans produced large quantities of other metals distinct than gold, especially lead, copper, zinc, and mercury. An example is the large-scale, industrial mining, and production of the copper

1

Box 1.3

Las Médulas Roman Gold Mine

Spain was the most important mining region for the Roman Empire, especially because of the gold metal. The largest site of Roman gold mines was Las Médulas (León, Spain), nowadays showing a spectacular landscape that resulted from the cited «ruina montium» mining method (Fig. 1.7). Pliny the Elder, who was a procurator in the region about 74 AD, described this technique of hydraulic mining that may be based on direct observation at Las Médulas: «What happens is far beyond the work of giants. The mountains are bored with corridors and galleries made by lamplight with a duration that is used to measure the shifts. For months, the miners cannot see the sunlight and many of them die inside the tunnels. This type of mine has been given the name of ruina montium. The cracks made in the entrails of the stone are so dangerous that it would be easier to find purpurine or pearls at the bottom of the sea than make scars in the rock. How dangerous we have made the Earth!» Newly, Pliny stated that: «about 20,000 roman pounds of gold were extracted each year and 60,000 free workers generated 5,000,000 roman pounds in 250 years», mostly travelling to Roma, capital of the Roman Empire. Present-day studies indicate that the number of workers probably fluctuated between 10,000 and 20,000. Now, Las Médulas Cultural Landscape was listed at 1997 by the UNESCO as one of the World Heritage Sites.

Las Médulas is an alluvium or secondary deposit made of alternate layers of boulders with clayey matrix, gravel, sand, and mud. These are all red deposits formed in the Miocene era and come from the erosion of rock or primary materials, located mainly towards the east near the Aquiliano mountains and constituting the mother rock in which the gold associated with quartz seams was to be found. Las Médulas mine was exploited from approximately 30/40 AD until the end of the second or beginning of the third century AD. It seems that the volume of Earth moved was about 93,550,000 m^3.

Regarding the «ruina montium» mining method, once prospecting had uncovered and valued the richness of the mine deposits, the exploitation consisted of cutting down the gold-bearing conglomerate. Thus, the conglomerate was mined using a set of galleries and shafts through which the water was released so as to cause their total collapse. This technique was applied by reaching only once the levels containing most gold. Water in Las Médulas was supplied by interbasin transfer (at least seven long channels coming from the mountains of La Cabrera district where rainfall was relatively high). The mining method, which involved undermining a mountain with huge volume of water, used the strength of torrents of water to wash enormous portions of alluvial sediments. The largest water reservoir (stagnum) had a capacity of 16,000/18,000 m^3. Thus, the bench collapsed, previously building a raft for retaining water that communicated with several galleries excavated inside the mountain.

The extraction activity ended with the mineral processing, probably by panning the sediments. Water was also important here, as after being used as extraction and dragging force for the conglomerate, the resulting flow was channeled towards wooden gutters in which gold particles were deposited by gravity. The last phase of the gold mining process was the evacuation of the waste material. The thickest material (the largest boulders) was stacked by hand in large piles. The finest wastes were evacuated out of the mine. The magnitude of these materials was such that they filled old valleys.

Fig. 1.7 Las Médulas Cultural Landscape (UNESCO World Heritage Site – 1997–)

Roman mines of Rio Tinto. Moreover, the Romans created the concept of cement, probably first by accident, and lime for mortar was developed by burning limestone. Even the volcanic ash from Pozzuoli was used to solve the problem of an excessive free lime in the mixture. During the Egyptian and early Roman Empire, the miners were generally slaves, criminals, and prisoners of war. Because slaves were plentiful, conditions in the mines were terrible. Some Roman shafts reached up to 200 m in depth, which tells about the painful and dangerous work in the mines.

1.3.4 China

Commercial relations between Europe and China have existed since prehistoric times. Probably the Silk Road was not only determined by the transport of this famous textile product, but also mining areas, minerals, metals, and many derivative products configured the Silk Road. More than 5000 years ago, it is documented that Chinese workers knew the principle of heat expand and cold contract for breaking the rocks to extract minerals or metals. In the Erligang period (1500–1300 BC), Erligang was the first archaeological culture in China to show widespread use of bronze vessel castings, being this culture centered in the Yellow River valley. It is important to bear in mind that the ternary alloy of copper, tin, and lead for manufacturing bronze is unique to China. Since all three constituents were deliberate and independent additions, and since all three have different properties, metallurgical knowledge was developed at a deeper level than with a binary alloy alone, typical in the rest of the world (Reinhardt 1997).

Located on the southern bank of the Yangtze River, Tonglüshan is the largest known ancient mining site in China, with shafts extending over an area 2 km long and 1 km wide; there was a rather long period of open-pit mining prior to underground mining. Evidence of smelting at the site suggests that local population initially processed the raw material before refined copper was transported elsewhere, possibly in the form of ingots. This site had also the longest period of activity, from 1100 BC to 200 AD, and the mined area exhibits more than 400,000 tons or ancient slags and hundreds of shafts and drifts in various structures, ranging in age from 1100 BC to 200 AD (Shelach-Lavi 2015). The underground mine at Tonglüshan consists of shafts which descend from the surface outcrop, and tunnels which extend from the shaft to follow the ore body. Through time, shaft and tunnel show enlargement in both cross section and depth as well as improvement in design.

The ancient Chinese bronze ritual vessels of the Shang and Zhou Dynasties (2200–770 BC) are exquisite works of art which have survived more than three and a half thousand years to the present day and are now artifacts that bear witness to the importance of bronze in China's first and second millennia B.C. (Reinhardt 1997). Bronze played a crucial roIe in the acquisition and maintenance of political power in ancient China.

1.3.5 Middle Ages

The fall of the Roman Empire in the fifth century was followed by widespread political and economic chaos (the Dark Ages). The rebirth of learning was under way by AD 1300 and three metals, antimony, bismuth, and platinum, were added to the six already in use (Wolfe 1984). Previously, another important discovery was made, first in China and then in Damascus. This technological progress was comparable to that resulting from alloying tin with copper many years before. Iron was alloyed with carbon and tempered to produce steel, earning for the Arabs the reputation of being invincible due to their steel swords.

Mining as an industry changed in the Middle Ages. Copper and iron were usually extracted by surface mining from shallow depths, rather than deep mines. In that period, the need of weapons, amour, and horseshoes (military purposes) manufactures with iron increased dramatically the demand of this metal. By the same time, in the early colonial history of the Americas, native gold and silver were sent back to Spain in fleets of gold and silver galleons, being the metals mainly used to pay Monarchy wars. Aztec and Inca treasuries of the Mexico and Peru civilizations, respectively, were plundered by the Conquistadors of the New World.

Towards the second half of the sixteenth century, many mines were developed from central Europe to England to extract different metals such as iron, zinc, copper, lead, and tin. The use of water power in the form of water mills was extensive. Black powder was first used for mining in Hungary at 1627, allowing blast the rock and look for the mineral veins. The use of explosives eliminated much of the arduous work needed to break the rock. In 1762, the first mining academy of the world was founded in Hungary. The book entitled *De Re Metallica* (Agricola 1556) is a classical text for medieval mining techniques. Agricola detailed many different mining methods, including the types of support used in the shafts during that period: «Now shafts, of whatever kind they may be, are supported in various ways. If the vein is hard, and also the hanging and footwall rock, the shaft does not require much timbering, but timbers are placed at intervals, one end of which is fixed in a hitch cut into the rock of the hanging wall and the other fixed into a hitch cut in the footwall … If the vein is soft and the rock of the hanging and foot walls is weak, a closer structure is necessary; for this purpose timbers are joined together in rectangular shapes and placed one after the other without a break».

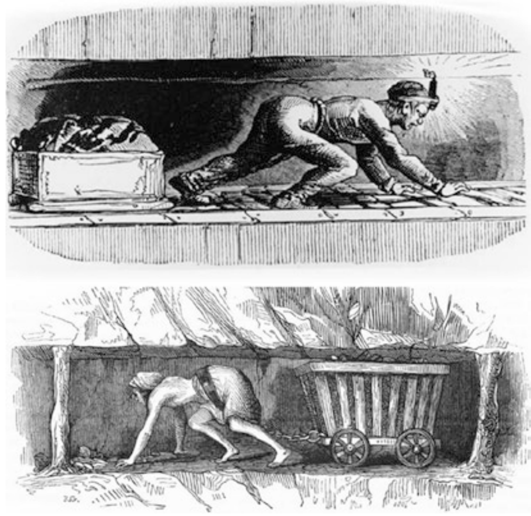


Fig. 1.8 Illustration of child labor in coal mines in the sixteenth century at Great Britain

1.3.6 Industrial Revolution

Industrial Revolution begins in Britain at the end of the eighteenth century. It produces an exponentially growth in the demand of metals, being coal the heart of the Industrial Revolution. Thus, one of the most important economic effects of the Industrial Revolution was the dramatic increase in the exploitation of mineral resources. For instance, pig iron production increased in England from 25 tons in 1720 to 2,000,000 in 1860, being this raw material essential to make steel through the Bessemer process. Some years before, as early as 1705, coal miners were using steam-powered pumps to remove water from deep mine shafts in coal mines in England; thus, pumps were the first modern machines used in mines. It is important to note that, at a certain depth in an underground mine, the water table is reached and drainage becomes priority. Later, the railroad locomotive changed the world industry, including the mining sector.

Unfortunately, the working conditions at the factories were, in general, terrible. In order to increase production, the average worker spent 14 h a day at the job, 6 days a week. Many women and children were employed in the mining industry because they were the cheapest source of labor; child labor, especially in coal mines, is one of the most important topics in this period (Fig. 1.8). In 1840, Lord Shaftesbury persuaded the Parliament to set up a Royal Commission to investigate conditions in the mines. Its report, published in 1842, was the first government report to use pictures (the Commissioners' Report was graphically illustrated with images of women and children at their work), and it deeply shocked the public. Consequently, in 1842, the Mines Act prohibited all underground work for women and girls, and for boys under ten.

The minerals economy was also affected by demand to fertilizer, and big sulfur deposits were opened to produce the acid to make phosphatic fertilizer from rock phosphate. Thus, potash mines were opened in Europe and North America to supply another ingredient to fertilizer. On the other hand, asbestos became an important industrial mineral and light metals were produced (e.g., aluminum and magnesium). The period from 1760 to 1900 was more dynamic than any previous time in the history of man and minerals.

1.3.7 Last Two Centuries

One of the most famous mining activity in the last two centuries, immortalized by numerous motion pictures such as Paint your Wagon at 1969, was the gold rush in California, which started in 1849. About two-thirds of the forty-niners (someone who went to California to find gold) were Americans, but also foreign miners came from Europe, South America, Australia, and China. To obtain the gold, miners used tools especially designed to separate the gold from the sediment around it since the gold in origin was washed by flood waters and concentrated into stream beds. The gold rush in California suddenly started and suddenly finished: by 1852, it was over. Meanwhile, Australia also suffered gold rushes; in 1850s, this area produced 40% of the world's production of gold.

According to ICMM (2012), «tracing the center of gravity of global mining over the past two centuries demonstrates its role as a foundation of society throughout history; the observation of the percentage of world mining by region from 1850 to the present day allows to conclude that: (1) by the late nineteenth century, the role of mining in Europe declined as the economic and political power passed to North America; the US in the late nineteenth and early twentieth centuries then saw a dramatic increase to be followed after Second World War by the same dramatic decline experienced previously in Europe, and (2) the shift of mining locations from developed to developing countries has been a trend from the mid-twentieth century».

During the last two centuries, there was great progress in mining technology in many different areas. In the nineteenth century, the invention of dynamite by Nobel and its application to mining was probably the most important advance. After the Second World War, the era of mechanized mining started. The intense mechanization of mines, including big trucks and shovels, permitted to develop very large exploitations, especially open-pit mines, many times in response to the introduction of economy of scale theory. Finally, as the twenty-first century begins, a globalized mining industry of multinational corporations has arisen and environmental impacts have become a continuous concern. On the other hand, a variety of raw materials, particularly minerals with rare earth elements (REEs), are increasing in demand because of the use of new technologies.

1.4 The Mining Cycle

Human societies need natural resources for their existence, including wood, water, and minerals, which are essential in the growth and prosperity of the modern way of life. It is necessary to construct roads with aggregates and bituminous materials, to build houses with concrete (formed principally by aggregates and cement, both obtained from minerals), or to manufacture cars with aluminum and steel. In the day-to-day, minerals are present everywhere, since the carpet for our feet very early in the morning (made with calcium carbonate, among others) or the coffee pot (made of either glass or ceramics), until the medicine or pharmaceuticals (many excipients are minerals). The communications equipment incorporates numerous minerals, for example, quartz or silica for the silicon chips in PC or coltan used in many digital products, including the cell phones. Finally, high-level technological products can incorporate more than 70 different metals.

Therefore, exploitation of minerals provides the necessary raw materials for manufacturing, construction, and chemical industries. To obtain mineral raw materials, the mining companies have to develop a complex, time-consuming, and high-risk process. However, the consumption generates sometimes harmful consequences, being an example the everyday violence in Congo. This is because the control of coltan production, essential when making the new generation of cell phones, since demand of coltan is growing exponentially in the last years. The so-called three T's, Tin, tungsten, and tantalum, which can be found in coltan, are the best known conflict resources produced in the Congo. In this sense, the OECD promoted at 2011 a Due Diligence Guidance. The Guidance aims to help companies respect human rights, observe applicable rules of international humanitarian law in situations of armed conflict, avoid contributing to conflict, and cultivate transparent mineral supply chains and sustainable corporate engagement in the mineral sector.

The distinct phases in successful mine development and production involves finding, outlining, and evaluating a mineral prospect, mine construction and exploitation, processing of run-of-mine (material obtained in the mine), and post-mining closure and reclamation. All these activities jointly form what is called the mining cycle or mining life sequence (Fig. 1.9). Each step in the sequence is unique and most mining projects proceed progressively from one step to the next. It is important to bear in mind that the timescale from discovery of a mineral deposit to mine production is generally a very long one (Fig. 1.10). For example, small mining projects may pass from exploration to mine production within a few years, followed by closure of the mine 10 years after the start of operation. On the contrary, large and complex mining projects may spend 20 years for exploration and several decades for mining. Regarding expenditures of the mining projects, overall expenditures can range from USD 100 million for small projects to several USD billions for large projects.

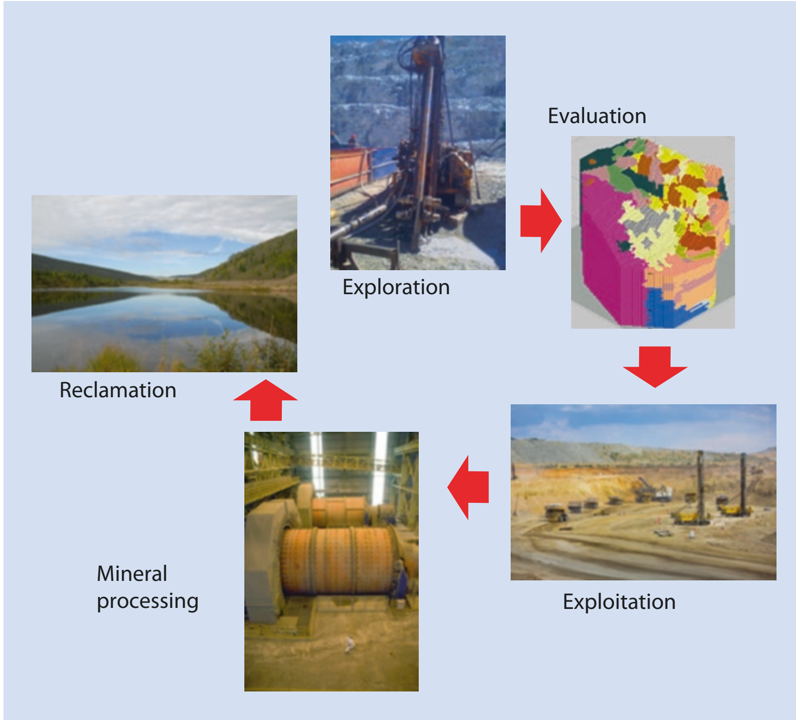


Fig. 1.9 The mining cycle

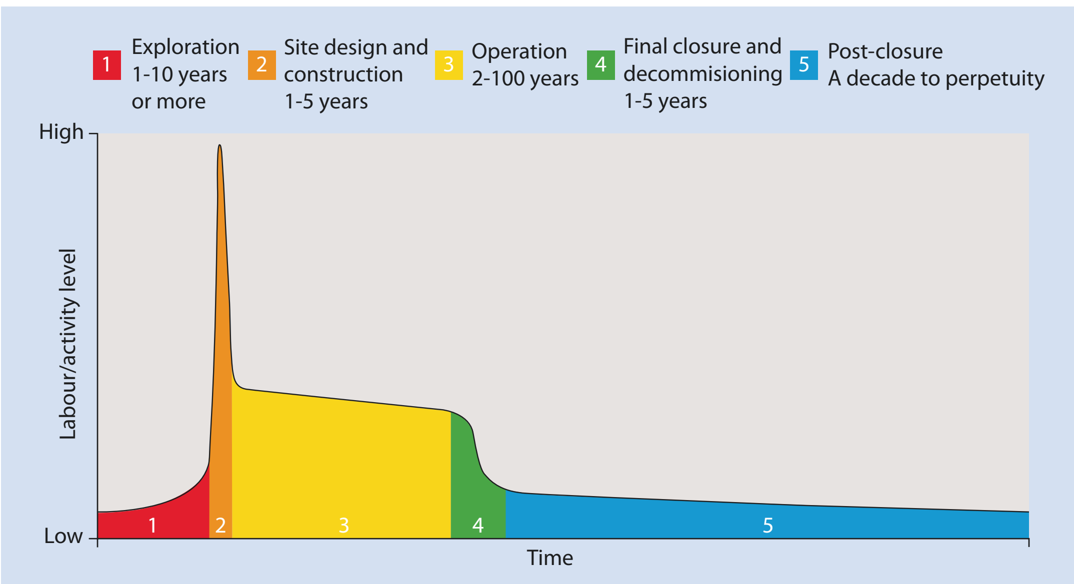


Fig. 1.10 The mine project life cycle (ICMM 2012)

1.4.1 Exploration

The search for mineral resources, usually called exploration or prospection, is the first step in the mine cycle and includes a complete sequence of multidisciplinary activities. Mineral resources are rare and they are buried beneath the surface of the Earth. Economic mineral resources are even rarer. Since mineral deposits are rare, finding one is challenging and the odds of success of any exploration program are relatively low. «Even where a mineral deposit has been defined, the probability of it becoming an operating mine is, at best, one in a thousand» (Stevens 2010).

At a preliminary exploration stage, large areas of possible economic interest (prospects) are evaluated by airborne or ground-based geophysical methods. From the obtained maps and existing data, specific areas are singled out for more detailed studies. A second stage involves specific surveys, including additional mapping, sampling (take a small representative portion of a larger mass), and drilling. Preliminary understanding of the deposit type facilitates the design of appropriate and effective exploration program. In this regard, mineral deposit models play an active role because these models provide a framework for research in economic geology as well as background for mineral exploration.

Prior to geological mapping, satellite imagery (science of acquiring, processing, and interpreting images obtained mainly from satellites) and aerial photography have proven to be important tools to define mineral exploration projects, providing reflection data and absorption properties of soils, rocks, and vegetation. This makes it easier to map terrain elevation, large-scale geological structures, like faults or geological contacts, and also to plan regional mapping or soil/stream sampling campaigns. In some regions such as deserts, color changes may denote variations in rock type or show places of rock alteration.

Used extensively for exploration for more than 100 years, geological mapping provides many types of information essential in exploration for new mineral resources. The map scales can range from 1:100,000 to 1:25,000 or even 1:10,000, depending on the stage of exploration, from regional-scale geological mapping to district-scale exploration targeting. Geological mapping of outcrops is also utilized to describe the lithology and morphology of rock bodies as well as age relationships between rock units and improves the utility of geophysical data for refinement of subsurface targets. Traditional paper-recorded geological mapping data are now commonly converted to digital format in the office and analyzed with GIS software.

Geochemical exploration, also known as geochemical prospecting and exploration geochemistry, as defined by Hawkes (1957) «… includes any method of mineral exploration based on systematic measurement of one or more chemical properties of a naturally occurring material». The purpose of the measurements is the discovery of a geochemical «anomaly» or area where the chemical pattern indicates the presence of ore in the vicinity. The database obtained in geochemical prospecting (the sets of data often contain thousands of observations with as many as 50 or more elements) provides an opportunity to discover a wide range of geochemical processes that are associated with underlying geology, alteration, or weathering and mineralization. The interpretation of database obtained in geochemical prospecting, commonly including thousands of data, requires special handling using univariate analysis combined to bivariate and multivariate analysis.

Another classic method used in mineral prospecting is geophysical exploration, working on principles of physics to study Earth. Interpretation highlights «signal» of mineral-related features under investigation or nongeological «noise». The geophysical anomalies must be explained geologically and can indicate possible occurrences of mineral resources. First, airborne geophysical surveys (▣ Fig. 1.11) provide the quickest and

▣ **Fig. 1.11** Airborne geophysical survey (Image courtesy of Geotech)

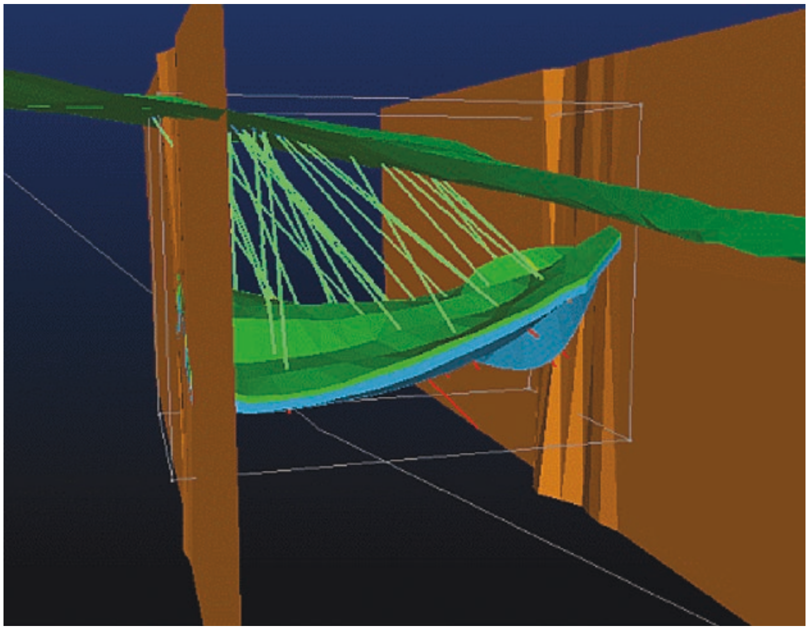


Fig. 1.12 Technical evaluation using mining software (Illustration courtesy of Datamine)

often the most cost-effective ways of obtaining geological information about large and unexplored areas. A more detailed ground survey is carried out, once a target area is identified, using techniques such as seismic surveys, direct sampling, and drilling.

Finally, exploration drilling is performed mainly from the surface with holes laid out on a prescribed grid or pattern. There are two main methods of drilling in mineral resources exploration: core drilling and reverse circulation drilling. Core drilling is the most commonly used method of getting information about the subsurface presence of minerals. This technique yields solid cylinder-shaped samples of the ground at an exact depth. The other method is reverse circulation drilling, which produces samples called chips, formed by small particles of sediment or rock. In reverse circulation drilling, the cuttings from the hole are transported to the surface where they are collected in plastic bags. This method offers higher productivity than diamond core drilling, but the quality of the samples is obviously lower. For this reason, diamond core drilling has long been the preferred choice of many exploration companies. Nevertheless, a combination of these two methods can often provide the optimum solution, offering the most cost-effective way of working. On the other hand, underground drilling, often drilling the holes at any angle, is essential to explore and define new mineral resources to mine in the future.

1.4.2 Evaluation

The mineral resource evaluation process commonly involves a technical and an economic stage, plus a socioeconomic one. Technical evaluation (Fig. 1.12) leads to the estimation of tonnage (quantity) and mineral or metal content (quality) from analytical data calculated in samples assays, either globally or for parts of the deposit. The estimation is obtained through classical or geoestatistical methods. The first methods are old style but easy to understand methods. Some of them were developed before the twentieth century, and examples of these methods are panel/section, polygons, inverse distance weighted, triangulation, and contour methods. The selection of the specific classical method can be modified based on the type and form of the material contained (Annels 1991). The uncertainties in determining the level of significance and confidence of traditional estimations with classical methods are overcome by application of geostatistical procedures (Matheron 1962), being this modeling method paramount in modern mineral resource estimation. The most important step in geostatistical procedure is the spatial correlation among samples (regionalized variables). It is expressed by the semivariogram, and the kriging technique, using the obtained semivariogram, let to interpolates the needed values (e.g., grades) for mineral resource estimation.

After technical evaluation, the selection of the most adequate economic evaluation method is of crucial importance. To carry out this economic evaluation, many other variables of the project such as production cost, capital cost, royalties, taxes, among many others, are also needed. It must be taken into account that mining industry presents different characteristics than other industries. For instance, mining industry needs many years of production before a positive cash flow and requires longer project life. Moreover, the overall process is extremely capital intensive. The most significant feature, which sets mining projects apart from other commercial activities, lies in the nature of the main asset, the mineral deposit. This asset is imperfectly defined, it is not possible to move, it is depleted and exhausted in several years, and it cannot be replaced.

The predominant economic evaluation technique for a mineral project is the discounted cash flow method, using net present value (NPV), internal rate of return (IRR), and payback period (PP) calculations. This methodology is easy to understand and accepted by the industry and the financial community. For this reason, all the mining project evaluation processes for investment decision worldwide are based on these indexes. Because many of the items included in the calculation of NPV, IRR, and PP are almost impossible to be predicted, the process must be adjusted to risk. From a financial viewpoint, the risk is the possibility that shareholders will lose money where they invest in a company. There are quite a number of methods to evaluate the risk, being the Monte Carlo method probably the most used and well known, especially since the introduction of the computing equipment. The Monte Carlo method is based on the simulation of the various sources of uncertainty affecting the studied value and subsequent determination of the average value over the range of resultant outcomes.

1.4.3 Exploitation

If the economic evaluation of a mining project offers positive results, pointing to high probability that the exploitation of the mineral deposit will produce benefits, mining will be the next step. Exploitation or mining is the process of excavation and recovery of ore and associated waste rock from Earth's crust. Mine method selection criteria is based on rock competency, distance to surface, characteristics of the mineral, and economics. Conditioned by the distance to surface, the mining methods are broadly grouped into surface (◘ Fig. 1.13) and underground (◘ Fig. 1.14). About 85% of the global tonnage is produced in open-pit mines, including placer operations, while the rest 15% from underground mines (Ericsson 2012). Operating mines range from small size underground operations to large open pit, some of them moving tens of thousands of rock per day.

Surface mining is a form of operation led to extract minerals lying near the surface. In the last decades, surface production spreads out since open-pit mining is less expensive than underground

◘ **Fig. 1.13** Surface mining at Los Filos (Mexico) (Image courtesy of Goldcorp Inc.)

Fig. 1.14 Underground mining at Mponeng (South Africa) (Image courtesy of AngloGold Ashanti)

mining, due to the higher cost of underground extraction methods. The depletion of richer mineral bodies combined with the development of new technologies makes necessary to work in open-pit mines with lower mineral contents. The present-day tendency is large-scale surface mining using the economy of scale, that is, the saving in cost of production that is due to mass production. In mining language, a big mine will produce significantly more output per unit of input than will a small mine. Thus, large-scale equipment are used to make operations efficient and economical. For example, today trucks are huge, carrying up to 500 tons per load.

If the depth of an ore deposit is such that removal of overburden makes surface mining unprofitable, underground methods must be considered. Underground mining refers to extraction of raw materials from below the surface of the ground. One logical procedure to categorize underground mining methods is to divide them into the following three groups: (a) methods producing openings naturally supporting or requiring minimum artificial support (e.g., room and pillar); (b) methods requiring substantial artificial support (e.g., cut and fill), and (c) caving methods where collapse of the rock is integral to the extraction process (e.g., block caving). For instance, in the room-and-pillar method, very common in underground mining, the minerals are obtained from large voids (rooms) and pillars are left between the rooms to support the overlying rocks. In general, the mineral body included in the pillars remains upon completion of mining and is not recovered. If the mineral body extends from surface to great depth, mining sometimes starts near the surface from an open pit and later continues the exploitation with underground mining for the deeper parts of the mineral body; this method is usually called combined mining.

The use of explosives is often indispensable in mineral resources exploitation. Therefore, blasting is usually a part of the mining cycle. Blasting is the process of fracturing material by the utilization of an amount of explosive loaded in special holes. There are many different types of explosives used today such as ANFO (ammonium nitrate plus fuel oil), slurries, and emulsions. The holes drilled for blasting are loaded so that each one is fired in a designed sequence to obtain the desired break of the rocks. The explosives are then detonated in the drill holes (Fig. 1.15).

1.4.4 Mineral Processing

After exploitation, mineral processing separates useful minerals from waste rock or gangue, producing a more concentrated material for further processing. The concentration of valuable minerals in the run-on-mine material is also known as beneficiation or concentration process. The objective is to reduce the bulk of the material using cheap and low-energy physical methods to sort out the valuable minerals from the waste rock. In general, the heavier the material, the more costly

Fig. 1.15 Blasting in an open-pit mine (Image courtesy of Anglo American plc)

Fig. 1.16 Particle size reduction using mills (Image courtesy of North American Palladium Ltd.)

it is to transport. Therefore, metal ores become much lighter once upgraded to concentrates or processed into semi-finished products, making them more economical to transport long distances. Thus, metal ores are commonly processed, at least partly, close to their extraction site.

The material obtained in the mine is concentrated using particle size reduction (Fig. 1.16), liberation, and concentration with mainly physical methods. To begin, the rock is crushed, grinded, and classified utilizing a very broad variety of equipment. The primary crushing can be carried out during the mining, especially where an underground method is selected. Regarding concentration or beneficiation methods, there are many types since metal content and physicochemical properties of minerals are quite different. It is necessary to produce concentrates of every category with maximum efficiency. The three main groups of concentration systems methods are magnetic, gravity, and froth flotation methods, the last one being the most used to concentrate metallic minerals. Magnetic methods use the difference in magnetic properties of the mineral particles. They are implemented in four different ways, being the devices distinguished firstly on the basis of dry or wet material, and, secondly, on the basis of magnetic field intensity (high or low).

Gravity concentration separates grains of minerals depending on their density. The separation process is also determined by the size of the particles. Minerals with value can be removed along with the material despite differences in density if the particle sizes change. For this reason, particle sizes must be uniform and the use of as screens and hydrocyclones is essential. However, most of profitably minerals (e.g., sulfides of Cu, Zn, or Pb, PGE minerals, and many others) are best suited to froth flotation method. This concentration method is a technique where particles, in a mineral/water slurry, are adhered to air bubbles using chemical reagents, which preferentially react with the desired mineral. Then, the particles are carried to the surface and removed. In general, it is very useful for processing fine-grained ores and can be applied to many types of mineral separations (e.g., separating sulfide minerals from silica gangue, removing coal from ash-forming minerals, or separating different industrial minerals, among others). The nonvalue minerals obtained in froth flotation or any concentration process are disposed to tailing pond or void filling stabilization of underground mines.

Since water is usually involved in the concentration process, the last stage in mineral processing is to remove water in the slurry. This process is called dewatering and commonly starts with thickening, utilized if the liquid-to-solids ratio is high. The mechanism employed is based on sedimentation where the solids are allowed to settle through the liquid phase, resulting in a liquid essentially solid free and a thickened slurry. Afterwards, vacuum or pressure filtration is applied to remove water from the slurry using a porous filter medium, which prevents the passage of the solid particles. The product obtained, usually named cake, can be already sent to metallurgy process. Two different types of equipment are commonly used in vacuum filtration: drum and disk filters. On the other hand, pressure filtration is carried out with plate filter.

1.4.5 Closure and Reclamation

Mine closure is the last phase in the mining cycle since mining is a temporary activity, with the operating life ranging from some years to several decades. Closure starts when the mineral resource is exhausted or operations are no longer profitable. Mine closure plans are required by most regulatory agencies worldwide before a mining permit is granted. Financial assurance is required in many countries as a guarantee that the funds needed for mine closure will be available if the responsible company is unable to complete the process as it was planned. Reclamation, which occurs at all stages of the mine life (environmental analysis begins at the earliest stages of the exploration of the mineral resource), involves earthwork and site restoration including revegetation of disposal areas (Fig. 1.17). The aim of reclamation must always be to return the site to a condition that match the premining condition. Other possibilities include to

Fig. 1.17 Revegetation in mining reclamation (Image courtesy Newmont Mining Corporation)

use the mine sites recreational areas, gardens, parks, etc. Previously, an environmental impact assessment (EIA) is requested and presented. An EIA can easily be defined as a study of the effects of a proposed mining project on the environment. The environmental impact assessment process is an interdisciplinary and multistep procedure to ensure that environmental aspects are included in decisions regarding mining projects.

Potential environmental impacts linked to mining activities include impacts such as hazardous materials, land use, biodiversity, visual impacts, and air and water quality, among others. In general, underground mines are much less apparent than surface mines, and they disturb a relatively small area of the land surface close to the principal shaft. Where underground mining activities finish, the shafts can be sealed and the area returns to previous condition, especially in which respects to visual impact. In relation to the impact of waste disposal (tailings dam), which usually incorporate small amounts of harmful elements, it can contaminate surface and groundwaters. Occasionally, tailings dam failures cause huge environmental disasters, as occurred in Alnazcóllar (Spain) in 1998, where Los Frailes Mine tailings dam failed and released five million cubic meters of acidic tailings. The fine-grained material contained dangerous levels of several heavy metals that travelled about 40 km before it stopped, just near a UNESCO World Heritage Site (Doñana National Park), which is one of the largest National Parks in Europe.

1.5 International Reporting Standards

Although the definition and classification of Mineral Resources and Reserves shown in ▶ Sect. 1.2 are widely accepted and used in many countries, the main guidelines to apply these terms are undoubtedly the International Reporting Standards, also known as Mineral Reporting Codes. They are essential nowadays to any project evaluation process of mineral resources, especially if the financial world is involved. It is important to note that mineral resources and reserves are the most important economic asset for a mining company since the financial strength of the enterprise depends mainly on the size and quality of its resources and reserves. However, reported mineral reserve data for mining projects include numerous types of uncertainty such as geological estimation and difficulties to predict the future commodity price (Dimitrakopoulos and Abdel Sabour 2007). Since a reserve is only a small portion of the total ore body, variations in price obviously alter reserve estimations. For this reason, regulatory guidelines for reserve estimation must take in consideration the potential economic variability in the complete lifetime of the mining project (Evatt et al. 2012).

Mining is historically important in many regions around the world, being the major mining centers countries such as Australia, South Africa, the USA, and Canada. Thus, these countries, together with Europe (especially the United Kingdom), are the main sources of capital for mining projects. For this reason, these countries have promoted the reporting standards most used worldwide. After its initiation in the USA, Australia got ahead in providing codes and guidelines for reporting and classifying mineral resources and reserves; omitting the McKelvey Box, really the precursor to mining codes, the first International Reporting Code was the JORC Code (Australia). It was published in 1989 and later updated. Afterwards, a rapid increase in the creation of these codes and standards in other countries was produced. This process of creating new standards began in the late 1990s and continues today.

These standards/codes derived from the globalization of the mining industry and their objectives are to give a certain level of comfort to investors and other stakeholders regarding quality and usefulness of valuation of mineral deposits (see Bre-X Affair in ◘ Box 1.4). The increasing investment by foreign countries in developing countries of Asia, Africa, and South America, among other reasons, needs to an international method to define the assets of mining companies and also the mineral wealth of the countries. Major mining companies commonly finance new projects internally and develop many times their own systems of control. Although these are usually similar to common codes and standards, a relevant exception is that the responsibility for determining and certifying ore reserves lies on a qualified team of professionals, rather than an individual expert. Broadly, International Reporting Codes can be classified into two groups: (a) International Systems (CRIRSCO and UNFC) and (b) National Codes (e.g., 43-101, JORC, or SMA). There are differences between them, but certainly many terms and definitions are similar in all documents.

Box 1.4

The Bre-X Affair

Bre-X Minerals Ltd., a member of the Bre-X group of companies, was involved in one of the major scandals of the entire history of mining. David Walsh founded Bre-X Minerals Ltd., a small Canadian exploration firm, in 1989 as a subsidiary of Bresea Resources Ltd. Initially, the focus of the company was to explore looking for diamonds in the Northwest Territories. The company did not make a significant profit before 1993, when Walsh followed the advice of geologist prospector John Felderhof and bought a property in March 1993 in the middle of a jungle near Busang River in Borneo, Indonesia, for US$ 80,000. In August 1993, Bre-X began to explore in Kalimantan (Borneo), and it soon reported significant drilling results at Busang. Assays of the drill samples indicated consistent gold mineralization, extending from the surface to a depth of hundreds of meters, and estimates of the size of the resource steadily grew. The evaluation of geologist Michael de Guzman, Project Manager, was impressive: the first estimate was 17 million ounces of gold, which would have made it the richest gold deposit ever. Thus, the company reported in October 1995 an enormous gold deposit located in Indonesia. The company moves to the Toronto Stock Exchange in April 1996 and splits its shares ten to one in May. The new shares trade at $28.65 (Canadian dollars).

Therefore, the stock's price of the company grew logarithmically in the Toronto Stock Exchange. From initial private offerings at 30 cents a share, Bre-X stock climbed to $286.50 (Canadian dollars) per share by 1997 (split adjusted). In its peak, it had a total market capitalization equal to approximately US$5 billion (Canadian dollars) on the open market. Near the maximum Bre-X share price, major banks and media were buying. In December 1996, Lehman Brothers Inc. strongly recommended a buy on «the gold discovery of the century». Obviously, major mining companies and top producers fought a battle to get a piece of Bre-X's Busang deposit. This was because the gold amount in the deposit was changing over the years from 30 million ounces (900 tons) in 1995 to 200 million ounces (6200 tons) in 1997 (or up to 8% of the entire world's gold). As the estimated size of the deposit grew, so did the stock price and the hysteria.

At that moment, the Indonesian government of President Suharto also got involved. It claimed that Bre-X was not playing by the «rules» of the country, and Bre-X's exploration permits were revoked. Stating that a small company like Bre-X could not exploit the site by itself, the Indonesian government suggested that Bre-X share the site with the large Canadian mining firm Barrick Gold in association with Suharto's daughter Siti Rukmana. Finally, a joint venture is reached in 1997 that gives Indonesia 40% share, Bre-X 45%, and Freeport-McMoRan Copper and Gold Inc. at 15% share of interests. The fraud began to unravel on March 19, 1997, when Bre-X geologist Michael de Guzman jumped to his death on a suicide (or was pushed) from a helicopter in Indonesia. Meanwhile (January 1997), a mysterious fire destroys at Busang the administration office and geology records. On March 26, 1997, Freeport-McMoRan announced that its own due-diligence core samples showed «insignificant amounts of gold». As a result, Suharto postponed signing the mining deal. A third-party independent company, Strathcona Mineral Services Ltd., was brought in to make its own analysis. They published their results on May 4, 1997: the Busang ore samples had been salted with gold dust. A year later, David Walsh dies of a brain aneurysm at his home in the Bahamas.

Consequently, Bre-X Minerals collapsed in 1997 after the gold samples were demonstrated to be a fraud and its shares became worthless, in one of the biggest stock scandals in Canadian history. Trading in Bre-X was soon suspended on the TSX and the NASDAQ, and the company filed for bankruptcy protection. Among the major losers were several Canadian public sector organizations such as The Ontario Municipal Employees Retirement Board, the Quebec Public Sector Pension fund, and the Ontario Teachers' Pension Plan. The fundamental problem was the lack of control in the gold assays data because the results were clearly manipulated and falsified using a salt process; salt process in mineral exploration means to add metal (typically gold or silver) to an ore sample to change the value of the ore with an aim to deceive. Bre-X salt process can be considered as the most elaborated fraud in the history of mining and one of the biggest stock scandals in Canadian history. This scandal accelerated the development and publication of standards worldwide and particularly the publication of NI 43-101 in Canada to protect investors.

1.5.1 CRIRSCO International Reporting Template

CRIRSCO Template is targeted to the international minerals industry, including mining and exploration companies, professional institutions, regulators, financial institutions, and others. One important aspect is that this document only relates to the reporting of economic and potentially economic mineralization as defined in the reporting standards. Thus, it does not consider the reporting of subeconomic or yet undiscovered mineralization. As the scope of the document explains, the main principles governing the operation and application of the Template are transparency, materiality, and competence. Transparency means that sufficient information is provided to the reader of a Public Report; materiality indicates that a Public Report contains all the significant information that investors would rationally require; and competence means that the Public Report is based on work carried out by appropriate qualified persons. Thus, the author of the Public Report is a Competent Person or Persons. A Competent Person is defined in the Template «as a minerals industry professional with a minimum of 5 years relevant experience in the style of mineralization or type of deposit under consideration and in the activity that the person is undertaking».

The Template, as well as all Standards, is very long to summarize here, but the most important terms must be incorporated, especially those devoted to concepts of Exploration Results, Mineral Resources, or Mineral Reserves. Regarding Exploration Results, an Exploration Target is «a statement or estimate of the exploration potential of a mineral deposit in a defined geological setting where the statement or estimate, quoted as a range of tons and a range of grade or quality, relates to mineralization for which there has been insufficient exploration to estimate mineral resources. Exploration Results include data and information generated by mineral exploration programs that might be of use to investors but do not form part of a declaration of Mineral Resources or Mineral Reserves».

The second main topic of the Template is the concept and the different categories of Mineral Resources. A Mineral Resource is «a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling». There are three categories of Mineral Resources: Inferred, Indicated, and Measured. Their definitions are somewhat different from those shown in the McKelvey Box. An Inferred Mineral Resource is «that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity». An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource. Since confidence in the estimate is usually not sufficient to allow the results of the application of technical and economic parameters to be used for detailed planning, there is no direct link from an Inferred Resource to any category of Mineral Reserves.

In turn, an Indicated Mineral Resource in the Template is «that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation». Concerning Modifying Factors, they are «considerations used to convert Mineral Resources to Mineral Reserves; these include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors» (Fig. 1.18).

Obviously, an Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and can only be translated to a Probable Mineral Reserve, but has a higher level of confidence than that applying to an Inferred Mineral Resource. Finally, a Measured Mineral Resource in the Template is «that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit; geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation». Of course, a Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource and it may be converted to a Proved Mineral Reserve or to a Probable Mineral Reserve. According to the Template, this category «requires a high level of confidence in, and understanding of, the geology and the controls of the mineral deposit; the choice of the appropriate category of Mineral Resource depends upon the quantity, distribution and quality of data available and the level of confi-

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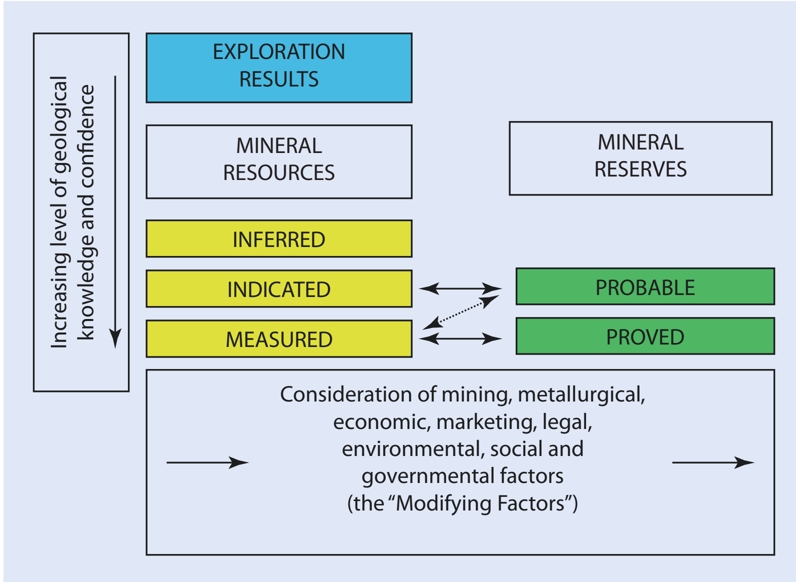

Fig. 1.18 Modifying factors and the conversion of mineral resources to mineral reserves

dence that attaches to those data». A Competent Person or Persons, as defined above, must determine the appropriate Mineral Resource category.

The third topic of the Template includes the concept and categories of Mineral Reserves. Thus, a Mineral Reserve is «the economically mineable part of a Measured and/or Indicated Mineral Resource; the term economically mineable implies that extraction of the Mineral Reserve has been demonstrated to be viable under reasonable financial assumptions». Therefore, Mineral Reserves «are those portions of Mineral Resources that, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Competent Person making the estimates, can be the basis of a viable project, after taking account of all relevant Modifying Factors».

Mineral Reserves are classified in the Template as Probable and Proved. A Probable Mineral Reserve is «the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource; the confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proved Mineral Reserve». Therefore, a Probable Mineral Reserve has a lower level of confidence than a Proved Mineral Reserve, but it is of sufficient quality to serve as a basis for decision on the development of the deposit. Related to Proved Mineral Reserve, «it implies a high degree of confidence in the Modifying Factors and represents the highest confidence category of reserve estimate».

Mineral Resources can be estimated mainly based on geological information. However, Mineral Reserves need consideration of the Modifying Factors affecting extraction. Thus, Mineral reserves should be estimated mainly with input from a different discipline. It is also very important the type and characteristics of the technical studies. Scoping, prefeasibility, and feasibility studies are carried out to establish the overall characteristics of the mining project in each stage of its development. The descriptions of each type of study are detailed in ► Sect. 4.5.1.

1.5.2 United Nations Framework Classification

This standard is entitled «United Nations Framework Classification for Fossil Energy and Mineral Reserves and Resources». The last update was made in 2009 (UNFC-2009) (Fig. 1.19). The document is a complex combination of categories, subcategories, classes, and subclasses, and it is still not completely accepted in the mining world. UNFC-2009 is a generic principle-based system in which quantities are classified based on three fundamental criteria: economic and social viability (E), field project status and feasibility (F), and geological knowledge (G). Combinations of these criteria and using a numerical coding system, a three-dimensional system is created. Categories (e.g., E1, E2, E3) and, in some cases, subcategories

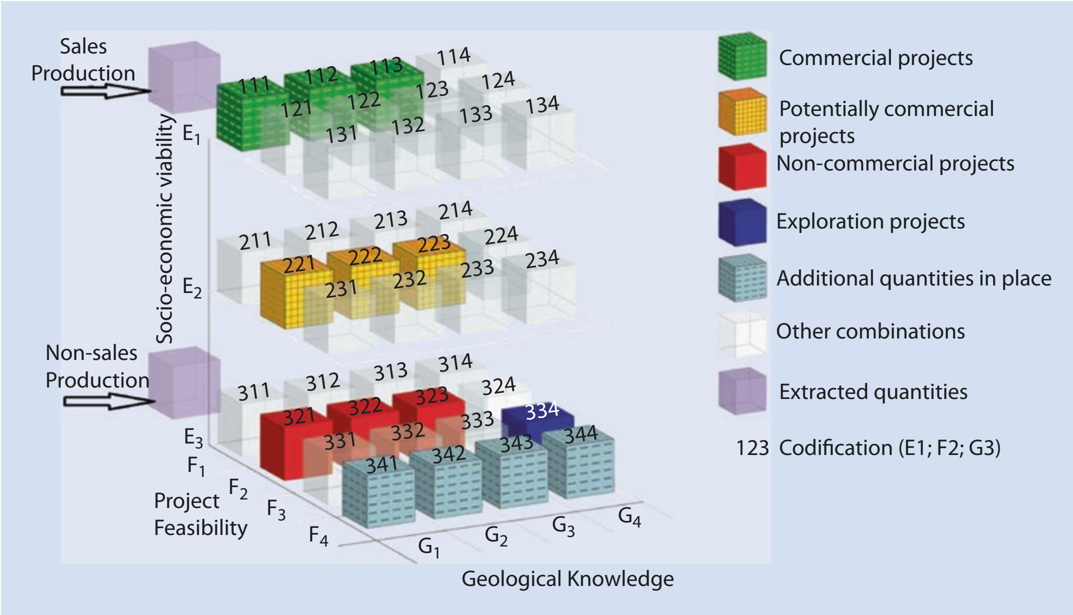

Fig. 1.19 UNFC-2009 categories and examples of classes

(e.g., E1.1) are defined for each of the three criteria. According to this classification, the first set of categories (the E axis) indicates «the degree of favorability of social and economic conditions in establishing the commercial viability of the project, including consideration of market prices and relevant legal, regulatory, environmental and contractual conditions». The second set (the F axis) «points to the maturity of studies and commitments necessary to implement mining plans or development projects». The third set of categories (the *G* axis) «designates the level of confidence in the geological knowledge and potential recoverability of the quantities».

This classification system is quite complicated regarding definitions of the different concepts and types of Mineral Resources and Reserves, always related to letters and numbers. Moreover, some terms are misleading on purpose, since the Standard states that «reserves is a concept with different meanings and usage; even the extractive industries, where the term is carefully defined and applied, there are some material differences between the specific definitions that are used in different sectors. It indicates that it is not ideal as a basis for global communication of such an important quantity». Obviously, this remark is probably true, but the solution is not to create an intricate network of letters and numbers to explain concepts already delimited in other classical and accepted classifications as the CRIRSCO mentioned above or the following 43-101 from Canada.

1.5.3 National Codes

A National Code is a standard whose utilization is obligatory in the proper country that promotes its development. There are many national codes, but only some of them are accepted worldwide. This is the case of NI 43-101, created in Canada but considered essential in many countries. Because many mines around the world are property of companies which report their results on stock exchanges within Canada, NI 43-101 is thoroughly used in the mining world. Other National or Regional Codes more or less similar to NI 43-101 include the JORC Code (the first code in time, as stated before) and the VALMIN Code in Australia, the SME Guide in the USA, the SAMVAL Code and SAMREC Code in South Africa, the PERC Standard in Europe, the NAEN Code in Russia, the MRC Code in Mongolia, and the CCEP in Chile.

The development of NI 43-101 in Canada is partly related to the Bre-X Affair, which is considered the most elaborated fraud in the history of mining (Box 1.4: The Bre-X Affair). In broad terms, NI 43-101 is very similar to CRIRSCO Template and the rest of National Codes. It is clear that most of the national codes are derived or

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extracted from the JORC Code, since it was established earlier and has been relatively reliable as compared to the rest of codes. The Standard includes definitions of Qualified Person (the author of the Public Report), Mineral Resources, Reserves, different categories, etc., in a same way to that of CRIRSCO or JORC Code. In general, public reporting of these topics in the diverse National Codes benefits from considerable international conformity, due to the efforts of national reserves committees and CRIRSCO. Interestingly, the person responsible for the preparation of the report in the Certification Code of Chile is called «Qualified Competent Person», which is a mixture of «Qualified Person» (NI 43-101) and «Competent Person» (JORC Code).

NI 43-101 includes a long table of items in the content of the technical report (Table 1.1). NI 43-101 and JORC Code technical reports are commonly regarded interchangeable, since their contents and scientific rigors are often very similar. As a result, both codes are accepted as industry reporting standards by numerous professional institutions. The difference lies in how the different classes of resources are used in economic studies, especially the inferred resources, since the required levels of confidence can be change for every category. For instance, inferred resources can be used with some cautions in economic studies in the SME Guide (USA) and the SAMREC (South Africa), but not in the NI 43-101 (Canada), because of the low confidence and insufficient data. It clearly states that inferred mineral resources may be only utilized for internal planning.

Table 1.1 Items in the Ni 43-101 technical report

Item 1	Title Page
Item 2	Table of Contents
Item 3	Summary
Item 4	Introduction
Item 5	Reliance on Other Experts
Item 6	Property Description and Location
Item 7	Accessibility, Climate, Local Resources, Infrastructure, and Physiography
Item 8	History
Item 9	Geological Setting
Item 10	Deposit Types
Item 11	Mineralization
Item 12	Exploration
Item 13	Drilling
Item 14	Sampling Method and Approach
Item 15	Sample Preparation, Analyses, and Security
Item 16	Data Verification
Item 17	Adjacent Properties
Item 18	Mineral Processing and Metallurgical Testing
Item 19	Mineral Resource and Mineral Reserve Estimates
Item 20	Other Relevant Data and Information
Item 21	Interpretation and Conclusions
Item 22	Recommendations
Item 23	References
Item 24	Date and Signature Page
Item 25	Additional Requirements for Technical Reports on Development Properties and Production Properties
Item 26	Illustrations

1.6 Distribution of Mineral Resources in the Earth

Mineral resources are present in the Earth's crust, a thin outer shell about 10–100 km thick, which comprises no more than 0.4% of the Earth's mass, being assigned the rest of the mass to the core and mantle. The oceans are underlain by a thin (10 km) homogeneous crust that covers approximately 70% of the surface, and the continents have a much thicker crust (30–100 km) covering the remaining 30%. The continental crust is quite more inhomogeneous, since magmatic, sedimentary, and metamorphic processes have led to segregation and local concentration of elements. It represents the main focus of exploration and exploitation of mineral resources. The average concentration of elements in the Earth's crust controls the occurrence of mineral deposits. Nine elements, called major elements, make up over 99.5% of the continental crust, while the rest of

Table 1.2 Element abundance in the Earth's crust

Element	Crustal abundance (%)
Oxygen	46
Silicon	27
Aluminum	8.1
Iron	6.3
Calcium	5
Magnesium	2.9
Sodium	2.3
Potassium	1.5
Titanium	0.66
Carbon	0.18
Hydrogen	0.15
Manganese	0.11
Phosphorus	0.099
Fluorine	0.054
Sulfur	0.042
Strontium	0.036
Barium	0.034
Vanadium	0.019
Chlorine	0.017
Chromium	0.014
Zirconium	0.013

the elements, called minor and trace elements, account for less than 0.5% (Table 1.2).

Major elements are abundant enough to form the most usual minerals and rocks, but minor and trace elements, including most of the metals, need to be enriched under exceptional geological conditions to form exploitable mineral deposits, usually combined with other elements like oxygen (oxide), sulfur (sulfide and sulfate), and carbon (carbonate). Thus, although any piece of rock in the Earth's crust contains small amounts of metallic and nonmetallic elements, a specific natural process is necessary to produce sufficient enrichment of the element to form an exploitable mineral deposit (Fig. 1.20). An additional point is that processes forming mineral deposits operate

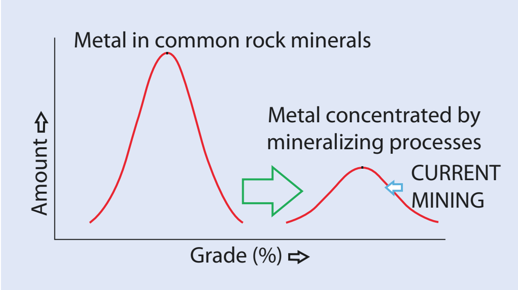

Fig. 1.20 Enrichment of the metals to produce an exploitable mineral deposit

at geological time scales, so that most economic mineral resources are basically nonrenewable. In other words, new deposits are impossible to be generated in human timescales.

The distribution of mineral resources in the Earth's crust is irregular, from not only a commodities point of view but also considering the geographical position of the mineral deposits across the different continents; the major reserves of metal ores are geographically concentrated in a handful of countries. Even the distribution of the valuable mineral in each deposit is varied according to the grade and the tonnage of the mineralization. Geographically, the global distribution of mineral resources depends on the type of mineral or metal. Thus, gold deposits are present in more than 100 countries, but the largest reserves are concentrated in South Africa (27–28% of world's reserves). On the contrary, platinum metal group deposits are known only in 16 countries, and the share of two largest countries (South Africa and Russia) covers 97% of world's reserves. In general, the global distribution of mineral resources is very conservative and it was not changed over a long time. For instance, today world's reserves of tin are distributed in the same countries and regions as they were 30 years ago, although the contribution of each country or region has changed. These variations of the contribution are related to the world economic development and the situation of the raw material markets.

For every substance, it is possible to calculate a value called concentration or enrichment factor, dividing the economical concentration, that is, the necessary concentration in a mineral deposit for profitable mining, by the average crustal abundance for that substance. Concentration factors and average crustal abundances for some of the most important metals are listed in Table 1.3

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Table 1.3 Average crustal abundances and concentration factors for some of the most important metals

Element	Chemical symbol	Average concentration %	Quantity/km^3 (000 mt)	Typical ore grades %	Enrichment factor
Aluminum	Al	8.1	250,000	30	4
Iron	Fe	5.4	150,000	53	10
Titanium	Ti	0.5	15,000	0.7–15	2–40
Manganese	Mn	0.10	3000	31	310
Chromium	Cr	0.01	300	30	3000
Nickel	Ni	0.008	200	1	130
Zinc	Zn	0.007	190	4	570
Copper	Cu	0.005	135	0.5–4	100–800
Cobalt	Co	0.002	60	0.4–2	200–1000
Lead	Pb	0.001	35	5	3850
Uranium	U	0.0003	7	0.3	1100
Tin	Sn	0.0003	7	0.3	1200
Molybdenum	Mo	0.0002	4	0.2	1300
Tungsten	W	0.0001	3	0.7	5800
Silver	Ag	0.00001	0.2	0.01	1400
Gold	Au	0.0000003	0.01	0.001–0.0001	300–3000

(Gocht et al. 1988). For example, aluminum (average crustal abundance of about 8%) has a concentration factor of 3–4 because a mineral deposit of aluminum (e.g., gibbsite mineral) can contain between three and four times the average crustal abundance to be economical (between 24% and 32% aluminum). Obviously, this enrichment or concentration factor is very different for each element, ranging from low values (3–4 for aluminum) to very high values (4000–5000 for gold).

It is essential to estimate the amount of a given mineral resource in the world from its abundance in the Earth's crust. The reason is that strategic planning for future supply of a mineral or a metal is controlled by the estimates of prognostic resources, although the predictions of undiscovered resources quantifications are obviously very difficult. The methods proposed are usually based on extrapolation of resources in well-known regions to less known, but geologically similar, parts of the Earth (e.g., Singer and Menzie 2010). However, the topic of mineral resource assessment is quite complex. Assessment methods considered were time rate, crustal abundance, cumulative tonnage versus grade, geometric probability, and discriminant analysis, among many others. The selection of the method to be employed in an assessment must be based on different factors such as adequacy of the material to the problem, constraints in resources (e.g., information or time forthcoming for the assessing), the level of uncertainty and acceptance of errors in the evaluation, and finally the requirement for checking outcomes and approval of the technique (Singer and Mosier 1981).

McKelvey (1960) was one of the first authors to analyze the distribution of mineral resources in the Earth. He pointed out that «the tonnage of mineable reserves in short tons (R) for many elements in the United States was equal to crustal abundance in percent (A) times 10^9 to 10^{10}, and that the linear relation that appears to prevail between reserves and abundance is useful in forecasting reserves in large segments of the Earth's crust or over the world at large». Even for purposes of estimating world reserves of unexplored elements, McKelvey affirmed that «a figure of $A \times 10^{10}$ to 10^{11} probably will give the right order of magnitude». In other words, reserves for some elements exhibit a constant ratio to their average crustal abundance and for less explored commodities the reserves can be estimated from well-explored ones. With regard to the question as to whether or not the USA is a representative sample of the Earth's crust, it does have all the major kinds of geological terranes found anywhere and may be accepted as a reasonably representative sample of the Earth's crust.

The total amount of different metals in the Earth's crust can be calculated combining crustal abundance data and the McKelvey reserve-abundance relationship. Thus, the potential recoverable resource in metric tons for most elements should approach $2.45A \times 10^6$, where A is abundance expressed in parts per million (Erickson 1973). If the abundance-reserve relationship is accepted, the amount expressed is a minimum total resource estimate because the relationship is based upon currently recoverable resources and does not include resources whose feasibility of economic recovery is not established. The abundance-reserve relation should become more closely defined as analytical techniques progress, as the understanding of geochemical processes enhances, and as exploration techniques advance and it was possible to explore and examine the crust until a reasonable depth.

Estimating contemporary reserves is very useful, but it is only a starting point because the focus of the question is not on the short-term, but on long-sighted availability (e.g., Graedel et al. 2011). These authors tried to estimate the extractable global resource (EGR), that is, «the quantity of a given resource that is judged to be worthy of extraction over the long term given anticipated improvements in exploration and technology», for most metals, considering that information available on the potentially extractable geological resources of metals is negligible. The main conclusion of this study was that it is not possible at the moment reliably to estimate the extractable global resource (EGR) for any metal.

The aforementioned aspects deal with the distribution of mineral resources in big areas or regions, even in the Earth as a whole. However, a crucial point is to know how the resources/reserves (grades and tonnage) are distributed in a mineral deposit with a gradation from relatively rich to relatively poor mineralization, that is to say, the relation between grades and tonnages in an ore deposit. Lasky (1950) was the pioneer in applying mathematical laws to predict reserves of ore deposits and to study how the recoverable reserves of porphyry copper deposits (a copper deposit type, see ▶ Chap. 2) varied as a function of the usual selection criterion, the grade in Cu percentage (◘ Box 1.5: Lasky's Law).

The relationship between ore grade and tonnage in a mineral deposit can be analyzed in terms of fractals (Turcotte 1986) because grade relations and tonnage for economic ore deposits show a fractal behavior if the tonnage of ore with a specific mean grade is proportional to this mean grade raised to a power. If it is assumed that the concentration of elements in ores is statistically scale invariant, the renormalization group approach can be used to derive a fractal relationship between mean grade and tonnage. Moreover, the obtained results are independent of the mechanism of mineral concentration as long as the concentration mechanism is scale invariant. This approach would not be expected to be valid if different concentration mechanisms are operative at different scales. Thus, in terms of fractals, the relationship between ore grade and tonnage in a mineral deposit can be defined by using the following equation (Turcotte 1997):

$$C_{\text{ore}} / C_{\min} = \left(M_{\min} / M_{\text{ore}}\right)^{D/3}$$

where C_{ore} is the average grade of the tonnage M_{ore}, $C_{\min}$ is the minimal grade included of the mass $M_{\min}$, and D is the fractal dimension. $M_{\min}$ may be the mass of ore exploited at the lowest grade mine or even source rock from which the ore in a district is thought to be derived. Based on this correlation, undiscovered resources can be estimated. Tonnage versus cut-off and average grade versus cut-off models can be outlined according to the fractal distribution of element concentrations, considering that the cut-off grade has great influence on the reserve and resource calculation in a single deposit (Wang et al. 2010).

Another possibility is to combine the fractal modeling and geostatistics for mineral resource classification to look for a clear separation, identification, and assessment of high-grade ore zones from low-grade ones in a deposit, which are extremely important in mining of metalliferous deposits (e.g., Sadeghi et al. 2014). Compared to existing methods of mineral resource classification, the technique that combines geostatistics and fractal modeling can address the complexity of the data for different parts of a mineral deposit.

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Box 1.5

Lasky's Law

Samuel G. Lasky, Chief Mineral Resources Section of the U.S. Geological Survey, was a pioneer in applying mathematical laws to predict reserves of ore deposits and to study how the recoverable reserves of porphyry copper deposits (a copper deposit type) varied as a function of the usual selection criterion, the grade in Cu percentage. For the average U.S. porphyry copper deposit, he found that a decrease in grade of 0.1% Cu was associated with an increase in tonnage of about 18%. Thus, the famous Lasky's equation (Lasky 1950), also known as Lasky's law (Matheron 1959) or the arithmetic-geometric (*A*/*G*) ratio was derived from production records of several porphyry copper deposits. The equation states that if ore grades are distributed log-normally, the increase of reserves is exponential. Therefore, the tonnage of ore that has been produced plus the estimated reserves (*T*) and the weighted average grade of this tonnage (*G*) distribution follow the equation:

$$G = K_1 - K_2 \times \ln(T)$$

where K_1 and K_2 are constants to be determined for each ore deposit. They usually have to be determined empirically, using historical data. K_2 is always preceded by a minus sign to indicate the inverse relationship between tonnage and grade. A typical curve for such a relationship will be plotted as a straight downsloping line on a semilogarithmic paper, with the grade on the logarithmic horizontal axis and tons of ore on the arithmetically spaced vertical axis. The cumulative contained copper curves become flat at a point that Lasky called the zero cut-off grade, at which the copper grade tends to approach average abundance of copper in the Earth's crust.

The most common mistake made in quoting Lasky's law is to assume that it states that the quantity of metal increases geometrically as the ore grade declines arithmetically. In fact, Lasky's results show a definite limit to the cumulative quantity of metal in a deposit. The aim in reporting the relationship was to enable mining engineers to forecast the recoverable reserves from a given deposit. Therefore, his results say nothing about the distribution of different deposits, only about the distribution of ore within a given deposit (Chapman and Roberts 1983).

Lasky's relationship is consistent with studies of the correlation between average grades and cumulative ore tonnage of mineralized material in the resource analysis of groups of deposits as well as to individual deposits. However, it is also possible that a linear relation is obtained if the logarithm of the tonnage is plotted against the logarithm of the grade (Cargill et al. 1981). One of the issues of this relationship is that the projection of Lasky's analysis to lower grades is limited because the mathematical formulation predicts physically impossible situations below some limiting grade: as average grade (*G*) approaches zero, the derivative of tonnage (*T*) with respect to average grade produces «astronomical» increases in tonnage. As a rule, Lasky's law is only true for specific deposits, but it cannot be used as a general tool because Lasky's technique had geological limitations in addition to the empirical limitations of the equation at high or low grade. Lasky's relationship should be found notoriously ineffective for the majority of mercury, gold, silver, tungsten, lead, zinc, antimony, beryllium, tantalum, niobium, and rare earth deposits. Singer (2013) affirmed that significant deviation from lognormal distributions of most metals when ignoring deposit types demonstrates that there is not a global lognormal or power law equation for these metals.

1.7 Mineral Resources Consumption

Natural resources provide essential inputs to production. World population is growing faster than at any time in history. Meanwhile, mineral consumption is increasing more quickly than population as new consumers enter the market for minerals and as global standard of living rises. According to Glöser et al. (2015), the rapid economic development of emerging countries in combination with an accelerating spread of new technologies has led to a strongly increasing demand for industrial metals and minerals regarding both the total material requirement and the diversity of elements used for the production of specific high-tech applications. Therefore, minerals and metals are consumed in greater quantities than ever before. Since 1900, the mine production of the main metals has increased by several orders of magnitude (Graedel and Erdmann 2012) (Fig. 1.21). Increased world demand for minerals will be affected by three factors (Kesler 2007): applications for mineral commodities, the level of population that consumes these raw materials, and the standard of living that will establish how much each person consumes. As new materials and applications are found, markets for mineral commodities can expand considerably. In this sense, present technologies utilize almost the entire periodic table.

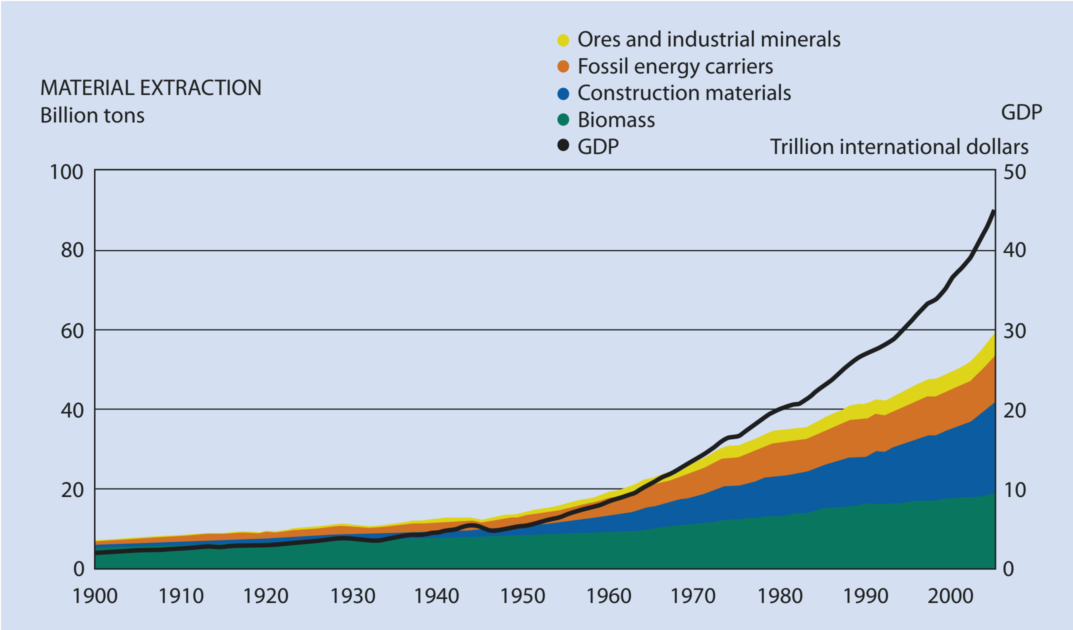

Fig. 1.21 Global material extraction in billion tons from 1900 to 2005 (Modified from Krausmann et al. 2009)

Demand forecasting is difficult, but it is needed to guide supply policies. Human population was 6.5 billion in 2005, with a 95% of population growing in developing world, and United Nations forecast near ten billion by 2050 (an increase of 40%). The developing economies need minerals for construction, energy, etc., and in the near future mining deeper will be necessary, with lower grades and larger scales, which means more health risks and carbon footprint will increase as well. Nowadays, the easy mineral resources, the least expensive to extract and process, have been mostly exploited and depleted (Bardy 2014). There are plenty of minerals left to extract, but at higher cost and increasing difficulty.

There are two competing views of mineral depletion: the fixed stock paradigm and the opportunity cost paradigm. Under the opportunity cost paradigm, mineral depletion is considered mainly a matter of economics and availability a function of price; long-term tendencies in real mineral prices indicate few problems of availability (Humphreys 2013). With regard to the effects of consumption on worldwide mineral supply, there are two classical schools of thinking: those who think that mineral resources are being depleted and consequently will be exhausted, and those who believe that there are infinite possibilities in the supply of mineral resources. Numerous studies about the quantification of supply risks of mineral and metallic raw materials have been carried out in the past 15 years due to the current tensions in raw material markets (e.g., Speirs et al. 2013; Glöser et al. 2015).

As a representative of the former school, there is an impending shortage of two fertilizers: phosphorus and potassium, and these two elements cannot be made, cannot be substituted, are necessary to grow all life forms, and are mined and depleted (Grantham 2012). Therefore, according to this author, their consumption must be completely decreased in the next 20 or 30 years or the humanity will start to starve. Grantham's article concluded that the society is actually in a hopeless situation related to these two raw materials. Similar are the conclusions of a study based on potential substitutes for 62 different metals in all their major uses and of the performance of the substitutes in those applications (Graedel et al. 2015). The study concludes that any of the 62 metals have exemplary substitutes available for all major uses. Other alarmist forecasts suggest that for some minerals and metals, depletion may occur over relatively short timescales of a few decades or even years. These forecasts are usually based on reserves estimates. It seems that they are untrustworthy previsions of the long-sighted accessibility of metals since their definitions depend on economics, present science, and technology (Graedel et al. 2014).

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Box 1.6

Hubbert Peak Theory

It is well known that there are three main laws to describe the depletion of any finite resource: (a) production starts at zero, (b) production then rises to a peak which can never be surpassed, and (c) once the peak has been passed, production declines until the resource is depleted. These simple rules were first described in the 1950s by Dr. Marion King Hubbert (American geophysicist) and applied to depletion of the world's petroleum resources. In 1956, Hubbert developed a theory (The Hubbert curve) now referred to as «The Hubbert Peak Theory» predicting that petroleum supplies did not come in an endless supply. The Hubbert peak theory says that the rate of oil production tends to follow a bell curve in which there is a point of maximum production based on discovery rates, production rates, and cumulative production. Later, production declines because of resource depletion, and finally the oil production would enter in a final decline. The theory can be applied to any given geographical area, from an individual oil-producing region to the planet as a whole. In fact, Hubbert has believed in 1956 that nuclear energy would become a long-term source of energy at a magnitude far greater than that of fossil fuels. This approach assumes that the level of production is mainly driven by technical factors that can reasonably be approximated by a logistic function.

Based on his theory, Hubbert presented a paper to the 1956 meeting of the American Petroleum Institute in San Antonio (Texas, USA), which predicted that overall oil production would reach the peak in the USA at 10.2 million barrels of petroleum/day between 1965 and 1970, which he considered an upper-bound. The term Peak Oil refers to the maximum rate of the production of oil in any area under consideration, recognizing implicitly that it is a finite natural resource subject to depletion. Almost everyone, inside and outside the oil industry, rejected Hubbert's analysis. However, the controversy raged until 1970, when the US production of crude oil started to fall: Hubbert was right. As an example, the oil production in the USA by the mid-2000s had fallen to 1940s' levels. Moreover, in 1974, Hubbert projected that global oil production would peak in 1995 «if current trends continue».

The theory does not take into account any other sources of petroleum besides crude oil. However, although unconventional oil is not included, the basic principle behind the Hubbert curve that production will eventually peak and decline still stands. Oil production in the mid-nineteenth century recovered 50 barrels of oil per barrel that was extracted. The number of barrels recovered today is 1–5 per barrel extracted. As ever, two views are possible, optimistic and pessimistic. Optimistic view holders predict a world peak in oil production around 2020, becoming critical closer to 2030. Pessimistic view holders believe that a peak has already happened. With the varying estimations of data, it is difficult to conclude when the world will peak or if it has already happened. Most data, however, support optimism, placing a peak date around 2020–2030. This date varies from Hubbert's predicted date due to the attempted regulation of Organization of the Petroleum Exporting Countries (OPEC) and the use of alternative energies and unconventional oil and gas. Regarding the former, predicting production for the OPEC has widely diverged from the Hubbert curve since the 1970s.

One of the most classical examples of mineral resource exhaustion school is the peak theory in the context of an earlier debate about the future of the US oil production (Hubbert 1956) (Box 1.6: Hubbert Peak Theory). Examining the world production of 57 minerals reported in the database of the USGS, Bardi and Pagani (2007) affirm that eleven minerals where production has definitely peaked and is actually declining while some more can be peaking or near peaking; adjusting the production curve with a logistic function, the definitive quantity extrapolated from the adjusting corresponds well to the quantity obtained adding the cumulative production so far and the reserves calculated by the USGS. The results obtained by these authors clearly indicate that in general the Hubbert model is valid for worldwide production of minerals and not just for regional cases.

However, the application of the peak concept to metals production has been criticized (Crowson 2011; Ericsson and Söderholm 2012). For instance, data from the last 200 years show that prices of major metals are mainly cyclical, with intermittent peaks and troughs closely to economic cycles. Thus, declining production is usually generated by falling demand rather than by declining resources or lack of resource discovery. For this reason, the peak concept is not very useful for modeling mineral resource depletion. The reserves could be «infinite», but other problems can arise. For example, the reserves of some raw materials (e.g., coal) seem to be very extensive, but the use of these raw materials has a negative impact on the world's climate. Therefore, climate policies can be a more restrictive factor on raw materials use than its availability. Moreover, many of the important raw materials are located in countries that are economically and/or politically unstable. Hence, exploitation of these resources originates local conflicts, a high risk of instability, and supply interruptions. Regarding the former, these are the so-called conflict minerals because miners are forced to take part in the illicit mining economy, and money earned from the sale of these conflict minerals is utilized mainly to promote violent causes and wars.

Regarding the second school of thinking, the changes of global reserves to global consumption between 1995 and 2010 for several minerals show that available reserves have been able to keep up with global demands. Overall analysis suggests that increasing demand and prices led to expansion of supply (Wellington and Mason 2014). The data also show that although there is a greater worldwide demand for a selected number of minerals, new sources are being exploited and the global mineral reserves should meet world demand for the next 50 years. In this sense, demand is likely to remain the dominant factor in world mineral supplies for the next few decades. Obviously, the problem of mineral resources supply can be diminished by the application of more detailed mineral exploration strategies, better mining and mineral processing technology, resource efficiency and improvements in recycling, and processes of substitution for many raw materials. For instance, in addition to identified copper resources of 2100 million metric tons (Mt), a mean of 3500 Mt of undiscovered copper is expected globally using a geology-based assessment methodology (Johnson et al. 2014).

On the other hand, history of mining proves that increasing demand for minerals and higher prices will generally lead to technological and scientific innovations that result in new or alternative sources of supply.

According to Lusty and Gunn (2015), widespread adoption of low-carbon mining technologies, supported by multidisciplinary research, and incremented global use of low-carbon power sources will enable challenges such as power consumption and varying the present link between metal production and greenhouse gas emissions to be met. In this sense, new focused research will improve the understanding of the processes mobilizing and concentrating these elements, enhancing the exploration models, and ability to identify new deposits. In addition, while it may generally be the case that properly functioning markets will provide solutions to mineral shortages, there are a variety of constraints on mineral supply response (Humphreys 2013).

1.8 Sustainable Development

The best way to mitigate the problem of mineral resources depletion is to use them in a sustainable way. The definition of sustainable development that is most commonly used today was presented by the United Nations in 1987 (The Brundtland Report). In this report, sustainable development is defined as «development that meets the needs of the present without compromising the ability of future generations to meet their own needs». This has become the most accepted definition of sustainable development internationally. This report stressed the need for the world to progress toward economic development that could be sustained without permanently harming the environment. In this sense, the discovery of new reserves may be viewed as only a temporary possible solution to mineral resource sustainability. Other potential solutions to sustainability of mineral resources include the following (Wellmer and Becker-Platen 2007): (1) improvement removal from the deposit, (2) finding new material to replace, (3) enhancing recycling processes, (4) decreasing consumption by more efficient use, and (5) looking for new possibilities.

On the other hand, the concept of sustainable production and consumption was implemented at the beginning of the 1990s. Its main goal is the correct production and use of natural resources, the minimization of wastes, and the optimization of services and products. Sustainable production and consumption intends to provide the utilization of goods and services over the life cycle so as not to jeopardize the needs of future generations (Sustainable Consumption Symposium in Oslo, Norway, 1994).

Regarding substitution of mineral resources to promote sustainable development, the importance of the hierarchy of relative mineral resource values must be emphasized. According to this model (Wellmer and Becker-Platen 2007), «the most valuable resources (energy resources) occupy the top of the hierarchy; the next lower value category consists of those mineral resources whose deposits are created by natural enrichment (for example, metalliferous deposits and some nonmetallic deposits like phosphate and barite); the next lower level consists of bulk raw materials such as those used in construction and those whose availability, from a geological viewpoint, is unlimited in the Earth's crust; finally, waste products and residues from beneficiation or burning of higher value resources occupy the lowermost part of the hierarchy». Obviously, if possible, the main goal of any policy leading to mineral sustainable development is to utilize low-value resources at the base of the mineral resources hierarchy. Thus, the high-value resources at the top are conserved. Another relevant tool for comparing materials consumption is eco-efficiency, which combines the notions maximum environmental and economic benefit and minimum environmental and economic cost. Eco-efficiency also decreases raw materials consumption throughout the life cycle to a limit more or less in line with the Earth's calculated capability. This concept focuses heavily on effective resource consumption and the reduction of waste (Fleury and Davies 2012).

The role of developed countries in sustainable development is crucial since these countries are mostly involved in mineral resource consumption and depletion. For instance, Europe environmental footprint is one of the largest on the planet; if the rest of the world lived like Europeans, it would require the resources of more than two earths to support them. For this reason, some developed countries such as EU countries have elaborated extensive programs trying to resolve the dualism rich countries – poor countries. As an example of these programs, the European Union (EU) is developing the named «Sustainable Consumption and Production and Sustainable Industrial Policy Action Plan». This plan includes different actions, of which 11 are devoted to Natural Resources. Europe 2020 strategy has as its flagship initiative a «resource efficient Europe: to help decouple economic growth from the use of resources», being the resource efficiency the key political priority.

Previously, the EU Raw Materials Initiative of 2008 included an integrated strategy based on the following three pillars: «(1) ensure access to raw materials from international markets under the same conditions as other industrial competitors; (2) set the right framework conditions within the EU to foster sustainable supply of raw materials from European sources; and (3) boost overall resource efficiency and promote recycling to reduce the EU's consumption of primary raw materials and decrease the relative import dependence».

There is a growing awareness that the construction structures, building, and other products in the economy today could be the urban mines of the future (OECD 2015). Thus, anthropogenic stocks have been less studied than geological stocks. They represent a growing area of interest, particularly in industrialized economies where geological stocks are limited but man-made stocks are believed to be large. Much of the study of anthropogenic stocks focuses on metals because they can be infinitely recycled, and unlike minerals, which dissipate with consumption (e.g., fossil fuels, salt for deicing), metals retain their chemical and physical properties over time. Many of the potential negative environmental impacts associated with the production and consumption of metals can be reduced using these anthropogenic stocks with recycling. Simultaneously, pressure on virgin stocks could be diminished.

1.9 Critical Raw Materials

The global market of mineral raw materials is characterized by: (a) increasing demand for minerals from both industrial and developing countries, (b) dramatic changes in where minerals are sourced, (c) volatile markets and pricing, and (d) increased vulnerabilities in the mineral supply chain. In this framework, modern society is increasingly dependent on mineral resources, which differ in their availability, the way of use, the cost of production, and their geographical distribution. Raw materials are essential for the development of the economy of industrialized regions. Sectors such as construction, chemicals, automotive, aerospace, and machinery are completely dependent on access to certain raw materials. European extraction covers only 29% of the demand for concentrates necessary to meet the requirements for production in metallurgical plants. Therefore, the potential effects of mineral supply disruption are essential for maintaining and improving the quality of life.

Moreover, a type of scarcity referred to as «technical scarcity» or «structural scarcity» presents a particular challenge and may be difficult and expensive to alleviate. Technical scarcity applies chiefly (Graedel et al. 2014) to «a range of rare metals used mostly in high-tech applications;

many of these are not mined on their own, rather they are by-products of the mining of the ores of the more common and widely used metals (e.g. aluminum, copper, lead, and zinc); these by-products are present as trace constituents in the ores of the host metals and, under favorable economic conditions, they can be extracted from these ores, which means that there is a little economic incentive to increase production at times of shortage».

Demand for a variety of mineral resources, such as rare earth elements (REEs), platinum group elements (PGEs), beryllium, and lithium, among others, has increased with continued consumption in developed economies and the emergence of other developing countries. Such elements are crucial to a variety of manufacturing, high-tech, and military applications. In this framework, many governments consider that a stable supply of some mineral resources is essential for economic prosperity. It is important to note that the production of minerals that supply many of these elements is concentrated in a few countries. Thus, China produces more than 95% of the global rare earth elements supply, and almost 80% of global platinum production is from South Africa. Table 1.4 (European Commission 2014) indicates the primary supply in percentage of some critical raw materials from most important producing countries. In this scenario, the concept of critical mineral or raw material must be introduced.

The term «criticality» was first used in 1939. The American administration decided in those days to build up a stock for 42 raw materials with military relevance. This was enforced by the so-called Critical Material Stock Piling Act. The geopolitical situation after the end of the Cold War relaxed, but the stock piling of military relevant raw materials is continued until today (Achzet and Helbig 2013). The National Research Council, in the book entitled *Minerals, Critical Minerals, and the U.S. Economy* (2008) mentions the difference between «strategic» and «critical», commenting that the terms «critical» and «strategic» as mineral or material descriptors have been closely associated but commonly not definitely differentiated. A mineral can be regarded as critical «only if it performs an essential function for which few or no satisfactory substitutes exist». Thus, the dimension of criticality is therefore related to the demand for a mineral that meets very precise specifications required in certain key applications, but it is not simply related to overall demand for all applications. Furthermore, a mineral can be regarded as critical «if an assessment indicates a high probability that its supply may become restricted, leading either to physical unavailability or to significantly higher prices for that mineral in key applications». Consequently,

Table 1.4 Primary supply (%) of some critical raw materials from the most important producing countries (European Commission 2014)

Critical raw material	Supply (%)	Major suppliers (>20%)
Antimony	93	China (87%)
Beryllium	99	USA (90%)
Borates	88	Turkey (38%)
		USA (30%)
Chromium	88	South Africa (43%)
		Kazakhstan (20%)
Cobalt	82	DRC (56%)
Coking coal	94	China (51%)
Fluorspar	84	China (56%)
Gallium	90	China (69%)
Germanium	94	China (59%)
Indium	81	China (58%)
Magnesite	86	China (69%)
Magnesium	96	China (86%)
Natural graphite	93	China (69%)
Niobium	99	Brazil (92%)
PGMs	93	South Africa (61%)
		Russia (27%)
Phosphate rock	66	China (38%)
REE (heavy)	100	China (99%)
REE (light)	100	China (87%)
Silicon metal	79	China (56%)
Tungsten	91	China (85%)
Total	90	China (49%)

the two main dimensions of criticality are importance in use and availability. Obviously, the criticality of a specific mineral can change overtime as production technologies evolve and new products are developed.

A more recent definition in the American Mineral Security Act of 2015 (US Congress) says that a critical mineral means «any mineral, element, substance, or material designated as critical pursuant to: (1) subject to potential supply restrictions (including restrictions associated with foreign political risk, abrupt demand growth, military conflict, violent unrest, anti-competitive or protectionist behaviors, and other risks throughout the supply chain); and (2) important in use (including energy technology-, defense-, currency-, agriculture-, consumer electronics-, and health care-related applications)». The term does not include here fuel minerals, water, ice, or snow.

With regard to the European Union, the last report on critical raw materials for the EU (2014) establishes that «non-energy raw materials are intrinsically linked to all industries across all supply chain stages, and consequently they are essential for EU way of life; sectors may rely on these materials as direct inputs, for instance metals refining relies on metallic ores as well as on industrial minerals; this primary industry underpins downstream sectors, which utilize processed materials in their products and services; thus, the healthcare sector uses equipment containing high performance magnets made from rare earth elements, electricity distribution relies on pylons and cables constructed of aluminium and copper respectively, and most vehicles are equipped with tyres that are comprised of natural rubber». The EU is in a particularly vulnerable position on imports for many raw materials (e.g., EU produced only 3% of the world metal production) which are increasingly affected by growing demand pressure. Moreover, the production of many materials is concentrated in a small number of countries. Supply risks may arise as a result of political-economic instability of the producing countries and export or environmental restrictions imposed by these countries.

To assess criticality, the methodology utilized in the EU is a combination of two components: economic importance and supply risk. The result

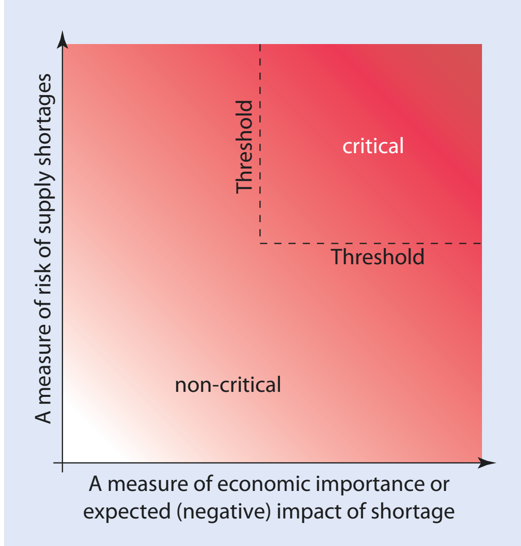

Fig. 1.22 General scheme of the criticality concept projected into two dimensions (Sievers et al. 2012)

is a relative ranking of the materials across the assessment components, with a material classified as critical if it exceeds both the threshold for economic importance and the supply risk (Sievers et al. 2012) (Fig. 1.22). For a country economy, the importance of a raw material is difficult to determine as it presents not only data but also conceptual and methodological difficulties. The analysis is carried out by evaluating the proportion of each raw material associated with industrial megasectors at an EU level and then scaled to define the overall economic importance for a material. On the other hand, the overall supply risks are a combination of factors such as substitutability, high concentration of producing countries with poor governance, and end-of-life recycling rates.

The last updated list of critical and noncritical raw materials (both metals or metallic ores and industrial minerals) from the European Commission (2014) includes 20 critical raw materials (Fig. 1.23). The main characteristics that make them critical for EU are: (a) the import dependence of the EU (generally more than 70%; in most cases 100%), (b) their use is fundamental in emerging technologies, (c) they are produced as by-products of other main metals treatment or coupled elements, (d) their recycling rate is quite low, and (e) the substitution options are limited.

Non-Critical Raw Materials	Clays (& Kaolin)				Gold
	Diatomite				Manganese
	Feldspar				Molybdenum
	Hafnium	Bentonite			Natural Rubber
	Limestone	Gypsum	Aluminium		Scandium
	Perlite	Potash	Copper	Barytes	Tantalum
	Sawn Softwood	Pulpwood	Rhenium	Bauxite	Tin
	Silica sand	Selenium	Silver	Iron Ore	Titanium
	Tellurium	Talc	Zinc	Nickel	Vanadium
EU Supply	>20%	<20%	<10%	<3%	<1%
Critical Raw Materials		Gallium	Silicon Metal	Chromium	Antimony
		Magnesite	Coking coal	Lithium	Beryllium
			Fluorspar	Tungsten	Borate
			Germanium		Cobalt
			Indium		Magnesium
					Natural Graphite
					Niobium
					PGMs
					Phosphate Rock
					REEs (Heavy)
					REEs (Light)

Fig. 1.23 Critical and noncritical raw materials in EU (European Commission 2014)

China is the major producer of the EU critical raw materials, and it is the most influential in terms of global supply. Other countries such as the USA for beryllium and Brazil for niobium provide specific raw materials.

The list of critical raw materials is being used to help prioritize needs and actions. It serves as a supporting element when negotiating trade agreements, challenging trade distortion measures, or promoting research and innovation. The list not only includes the name of the raw material, but also some data about main producers, main sources of imports, substitutability index, and end-of-life recycling input rate. These two last indexes are essential for the supply of critical raw materials. According the European list, «the substitutability index is a measure of the difficulty in substituting the material, scored and weighted across all applications; the end-of-life recycling input rate measures the proportion of metal and metal products that are produced from end-of-life scrap and other metal-bearing low-grade residues in end-of-life scrap worldwide». The European Commission adopted a strategy document as a result of previous considerations. The aim of this document is to secure and improve access to raw materials for the EU countries. Materials security and materials criticality have also been of growing interest for other international forums, leading to a number of studies and initiatives related to raw material supply and criticality (e.g., Resourcing Future Generations – IUGS).

1.10 Mineral Resource Recycling

Waste management priorities are organized according to the named «the three R's»: reduce, reuse, and recycling (Box 1.7: The Three R's). In a broad sense, there are three main groups of mineral resources that can be reused or recycled: construction and demolition waste, industrial minerals, and metals. Each of them has its own characteristics dealing with source and capability to be recycled. Recycling will never be 100% efficient and varies greatly among different mineral commodities due to the use and functionality in their respective applications.

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Box 1.7

The Three R's

Over the last half century, the amount of waste created per person in the developed countries has doubled. Thus, the concept and promotion of the three R's was created to help combat the drastic increase in solid waste production. As a rule, the three R's (reduce, reuse, and recycling) conserve natural resources, landfill space, and energy. The problem of the landfill space is nowadays very important, since siting a new landfill has become difficult and more expensive due to environmental regulations and public opposition. Moreover, the three R's ensure that future generations have clean air to breathe, clean water to drink, as well as forests, fields, and beaches to enjoy. For instance, it is possible to create a compost pile of the organic, biodegradable kitchen waste and apply it to the garden. Organic material in the compost stores carbon, keeping it from entering the atmosphere as a greenhouse gas. Nutrients in compost encourage healthy bacterial growth in soil, enabling plants to grow strong and healthy the natural way.

The three R's are really a waste management hierarchy with reduce being the most important strategy. The best way to manage a waste is not to produce it. Reducing waste yields the greatest environmental benefits. As less material is used, pollution from its manufacture and transportation is reduced, energy and water is saved, and material is kept out of landfills. Waste reduction therefore should be the main priority in the waste management plans. Thus, reducing is the most effective of the three R's. Aluminum cans are a good example of source reduction because they are now made with 1/3 less aluminum than they were 20 years ago.

The second most effective strategy for environmental stewardship is to reuse. An item might be reused for the same purpose or it might be used in a different way. Reusing extends the life of existing materials and decreases the resources needed for new products. This concept can be difficult because the life in the actual world includes many disposable items, and it takes some imagination and creativity to see how items can be reused. For instance, it is estimated that a glass beverage bottle can make about 15 round-trips between the manufacturer and the consumer before it must be recycled due to damage.

Finally, the last resort is to recycle. Recycling includes several steps that take a used material and process it, remanufacture it, and sell it as a new product. Where a product is recycled, a new life is given to the material and save it from going to the landfill. Materials like glass, plastic, aluminum, and paper can be mass collected (Fig. 1.24), broken or melted down, and made into entirely new products. The definition of recycling is varied. For instance, according to the Waste Framework Directive of the European Union (Directive 2008/98/EC on waste) recycling is defined as: «any recovery operation by which waste materials are reprocessed into products, materials or substances whether for the original or other purposes. It includes the reprocessing of organic material but does not include energy recovery and the reprocessing into materials that are to be used as fuels or for backfilling operations». Recycling again conserves resources and diverts materials from landfills. The possibilities of recycling are infinite. For example, metals form almost 9% of the municipal solid waste (MSW) in the USA in 2012 (EPA 2014). Sources of metals from MSW include residential waste (including waste from apartment houses) and waste from commercial and institutional locations, such as businesses, schools, and hospitals, and these include items such as packaging, food waste, grass clippings, sofas, computers, tires, and refrigerators.

Fig. 1.24 Recycling at Grand Canyon (USA)

1.10.1 Construction and Demolition Waste

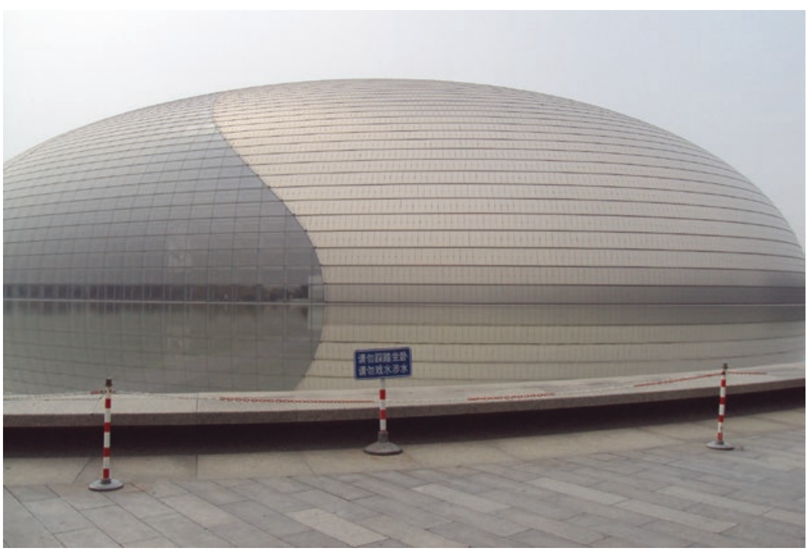
Construction and demolition waste (CDW) is one of the most significant waste streams in the world. It comprises very many materials such as metals, glass, concrete, gypsum, bricks, wood, plastic, solvents, and excavated soil, among others, many of which can be recycled. CDW has high potential for recycling because many of the components have high resource value. In particular, there is a market for aggregates derived from CDW waste in roads, drainage, and other construction projects. For instance, recycled and secondary materials account for 30% of the aggregates market in Great Britain. They include construction and demolition waste, asphalt planings, used railway ballast, etc. In the European Union, CDW has been identified as a priority waste stream since it accounts for approximately 25% to 30% of all the waste generated in the EU; approximately 900 million tons per year, two tons per capita. The quantitative target set by The Waste Framework Directive of the EU at 2008 is the following: «by 2020, the preparing for reuse, recycling and other material recovery, including backfilling operations using waste to substitute other materials, of non-hazardous construction and demolition waste … shall be increased to a minimum of 70% by weight». Concrete is the most important fraction in the CDW. It presents many treatment options (e.g., landfill, recycling into aggregates for road construction, or backfilling, among others), but barriers to recycling the waste are numerous. Among them, the misconception about the quality of recycled products compared to new materials is the most important, since ignorance of the good results of these materials in some applications probably will continue for years.

1.10.2 Industrial Minerals

The valuable physical properties of many minerals used in industrial and manufacturing processes are either destroyed in use or the minerals are dispersed, and they cannot be recoverable in their original form. Thus, plasticity of ceramic clays is lost during firing in the kiln. Some industrial minerals that are valued for their chemical properties are impossible to reuse or recycle. The most classical example is salt utilized to treat roads in the winter, and potassium or phosphorous minerals that are the basis of numerous agricultural fertilizers. However, many industrial minerals can be recovered and recycled in their manufactured form. For example, ceramic materials can be recycled as construction fills or as aggregates. Glass is an outstanding case of material with high recycling capacity (Fig. 1.25). It is a manufactured product that may simply be melted and reformed in a similar way than metals.

According to IMA Europe (2013), «in general, recovering these minerals from their end applications would be technically complicated, time

Fig. 1.25 Glass building (The National Grand Theater of China, Beijing)



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consuming and, ultimately, environmentally unsound; however, although the minerals themselves may not be recyclable per se, many of them lead second, third, fourth or even an infinite number of lives». The Industrial Minerals sector in Europe estimates that a total 40–50% of all the minerals consumed in Europe are recycled, which is the case for about 73% of all silica used in Europe. Markets for this recycled silica are varied: construction and soil, container and flat glass, foundry, ceramics, etc. Other data about recycling rates for industrial minerals in Europe are 50% of bentonite, 58% of calcium carbonate, 67% of feldspar, 49% of kaolin, or 60% of talc.

1.10.3 Metals

Despite the vast reserves of several industrially important metals, the growing world population cannot keep consuming metals at current standard for the western industrialized society. This is no doubt beyond what is likely to be sustainable. Altogether, metal production today represents about 8% of total global energy consumption and a similar percentage of fossil-fuel-related CO_2 emissions. Obviously, recycling will help in decreasing this footprint as it usually requires less energy than primary manufacture (UNEP 2013). When recycling metals, energy use is diminished because scrap metals commonly require less energy to convert back into high-grade materials than mining and refining processes. For this reason, carbon emissions from recycling are substantially lower than those derived from mining. As a rule, the main benefits from recycling metals are: (a) lowering energy consumption by 60–95% compared to primary production, (b) reducing CO_2 emissions and environmental impact on water and air, (c) preserving primary geological resources, and (d) decreasing the dependency on raw material imports. Depending on the metal and the form of scrap, recycling can save as much as an indicator of ten or twenty in power consumption (Reck and Graedel 2012).

In the metals industry, the term recycling is commonly used to include two fundamentally different kinds of scrap: (1) new scrap or process scrap: the material generated during processing and manufacturing, and (2) old scrap (Fig. 1.26) or obsolete scrap (also post-consumer scrap or end-of-life scrap): the material recovered after being built into a construction or a manufactured article that has been used and eventually discarded. Thus, scrap is generally categorized as new scrap or old scrap. A broad range of terms, such as external scrap, home scrap, internal scrap, mill scrap, prompt scrap, and purchased scrap, have been developed to design scrap originated by different industry operations (Papp 2014).

Hagelüken (2014) affirmed that recycling possibilities or recyclability of a product is based on various technical, economic, structural and

Fig. 1.26 Old scrap

organizational factors: (a) the intrinsic metal value of the base material depending on its absolute metal content and the metal price, and determines the economic attractiveness of recycling; (b) the material composition beyond the chemical composition to include physical characteristics such as shape, size, and the type of connection between materials and components; and (c) the application field of a product and how it is used referring to the area of use while the latter deals with new or reuse products, user behavior, risk of dissipation, etc.

Few metals are used in pure form, and most are components of alloys or other mixtures. In cases where these materials undergo reprocessing, some elements will be reprocessed to their elemental form (e.g., copper), but many will be reprocessed in alloy form (e.g., nickel or tin) (Reck and Graedel 2012). Tercero (2012) suggests that a main obstacle to recycling is the complexity of the products themselves. There are many difficulties such as the energy and labor required to separate the materials of interest so that they can be recycled. Sometimes, an adequate large-scale technology is not available (locally or worldwide) to recover the desired materials in a useful quality. This is the case for phosphors in energy-saving lamps, which are to date not recycled on a large scale. There is also an important difficulty yet recycling might be possible but too expensive given current technology and prices, forcing downcycling or preventing recycling altogether. An example of downcycling is lithium from discarded lithium-ion batteries. It is currently possible but too expensive to produce technical grade lithium carbonate out of recycled lithium compared to primary production.

In spite of the resulting benefits from an environmental, economic, and social perspective, current recycling rates are still rather low for most metals (Table 1.5). The world's most recycled material is steel, the metal used in 8–9 times greater quantity than all other metals combined. Of the three R's, recycling is probably the most recognized attribute of steel. «More than 475 million tons of steel scrap was removed in 2008 from the waste stream into the recycling stream; this is more than the combined reported totals for other recyclable materials, including paper, plastic, glass, copper, lead, and aluminum; in 2012, the United States recycled 69 million metric tons (Mt) of selected metals, an amount equivalent to 59% of the apparent supply of those metals, and more than 91% of recycled metal was steel» (Papp 2014). Obviously, steel recycling has an enormous impact on the reduction of CO_2 emissions. However, the greatest request for logistical and technological advance in steel recycling is in recovery and processing of scrap, covering enhancement in contaminant removal and recovery (Bowyer et al. 2015).

Table 1.5 Recycling rates in the USA (Papp 2014)

Element	USA (%)
Aluminum	52
Chromium	28
Copper	33
Iron and steel	50
Lead	68
Nickel	45
Tin	27
Titanium	63

One of the most promising recycling sources is waste electronic and electrical equipment (WEEE), which contains many of the metals of rising demand. Much of WEEE is typically metal, not only the 60% «metals» slice, but also the metal and metallic compounds found in printed circuit boards, LCD screens, cables and metal-plastic mixes, etc. Table 1.6 (Bakas et al. 2014) shows the critical metals included in EEE and hence in WEEE. With increasing gross domestic product (GDP), world consumption of these products accelerates and the size of their waste streams increases. WEEE volumes are already enormous, estimated between 20 and 50 million tons per annum, or 3–7 kg/person each year (assuming seven billion people) (UNEP 2013).

In summary, recycling represents a major way to mitigate negative impacts on increasing metals' demand and to ensure the potential of economic growth. For instance, the largest recycling park in China is able to recover one million tons of copper per year. It is important to bear in mind that the largest copper mine in this country generates less than half of that amount of copper. This «urban mining» (anthropogenic stock) is important in producing recycled raw materials. Hence, reinforcing the recycling of metals is a clear strategy for a sustainable future (UNEP 2011).

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Table 1.6 Critical metals included in EEE (Bakas et al. 2014)

Metal	Mobile phones	PC	Flat-screen TVs and monitors	Solar power converters	Rechargeable batteries	Notebooks/laptops
Cobalt	+[a]		+		+	+[a]
Indium	+	+	+	+	+	+
Lithium	+[a]		+		+	+[a]
Silver	+	+	+	+	+	+
Tantalum	+		+	+		+
Tellurium		(+)		+		
Tungsten	+	(+)	+			
Gold	+	+	+			+
Beryllium	+	+				+
Gallium	+	+	+	+		+
Germanium	+	+		+		+
Palladium	+	+	+			+
Ruthenium		+	+			+

[a]Within batteries

1.11 Trade and Markets

International trade in metals and gemstones has been carried out from two or more millenniums ago or more, but intercontinental traffic in minerals and metals of all kinds increased quickly in the last two centuries. Nowadays, all minerals, metals, and related products enter into the world trade, and some of them are produced for local consumption while others are transported to markets worldwide. In this sense, geology and economics have constrained the location of the minerals and metals that can be economically mined, so many of them need to be transported.

Most transoceanic minerals trade accounts for relatively low-value bulk minerals (e.g., iron ore Fig. 1.27 or phosphate rock), since economic deposits are few and the minerals must be transported for further processing in the industrial countries. Metalliferous ores, such as copper, lead, zinc, and nickel ores, are usually concentrated just near the mine, and they are also subject to a significant world trade. This is because of their high mining and processing costs and geographically restricted mineral deposits' character. The high value of these metals is commonly associated with the rarity of their economic deposits, which means that they can be transported over long distances and still they can maintain their value.

The commercial arrangements that govern minerals trade are the markets. A «market» is a hypothetical place where sellers and buyers of a selected commodity meet to determine its price. Mineral markets are material goods markets, and many are regarded as world markets because of the easy negotiability of the traded commodities. Markets may exist at several stages of production and for several levels of quality. However, the market usually referred to is that of the standard trade quality for the mineral commodities most important in world trade (Gocht et al. 1988). For instance, this is the case of Arab Light (34°API)

Fig. 1.27 A merchant vessel in Saldanha Bay (South Africa) September 2015 to transport iron ore from Kumba Iron Ore business (Anglo American) to Asian markets (Image courtesy of Anglo American plc)

for crude oil, although the OPEC basket of crudes daily price is made up to 12 crudes. The form of market is defined by the extent of free competition and hence pricing (oligopoly, monopoly, and competition), the latter being usually a result of peaceful adjustment.

Regarding the competitive prices, mineral commodities, particularly metals, can be standardized according to quality and quantity and are thus negotiable on exchanges. Here, prices are subject to competitive supply and demand. The major base metals and the precious metals are traded in open markets. Open market prices are very important price indicators used by all the parties concerned although most trade is through private contracts. Many bulk nonfuel minerals, such as iron ore, coal, and potash, or metals such as titanium tungsten, and uranium, are almost entirely set by private contracts between major supplies and consumers. Since they are bulk commodities, transportation and shipping become an important aspect of the contract price. In these cases, although the prices are set by contract, a number of organizations (e.g., UX Consulting Company for uranium prices) publish contract prices, which provides a level of transparency to the metal prices even though they are not traded on an open market (Stevens 2010).

At present, there are two important metal exchange in the world where competitive prices are fixed: The London Metal Exchange (LME) (Box 1.8: London Metal Exchange) and The New York Mercantile Exchange (NYMEX), including COMEX, the division responsible for metals trading. Related to NYMEX and its activity, the new entity was closely devoted mainly to energy (oil, natural gas, coal, etc.), although some ferrous products are also traded. In both LME and NYMEX, the prices for the commodities are established on a supply/demand basis on an open market. At a regional level, the Asian Market offers price metals for base metals, minor metals, ferroalloys, rare earths, precious metals, scrap metals, carbon steel, stainless and special, steel raw materials, refractories, and industrial minerals. Also at a regional scale, APMEX includes a complete information about precious metals in the USA.

Regarding the relationship between stock markets and commodity prices, it has received substantial attention over the past decades. An investigation based on the commodity-stock market nexus in gold and other metals, as well as the relevant stock market indices and stocks traded on the Toronto Stock Exchange for the period 1982–2011, shows that there is no indications that the market states detected for the individual stocks are related to those for the raw material price (Ntantamis and Zhou 2015). The objective of the investigation was to assess whether commodity price fluctuations were reflected in the stock prices of firms whose primary business is in extracting and trading the particular commodity.

Box 1.8

London Metal Exchange

The London Metal Exchange is the world center for industrial metals trading. More than 80% of global nonferrous business is transacted there, being the annual trading about USD$ 12 trillion. The prices on the London Metal Exchange are globally recognized and respected by the industry. The Exchange provides producers and consumers of metal not only with a physical market of last resort but also, perhaps most importantly of all, with the ability to hedge against the risk of rising and falling world metal prices. LME contracts trade in US dollars although exchange rates for euro, GB sterling, and Japanese yen are also published. LME pricing has the following advantages: (a) unique price set by supply and demand, (b) transparent, (c) traded and tradable real-time prices, (d) heavily regulated market, and (e) more accurate hedging. Contrary to popular belief, gold and silver (precious metals) are not traded on the London Metal Exchange but in London Bullion Market.

The origin of the LME can only be traced back as far as the opening of the Royal Exchange in London in 1571. It was there that traders in metal and a range of other commodities began to meet on a regular basis. In the early nineteenth century, there were so many commodity traders, ship characters, and financers using the Royal Exchange that it became impossible to do business and individual groups of traders set up shops in the nearby city coffee houses. At that moment, a merchant with metal to sell would simply draw a circle on the dusty floor and call out «change» at which point all those wishing to trade would assemble around the circle and make their bids. It was the origin of the so-called ring trading. In 1869, the opening of the Suez Canal reduced the delivery time of tin from Malaya to match the 3-month delivery time for copper from Chile. This gave rise to the LME's unique system of daily prompt dates for up to 3 months forward which still exists to this day.

As delivery tonnages grew to meet the increasing demands of the industry, more and more merchants were attracted to the metal trading. New metals have been introduced as demand dictated. For instance, copper and tin have been traded on the LME since the beginning, but lead and zinc were officially introduced in 1920, primary aluminum was introduced in 1978, and was followed by nickel 1 year later. In 2008, the LME made a move into ferrous metals with the introduction of two regional contracts for steel billet. In July 2010, these contracts merged into a single global contract. The most recent contract additions came in February 2010 with the launch of two minor metals futures contracts for cobalt and molybdenum. Currently, metals officially listed in LME are nonferrous metals (aluminum, copper, lead, nickel, tin, and zinc), steel, and minor metals (cobalt and molybdenum). The LME is also home to the LBMA Platinum and Palladium prices, which are discovered in a twice-daily auction. Each commodity information (e.g., copper) includes topics like stocks and prices, price graph, historical data, average prices, production, and consumption, among others. The date structure is as follows: daily from cash to 3 months; then weekly from 3 months to 6 months (every Wednesday); then monthly from 7 months (every third Wednesday); aluminum and copper contracts trade out to 10 years.

1.12 Mining as a Business

According to Graedel et al. (2014), expenditures included in creating a new mine and to put it into production nowadays commonly amount to hundreds of millions of USD or more than a billion of USD for a big mine on a greenfield place. In this sense, a metal mine commonly operates for a decade as the minimum period although, depending mainly on economic circumstances, it can go ahead for more than 100 years (e.g., Reocín mine, a lead-zinc deposit in northern Spain, operated continuously from 1856 to 2003) (Fig. 1.28).

The mining industry is a very large one, with a market capitalization of hundreds of billions dollars in FTSE100 index (this index includes the 100 companies listed on the LSE with the highest market capitalization). London lies as the heart of the industry and hosts the headquarters of some of the largest and more important mining companies in the world. The structure of the global mining industry is composed by three main types of mining companies. The largest companies are the so-called majors, and they operate across many geographies and minerals (e.g., BHP Billiton, Rio Tinto, or Anglo American). A second type is formed of companies more focused to a commodity or a country. Examples are Freeport Mc MoRan or Antofagasta, focused on copper; Barrick Gold or Newmont Mining, focused on gold; De Beers focused on diamonds; or Norilsk Nickel and Kazakhmys, both operating in the ancient Soviet Union. Finally, a third type is constituted by small companies, ranging from companies with two or three mines to small family companies. Some of these companies produce for international

Fig. 1.28 Reocín mine (Santander, Spain): **a** 1993; **b** actually

markets, but others just supply local markets. They often operate in markets where demand is small or where mineral deposits can be mined at small scale such as mining of precious metals.

On the other hand, the major mining companies can be subdivided in two groups according the property of the company. The first one, formed by the companies mentioned above, quotes in stock markets, from which their capitalizations are derived. The second group is formed of companies that are either wholly or predominantly owned by the states. Classical examples are Codelco owned by the State of Chile, which is the world's largest copper producer, and several Chinese companies (e.g., China Shenhua, Yanzhou Coal, Chinalco, etc.).

The other essential actors in the mineral market are the exploration companies. Since exploration is an extremely high-risk activity and the majority of exploration ends in failure with investors losing their money, the exploration companies use to have their own sources of funding, mainly stock markets. Examples of these markets for exploration companies are the Toronto Stock Exchange (TSX) or the Australian Stock Exchange (ASX). However, the global economy is formed not only of companies and consumers, but also of nations. Thus, nations have strategic interests and they view the mineral products in terms of the contribution that they can make to national projects.

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1.13 Questions

Short Questions

1. What is an ore?
2. Define the concept of grade in a mineralization.
3. What is a prospect in mineral exploration?
4. What are the two main edges of classification in McKelvey Box?
5. Set an example of the relevance of salt for humanity.
6. What is the mining cycle?
7. List the three categories of mineral resources according to International Codes.
8. What are the names of the studies carried out to establish the overall characteristics of a mining project in each stage of its development?
9. How enrichment factor for metals is calculated?
10. Explain in summary form the Hubbert peak theory.
11. Define the concept of sustainable development
12. What is the purpose of the concept of critical minerals?
13. Explain the concept of «the three R's» in mineral resource recycling.
14. What is the reason of transoceanic trade of metallic ores?
15. What is a market in minerals trade?

Long Questions

1. Outline the main stages of the mining cycle.
2. Define the concept of mineral resource and mineral reserve in International Codes. How do the modifying factors influence these definitions?

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Mineral Deposits: Types and Geology

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Summary

This chapter provides a brief introduction to the types of mineral deposits. These descriptions include both the commodities and the geology of the deposits. Previously, the evolutionary concepts about the origin of mineral deposits are established, giving a special consideration to the neptunism-plutonism controversy in the nineteenth century and the influence of plate tectonics theories in the genesis of mineral deposits. Ore-forming processes (magmatic, sedimentary, hydrothermal, and metamorphic) are described before entering in the description of the main types of mineral deposits. Energy (petroleum, natural gas, tar sands, bituminous shales, coal, and uranium), metals (iron and steel, base metals, precious metals, light metals, and minor and specialty metals), and industrial minerals and rocks (aggregates, ornamental rocks, carbonate rocks for cement and lime, and clays for brick and tiles) form the main groups of mineral commodities. From a geological point of view, a simple genetic classification of mineral deposits encompasses four main groups: magmatic, hydrothermal, sedimentary, and metamorphic/metamorphosed, each of them with several types and subtypes.

2.1 Introduction

Mineral deposits are concentrations in the Earth's crust of helpful elements that can be extracted at a profit. By definition, ores are somewhat unusual rocks. Like all crustal rocks, they consist of minerals formed through a variety of geological processes that collect the elements into a minor volume. One cubic meter of crustal rock contains approximately 0.15 kg of nickel, but the cost to mine and process this amount of rock clearly exceeds the value of the resulting nickel. For this reason, the existence of a concentrating geological process is crucial. The great goal of geologists is to know how the nature works to put all the elements into mineral deposits, trying to understand how these processes work. One of the most common expressions in mineral deposit is the association of specific ore types with determined host-rock assemblages; this expression is the ore-host-rock association (e.g., Stanton 1972). In general, this association represents the geological environment and processes that formed the mineralization. Several associations have been commonly and successfully utilized for searching new deposits.

McQueen (2005) suggested four basic geological requirements for any ore deposit to form: «(1) a source for the ore components; (2) a mechanism that either transports these components to the ore deposit site and allows the appropriate concentration or removes non-ore components to allow residual concentration; (3) a depositional mechanism (trap) to fix the components in the ore body as ore minerals and associated gangue; and (4) a process or geological setting that allows the ore deposit to be preserved». Other requirements comprise energy to power the transport mechanism and an appropriate crustal structure to locate the ore-forming components and reach their deposition. Therefore, the particular elemental composition of a mineral deposit results from a complicated interaction of favorable combinations of source, transport, and depositional variables. Thus, the type, character, and abundance of an ore deposit reflect the environment in which it was formed. It preserves evidence for the evolution of ore-forming processes and tectonic setting as well as the characteristics of the atmosphere and hydrosphere (Jenkin et al. 2015).

Ever since Agricola (1556) first classified ore deposits (◘ Fig. 2.1), successive writers have attempted classification of mineral deposits (Jensen and Bateman 1979). Classifications are very useful because they mainly provide a common reference scheme. Moreover, they are utilized for both scientific communication and practical application. A classification scheme is basically a means of grouping together known geological processes, minerals, and mineral-rock association. With regard to genetic classification of mineral deposits, including geological processes of ore formation, stringent genetic classification is very difficult. In this sense, some deposits result from interplay of volcanic, intrusive, sedimentary, and diagenetic processes. However, it is necessary to remember that genetic concepts are an essential component to find new mineral deposits. Thus, the genesis must be reflected in mineral deposit classification (Jenkin et al. 2015).

In this chapter, two main classifications of mineral resources are described: one is based on commodities, whereas the other is made

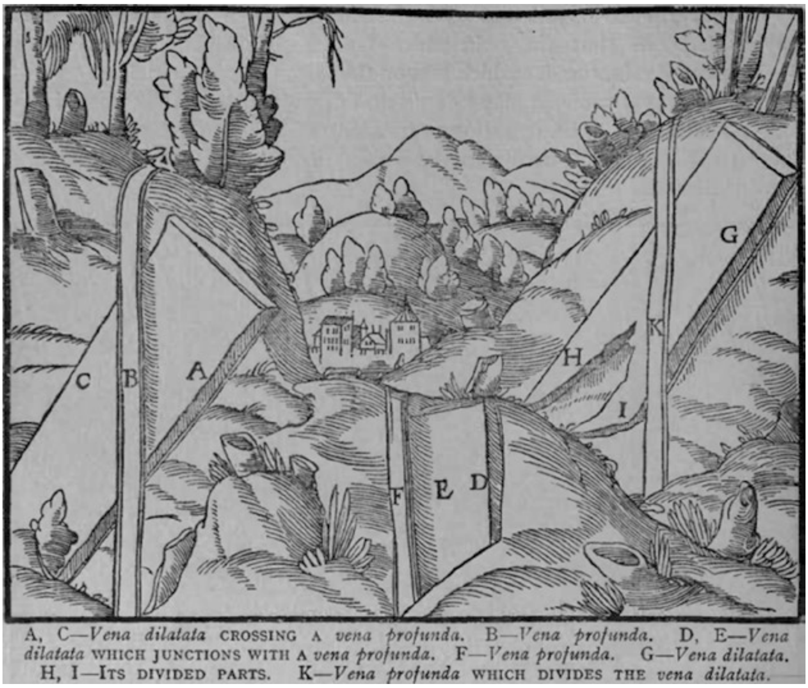

Fig. 2.1 Some types of veins according to Agricola (1556)

according to the ore-forming processes and genesis. A combination of both classifications makes it possible to describe in detail the overall characteristics of mineral deposits.

2.2 Basic Vocabulary

There is basic vocabulary dealing with formation of mineral deposits which is not used in other disciplines of mineral resources such as evaluation, exploitation, or environmental impact. Some terms are genetic, others are related to the geometry of the ore, and most of the following definitions are similar to those included in the *Glossary of Geology* (Bates and Jackson 1987). Since metallogeny is the synthesis of scientific endeavors to understand ore formation (Pohl 2011), expressions such as metallogenic maps (Fig. 2.2), metallogenic provinces, and metallogenic epochs are usually found in the literature related to mineral deposits. A metallogenic province may be defined as a mineralized area or region containing mineral deposits of a specific type or a group of deposits that possess features (e.g., morphology, style of mineralization, or composition) suggesting a genetic relationship; a metallogenic epoch is a geological time interval of pronounced formation of one or more kinds of mineral deposits (Turneaure 1955). The size of a metallogenic province can be as large as the Superior Province (Canadian Shield), and a metallogenic epoch can be as broad as the entire Proterozoic. A detailed way to define metallogenic epoch and metallogenic province is «as those time intervals of Earth history and regions of Earth, respectively, which contain a significantly greater number of deposits or larger tonnage of a specific deposit type than would have resulted from average rates of mineralization that have occurred over Phanerozoic time» (Wilkinson and Kesler 2009). Another relevant term is metallotect, a geological, tectonic, lithological, or geochemical feature that is believed to have played a role in the concentration of one or more elements and hence is thought to have contributed to the formation of ore deposits.

The use of genetic terms is also very varied. Thus, syngenetic denotes that ore or minerals have formed at the same time as their host rock (a rock serving as a host for a mineral or ore); it is commonly but not only used for sedimentary rocks. By contrast, epigenetic means that the ore or minerals have emplaced in pre-existing rocks of any origin (e.g., veins). Both terms are essential and commonly used in genetic descriptions of mineral deposits, although they have caused intense controversies through time. Other used terms are hypogene and supergene. The former

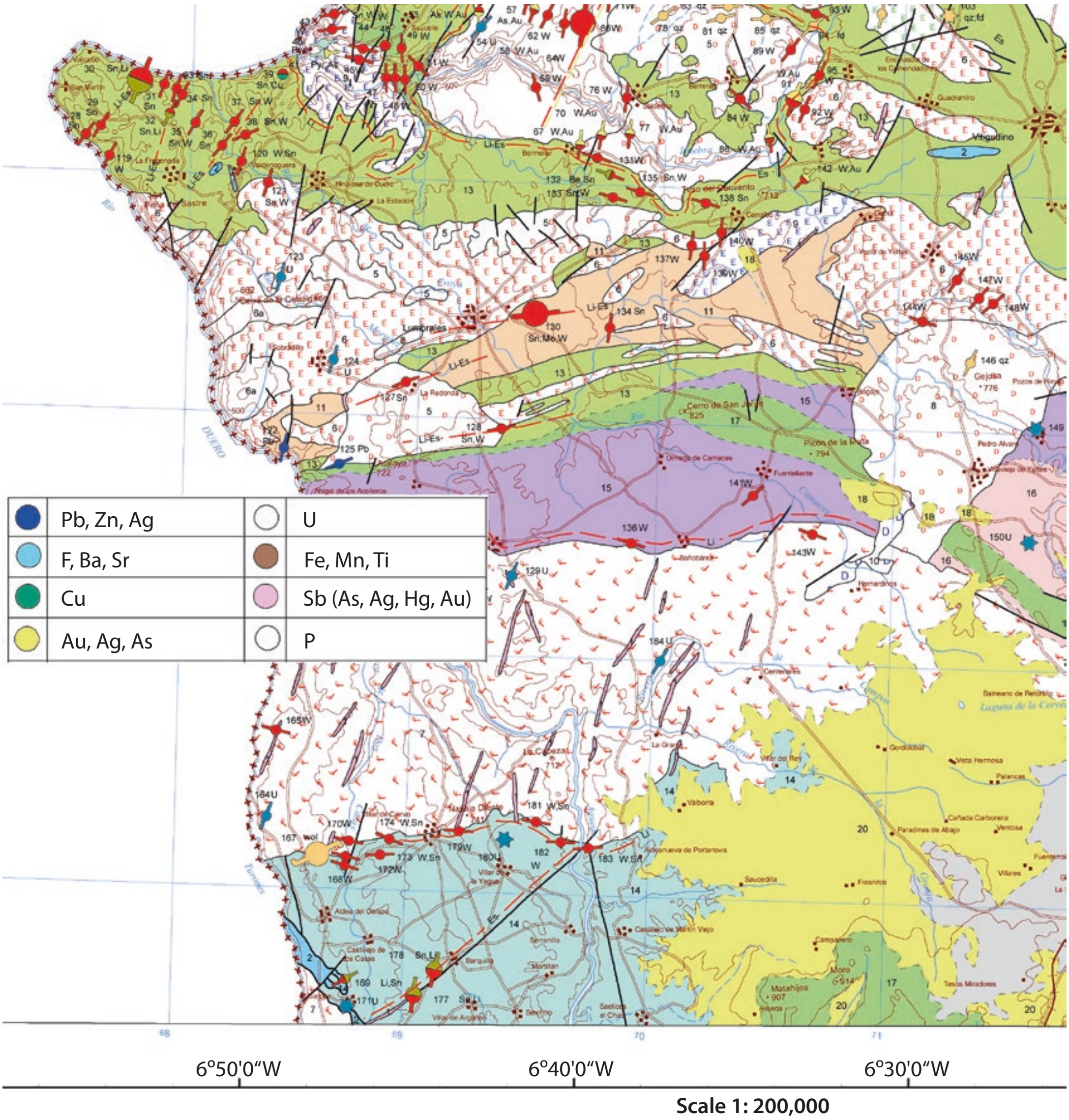

Fig. 2.2 Part of a Spanish metallogenic map (IGME)

refers to ores formed by ascending solutions, whereas the latter deals with ore formation by descending solutions, classically meteoric waters interacting with rocks during surficial weathering. Endogenetic indicates concentrations caused by processes in the Earth's interior (e.g., magmatism), whereas exogenetic points to concentration caused by processes in the Earth's surface (e.g., sedimentation). Stratiform and stratabound are also essential terms in mineral deposits formed by sedimentary processes. Thus, a stratiform deposit means a mineral deposit related to a concrete stratigraphic bedding, while a stratabound deposit is limited to a determined part of the stratigraphic column.

Many terms are used in relation to the shape of a mineral deposit since it is very variable, from concordant tabular and stratiform to discordant veins and breccia bodies. Veins are sheetlike zone of minerals that fill a fracture; they are formed from hydrothermal solutions and commonly composed of quartz and/or carbonates with minor sulfide minerals. A breccia is a fragmented rock in which the clasts are cemented together by minerals; it is a good host for hydrothermal mineral deposits. Other terms are disseminated

Fig. 2.3 Stockwork texture

(ore minerals dispersed through the host rock), stockwork (an interlacing network of small and narrow, close-spaced ore-bearing veinlets traversing the host rock) (Fig. 2.3), massive (mineralization comprising more than 50% of the host rock), tabular (an ore zone that is extensive in two dimensions but has restricted development in its third dimension), vein type (mineralization in veins, commonly discordant to the host rock layering) (Misra 2000), pipe shaped (the mineralization body has the form of a carrot and is typical of diamond deposits), and lens shaped (the mineralization body is much thicker in the center than around the edges, and it may be flat lying, dipping, or vertical).

The use of terms associated with formation temperature of ore deposits is common. Examples are epithermal (formed at less than 1500 m and temperatures between 50 and 200 °C), mesothermal (originated at intermediate depths, 1500–4500 m, and temperatures between 200 and 400 °C), and hypothermal (formed at greater than 4500 m and temperatures between 400 and 600 °C).

Mineral deposits can be named according to different criteria. Sometimes the name of a place, region, or city is used (e.g., Alpine type, Sudbury type, Cyprus type, Mississippi Valley type). Other times the deposits are known using their acronyms (e.g., BIF means banded iron formation ores, MVT means Mississippi Valley-type lead-zinc ores, or SEDEX means sedimentary exhalative ore). In addition, the deposits may be called according to the rock type, like pegmatite (large crystals), porphyry copper (disseminated stockwork linked to plutonic intrusives), and skarn (calc-silicate rock). Finally, deposits can be known by their shape, being the most representative example a type of uranium deposits, namely, roll-front uranium deposit.

2.3 Evolutionary Concepts About the Origin of Mineral Deposits

Agricola (1556) formulated the first reasonable theory of ore genesis. In his book «De Re Metallica, » he showed that «lodes originated by deposition of minerals in fissures for circulating underground waters, largely of surface origin, that had become heated within the Earth and had dissolved the minerals from the rocks.» Agricola made a clear distinction between homogeneous minerals (minerals) and heterogeneous minerals (rocks). Little progress was made in the study of ore genesis from the time of Agricola until the middle of the eighteenth century. By the 1700s, more remarkable progress was made in Germany, in the Erzgebirge mining district (Henke, Zimmerman, and Von Oppel, among others). At the end of the eighteenth century, the polarized views of either plutonist or neptunist theories were developed (Box 2.1: Neptunism vs Plutonism).

At the middle of the nineteenth century, Von Cotta affirmed judiciously the various theories of mineral genesis and correctly concluded that no one theory was applicable to all ore deposits. At the end of the 1800s and starting the 1900s, different authors (e.g., Élie de Beaumont, Bischoff, Hunt, Phillips, Sandberger, Posepny, Emmons, and many others) created a new controversy related to the descensionist, ascensionist, and lateral secretionist theories. Lindgren proposed, in his book «Mineral Deposits» (1913), a classification of mineral deposits based on their origin,

2

Box 2.1

Neptunism vs Plutonism

For the origin of mineral deposits, Abraham Gottlob Werner (1749–1817), father of neptunism (denominated after the Roman God – Neptune – of the sea), discarded early theories about interior source for the metals. Although Werner was not the first to propose water as origin of the rocks, he was the most consequent supporter and divulgator of this theory. Werner was a careful mineralogist who drew up an excellent system of classification of minerals based on their properties. He became an insistent advocate of the theory that mineral veins were formed by descending percolating waters derived from the primeval universal ocean, from which not only sediments but all the igneous and metamorphic rocks were precipitated. Because of his theory that what are known today as igneous rocks originated in the sea, Werner and his followers were called neptunists.

According to Werner, by successive sedimentation onto an irregular terrestrial core, four types of formations were supposed to be deposited: (1) primitive, crystalline rocks such as granite and gneiss; (2) transitional, limestones, slates, and quartzites; (3) floetz, the layered rocks from the Permian to the Cenozoic; and (4) alluvial, (superficial) deposits. The primitive formations would be found in the central parts of mountain ranges, from which the water would have withdrawn first. Thus, the granites were overlain by other layers of crystalline rock (metamorphic), followed by layers of sedimentary rock formed as a result of erosion of the primitive crystalline rocks and subsequent deposition. Rocks resulting from observed volcanic eruptions were attributed to the local action of «subterraneous fires.» In fact, geologists at that time had a clear understanding of the formation of many mineral ores, especially gold, which is generally formed by precipitation and fluid-induced changes. Therefore, these processes are more similar to ideas of neptunism than to plutonism.

Opposite to Werner's ideas, James Hutton (1726–1797), a prominent member of the Edinburgh scientific community (the Royal Society of Edinburgh was at that time one of the most active scientific bodies in the world), defined in his book entitled «Theory of the Earth» the true origin of magmatic and metamorphic rocks and applied his magmatic theory not only to rocks but also to all mineral deposits. He delivered his theory in two lectures to the Royal Society of Edinburgh in the spring of 1785. Hutton claimed that ore minerals were not soluble in water but were igneous injections, being thus one of the founders of plutonism (named after the classical mythology God – Pluto – of the underworld).

He recognized the significance of the intergrowth texture between quartz and feldspar in a sample of coarse-grained graphic granite and concluded that granite might have «risen in a fused condition from subterranean regions» and that the country rock should therefore be broken, distorted, and veined. Hutton also recognized the importance of unconformities and pointed out that many igneous rocks clearly intruded surrounding rocks and therefore were younger. Because Hutton and his followers held that igneous rocks came from molten material within the Earth, they were called plutonists, being thus Hutton the founder of plutonism. The controversy between plutonism and neptunism continued into the nineteenth century, and eventually the plutonist views on the origin of rocks prevailed.

whether they were products of mechanical or chemical concentration and, if chemical, whether they were deposited from surface waters, from magmas, or inside rock bodies.

Other theories include extreme magmatic views about the origin of mineral deposits. For instance, many ore deposits have resulted from the injection and rapid freezing of highly concentrated magmatic residues (Spurr 1923). A metallurgical interpretation of the ore deposits was also proposed: during the former molten stage of the Earth, the metallic minerals sank in deep zones due to their specific gravity, and they were later brought to the surface (Brown 1948). According to this model, the upper layers first and the lower layers later moved upward in the form of vapors, from which the metals and minerals were deposited. Simultaneously to this exotic theory, Bateman (1951) suggested that the formation of mineral deposits is complex, and eight diverse processes can account for their formation: magmatic concentration, sublimation, contact metamorphism, hydrothermal action, sedimentation, weathering, metamorphism, and hydrology.

The advent of plate tectonics (see next section) improved considerably the understanding of the lithotectonics of rocks and the ore occurrences. Because mineral deposit systems require a conjunction of processes to produce exceptional metal enrichment over background terrestrial concentrations that result in ore deposits, they can form only under specific conditions in particular tectonic environments. Thus, some mineral deposit types are diagnostic of given tectonic settings and can be used to define these settings in combination with more conventional tectonic and petrogenetic evidence (Groves and Bierlein 2007). Taking in mind this view, a logical first-order grouping of mineral deposit types can be proposed in terms of geodynamic setting, and this is most conveniently seen in the context of plate tectonics. As an example of modern theories on mineral deposit genesis, a classification based on the different geological processes that form mineral deposits can be outlined (Kesler 1994). Thus, ore-forming processes can be surface processes, including weathering, physical sedimentation, chemical sedimentation and organic sedimentation, and subsurface processes, involving water or magmas. This broad expression of ore-forming processes is the most used actually, and it will be explained with more detail in ► Sect. 2.6.

2.4 Mineral Deposits and Plate Tectonics

Plate tectonics is a theory of kinematic character showing that the lithosphere is divided into a finite number of plates that migrate across the surface of the Earth (Box 2.2: Plate Tectonics). It has revolutionized the theories about formation of mineral deposits since plate tectonics determine the origin and distribution of many ore deposits. Thus, plate tectonics plays an essential role in the detection of geological environments with different characteristics. Consequently, the classification of mineral deposits based on plate tectonics is intensively used, particularly when discussing the broad-scale distribution of ore deposits.

Tectonic setting controls factors favorable for the formation of mineral deposits such as the

Box 2.2

Plate Tectonics

The word tectonics derives from the Greek tektonikos, meaning «pertaining to building or construction. » In geology, tectonics concerns the formation and structure of the Earth's crust. From the late 1960s, the proposal of plate tectonics theory, supplanting the geosynclinal concept of lithotectonic associations, clearly caused a revolution in understanding the dynamic interaction of the Earth's crust and mantle as well as geological thinking. In fact, plate tectonics is one of the most important discoveries of the twentieth century. Earlier in this century, geologic paradigm was dominated by the belief that ocean basins and continental land masses were permanent and fixed on the surface of the Earth.

The theory of plate tectonics incorporates the ideas of continental drift and seafloor spreading in a unified model. Wegener (1912) is usually considered the first to have formulated the continental drift theory precisely, and seafloor spreading hypothesis was proposed by Harry H. Hess in 1960. The theory of plate tectonics attributes earthquakes, volcanoes, the mountain-building process, and related geophysical phenomena to movement and interaction of the rigid plates forming the Earth's crust. Thus, plate tectonics provides a unified mechanism explaining aspects such as the distribution of earthquakes and volcanoes, the origin of continents and ocean basins, the distribution of fossil plants and animals, or the genesis and destruction of mountain chains. Two major premises of plate tectonics are: (a) the outermost layer of the Earth, known as the lithosphere, behaves as a strong, rigid substance resting on a weaker region in the mantle known as the asthenosphere; and (b) the lithosphere is broken into numerous segments or plates that are in motion with respect to one another and are continually changing in shape and size.

The Earth is composed of layers of different composition and physical properties, principally the solid central core, the fluid peripheral core, the viscous mantle, and the solid lithosph ere. The lithosphere is comprised of the upper mantle and the crust, the outer shell of the Earth. There are two types of lithosphere: oceanic and continental (Fig. 2.4). The oceanic lithosphere has a 5–8 km-thick oceanic crust (with a basaltic composition), while the continental lithosphere has a 30–40 km-thick granitic-dioritic crust. The lithosphere is fragmented into pieces of variable shape and size, the so-called plates, and the edges of the plates are termed plate boundaries. The Earth has seven major plates (Africa, Antarctica, Australia, Eurasia, North America, South America, and Pacifica) and several minor ones (Adria, Arabia, the Caribbean, Nazca, the Philippines, and others). These plates move independently relative to one another, with a restricted independence from the seven large plates, however. The average rates of motion of the plates, in the past as well as the present, range from less than 1 to more than 15 cm per year.

The motion of lithospheric plates is a considerable consequence of thermally driven mass movements on the Earth. Thus, plates move because of the intense heat in the Earth's core, which causes molten rock in the mantle layer to move. However, the detailed mechanism by which tectonic

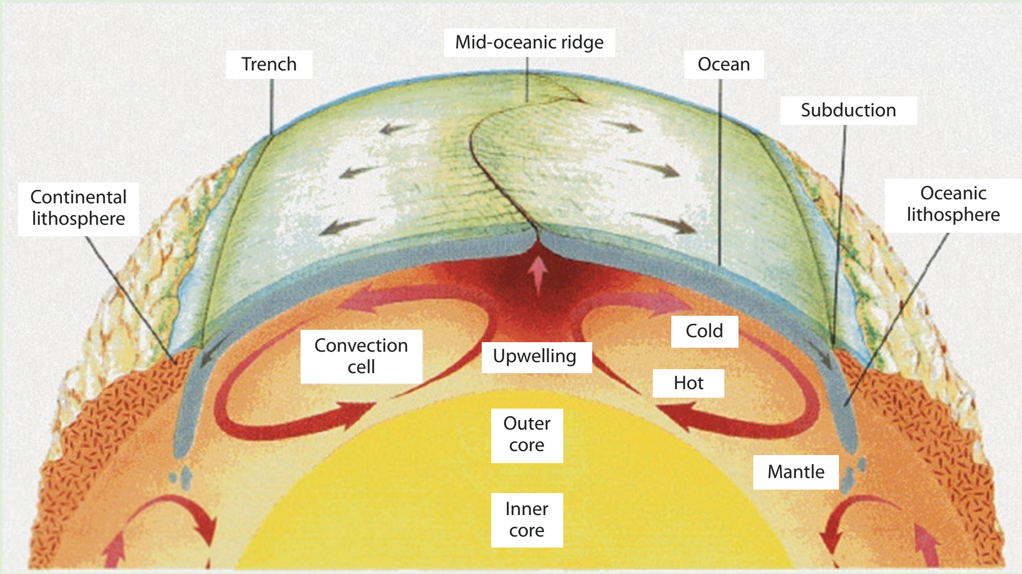

Fig. 2.4 Illustration of plate tectonics theory

plates move is still a subject of much debate among Earth scientists (convection cells vs slab pull). Plate tectonics, the study of such relative motions and their consequences, allows relating surface, geological, and geophysical structures with quantified movements attributed to deep processes of the Earth.

Each lithospheric plate consists of the upper roughly 80–100 km of rigid mantle rock capped by either oceanic or continental crust. Lithosphere capped by oceanic crust is often simply called oceanic lithosphere, and lithosphere capped by continental crust is referred to as continental lithosphere. Some plates, such as the Pacific Plate, consist entirely of oceanic lithosphere, but most plates, such as the South American Plate, consist of variable amounts of both oceanic and continental lithosphere with a transition from one to the other along the margins of continents. The plates move with respect to one another on the ductile asthenosphere below. As the plates move, they interact with one another along their boundaries, producing the majority of Earth's earthquake and volcanic activity. Most plates contain both oceanic and continental crust, and a few contain only oceanic crust. Essentially, the continents are lighter and more buoyant; hence, they float higher on the Earth's mantle than the ocean's crust does.

The three basic kinds of plate boundaries are defined by the type of relative motion between the plates: divergent, convergent, and transform. In a divergent boundary, two plates pull away or separate from each other, producing new crust. Examples are Mid-Atlantic Ridge and East Pacific Rise. In a convergent boundary, two plates move toward or collide with each other, consuming old crust. Examples are India into Asia and NW coast of the USA and SW coast of South America. In the third type of boundary (transform), two plates slide horizontally past each other. In this case, the process does not consume or create crust. Examples are North Anatolian Fault (Turkey), Dead Sea Transform Fault (Israel, Jordan), and San Andreas Fault.

form and composition of igneous bodies, the formation of sedimentary basins and the characteristics of sediments that infill the basins, and the development of faults and shear zones that provide conduits for mineralizing fluids or places for ore location. Thus, it is not surprising that many authors have attempted to relate the distribution of mineral deposits to plate tectonics. Tectonics not only controls the architecture of a basin but also facilitates the interaction between fluid and rock (Kyser 2007).

The study of relationships between mineral deposits and plate tectonics has been particularly successful for many kinds of deposits (e.g., porphyry copper deposits, volcanic-hosted massive sulfide deposits, and much more) (Fig. 2.5), but others (e.g., Precambrian massive sulfide and Ni sulfide deposits) cannot yet be easily

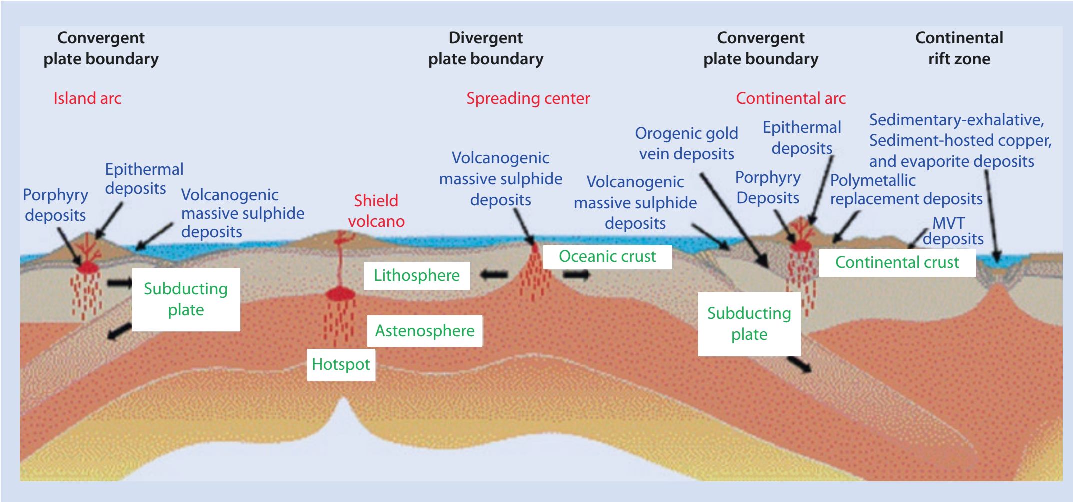

Fig. 2.5 Distribution of mineral deposits in relation to the main types of tectonic plate boundaries

assigned to specific plate tectonic processes. Some plate tectonic settings, especially during the Precambrian, are still highly controversial. It is important to keep in mind the overall influence of plate tectonics in each group of mineral deposits. Since mineral deposits can be commonly separated into those originated by endogenous processes and those formed by surficial ones, Sawkins (1984) proposed that: «the deposits formed by endogenous processes are invariably associated with thermal processes and, in general, can be related more readily to magmatic and tectonic events instigated by plate activity while deposits formed by surficial processes such as weathering or shallow marine sedimentation will show relationships to their tectonic environment that are more tenuous.» Moreover, since most mineral deposits are concentrated by subsurface chemical processes related to magmas and hot waters as well as by near-surface chemical and physical processes, such as erosion and evaporation, these processes are much more common on the continental crust, and their products are better preserved there because the continents are floating on the mantle. In contrast, ocean crust sinks back into the mantle at subduction zones. Thus, the oldest known ocean crust is only about 200 million years, whereas the oldest rocks on the continents are about 4 billion years old (Kesler 1994). Consequently, the continental crust is the archive of Earth history (Cawood et al. 2013).

Initial hypotheses of the relationship between distinct classes of ore deposits and their plate tectonic locations were well established (e.g., Mitchell and Garson 1981; Sawkins 1984). These accounted for the distribution of some ore deposit types in the Phanerozoic, but however there were limitations (Kerrich et al. 2005): (1) at the time, genetic hypotheses for many types of ore deposit were based on syngenesis; (2) where consensus existed on a syngenetic versus epigenetic origin, the age of mineralization was not well constrained; (3) epochs, or secular cycles, of metallogenic provinces were not accounted for; and (4) extrapolation to the Precambrian met with uncertainties as to tectonic processes during that era. Other classifications and descriptions include, for example, a concise list of metallic and nonmetallic resources for each era, including their geodynamic and geological settings (Windley 1995).

During the period of plate tectonics revolution, other discoveries had a major impact on theories of ore genesis such as the observable natural concentration systems, actually active at or near the Earth's surface. For instance, modern seafloor prospection shows the great magnitude of the manganese nodules outlined by the Challenger expedition. It demonstrates not only the enormous potential resource of Cu, Co, Ni, and other associated metals but also the potential of cold seawater as a dilute-mineralizing fluid. The hot brine pools and underlying soft ferruginous muds rich in Zn, Cu, and Ag in the Red Sea deeps

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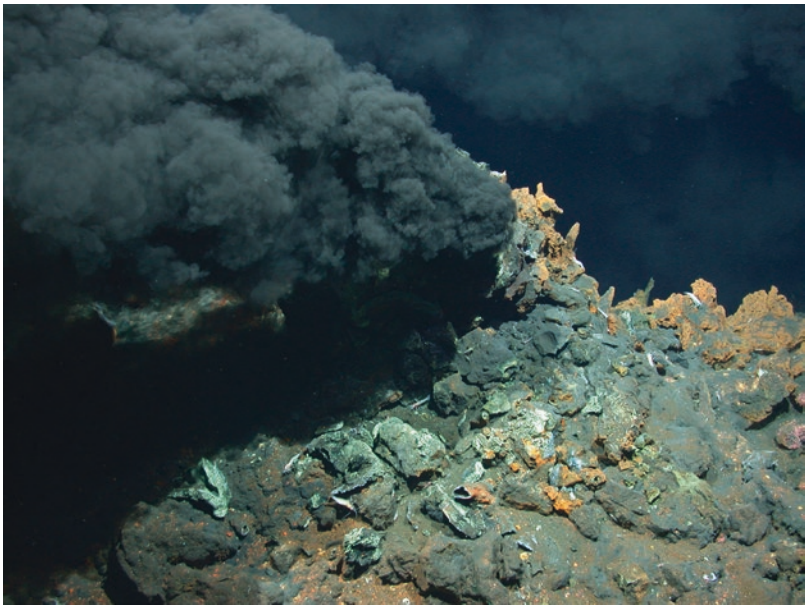


Fig. 2.6 Black smoker (Image courtesy of MARUM, University of Bremen)

are also another example of this type of concentration systems. They were discovered in 1965 and show an exhalative deposit actually forming in a continental rift system. Finally, the discovery of active «black smoker» hydrothermal vents and massive sulfide deposits on the Mid-Atlantic Ridge (Fig. 2.6) made a dramatic impact.

2.5 Criteria for the Classification of Mineral Deposits

Mineral deposits are found in so many different forms, and under so many varying conditions, the attempts of different writers to formulate a classification, founded upon a natural basis, have not been attended with much success (Park 1906). This assertion made more than a century ago is actually untruth since the knowledge of the mineral deposit formation processes obviously has increased dramatically in a century, but it shows how problematic it was to create a simple classification of mineral deposits. In many cases, the difficulty to avoid the dispute between plutonist and neptunist was insurmountable. Prior to the twentieth century, models for the formation of mineral deposits were subject to the often polarized views of either plutonist (all deep igneous origins) or neptunist (all sedimentary origins) theories for the origin of rocks. It was really only in the twentieth century that modern views of mineral deposit formation emerged. Arndt and Ganino (2012) observed that through the twentieth century: «many classifications were based on the types of rocks hosting the ore deposits or on the geometry of the deposit and its relation to the host rocks; thus, deposits in granites were distinguished from those in sedimentary rocks; vein-like deposits were separated from layers conformable with the stratification of the host rock; massive ores were distinguished from disseminated ores, and so on.»

Criteria used to classify mineral deposits vary widely. Since a perfect classification is utopic, a large number of items can be applied. A classification accepted implies that it has been derived by systematic application of certain principles. It must be understandable for the user and must be open so that new mineral deposit types can be added in the future. Geologists usually rank ore deposits according to the (a) commodity, (b) tectonic setting, (c) geological setting, (d) genetic model in the genesis of the mineral deposit, and (e) other aspects (e.g., form of the deposit, temperature of mineral formation, etc.). For instance, Gabelman (1976) shows up with different criteria to classify stratabound ore deposits such as major controlling processes, direct emplacement mechanism, host lithology, chemical reactivity, source of metals and/or transporting fluids, direction of transporting fluids, and relative age of deposit and host.

The genetic classification schemes are the most commonly used since they incorporate elements

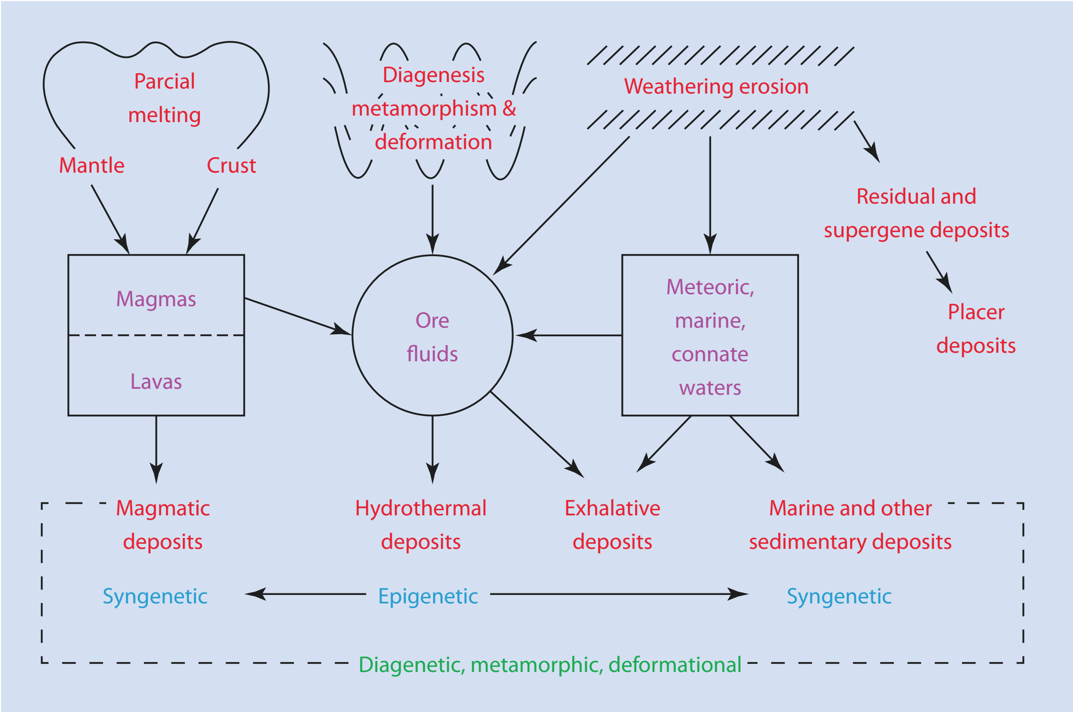


Fig. 2.7 Genetic classification scheme for ore deposits (McQueen 2005)

of composition, form, and association. This type of classifications allows to develop predictive models that can be used to search for geological environments in which appropriate ore-forming processes have possibly operated (McQueen 2005). In this sense, some authors think that classifications by commodity are geologically useless; thus, uranium deposits occur in sandstone and in granites, their formation processes being radically different. However, knowledge of uranium world production, regardless of their genesis, can be essential for other purposes, such as mineral supply, world trade, etc. Other authors underline that genesis is not a good classification criterion because there is considerable controversy among geologists as to the exact mode of formation of many mineral deposits.

A sound alternative is to classify deposits based on empirical features such as type of minerals or host-rock associations, which will lead to the unique fingerprint of a particular deposit (i.e., a descriptive model). Even though no two mineral deposits are identical, empirical descriptions of deposits tend to show natural groupings into a small number of loosely definable categories or types. In turn, these categories tend to coincide with genetically derived models; so even by using purely physically descriptive classifications, there is often a close coincidence between these and models defined using genetic criteria (Herrington 2011).

The classification of mineral deposits based on major Earth process systems is very easy. Rocks are classified universally as igneous, sedimentary, and metamorphic, which express the fundamental processes active in the crust of the Earth. Likewise, since ores are rocks, they can often be associated with each type of rock. Therefore, this character (igneous, sedimentary, or metamorphic) can represent a good basis for classification as it reflects the genetic process involved in ore formation. In this sense, Fig. 2.7 shows a genetic classification for mineral deposits showing the major clusters of ore-forming and modifying processes (McQueen 2005). The classification highlights the categories of ore-forming processes and the subsequent overprinting that can suffer the deposits.

In summary, linking deposit types directly to ore-forming processes and genesis is certainly the preferred way to classify (e.g., Herrington (2011); Table 2.1). It provides better criteria for

Table 2.1 Major classes of economically important mineral deposits (Herrington 2011)

Class	Type/Subtype		
1 Deposits in mafic magmas	1.1 Layered chromite deposits		
	1.2 Podiform chromite deposits.		
	1.3 Titanomagnetite deposits		
	1.4 Magmatic platinum group metal deposits		
	1.5 Nickel sulfide deposits	1.5.1 Sudbury	
		1.5.2 Flood basalt association	
		1.5.3 Ultramafic volcanic association	
		1.5.4 Other mafic and ultramafic intrusive associations	
2 Magmatic diamond deposits	Kimberlites and lamproites		
3 Deposits associated with felsic magmas	3.1 Porphyry Cu-Mo-Au deposits		
	3.2 Porphyry Mo (W) deposits		
	3.3 Granite-hosted Sn-W deposits		
	3.4 Intrusion-related gold deposits		
4 Deposits associated with peralkaline and carbonatite magmas	4.1 Peralkaline Ta-Nb, rate earth element deposits		
	4.2 Carbonatite Cu, rare earth element, Nd, Fe, P deposits		
5 Skarn and carbonate replacement deposits			
6 Iron oxide copper-gold deposits			
7 Hydrothermal gold end silver deposits	7.1 Sediment-hosted gold deposits		
	7.2 Epithermal gold and silver deposits	High-sulfidation epithermal	
		Low-sulfidation epithermal	
	7.3 Lode (or orogenic) gold deposits		
8 Volcanic-hosted or volcanogenic massive sulfide deposits	Mafic		
	Bimodal mafic		
	Pelitic mafic		
	Bimodal felsic		
	Siliciclastic felsic		
9 Sediment-hosted deposits	9.1 Sediment-hosted sulfide deposits	9.1.1 Sedimentary exhalative Pb-Zn (Cu) in clastic sediments (+*Broken-Hill type deposits*)	
		9.1.2 Mississippi Valley type (MVT) Ph-Zn	
		9.1.3 «Irish» type Pb-Zn (Cu)	
		9.1.4 Clastic sediment-hosted Cu	

Table 2.1 (continued)

Class	Type/Subtype		
	9.2 Sediment-hosted iron and manganese deposits	9.2.1 Ironstones	
		9.2.2 Banded iron formation (BIF)	9.2.2.1 Algoma BIF
			9.2.2.2 Superior BIF
			9.2.2.3 Rapitan BIF
		9.2.3 Manganese ore	
	9.3 Sedimentary uranium deposits	9.3.1 Unconformity vein type uranium	
		9.3.2 Sandstone-hosted uranium	
	9.4 Gold and uranium in conglomerates		
	9.5 Chemical sediments	9.5.1 Evaporites	
		9.5.2 Manganese nodules	
10 Ores related to weathering	10.1 Laterites	10.1.1 Bauxite	
		10.1.2 Nickel (cobalt) laterite	
		10.1.3 Lateritic gold	
	10.2 Supergene weathering	10.2.1 Secondary copper	
		10.2.2 Secondary zinc	
11 Placer deposits			

understanding the deposits with respect to associated features such as its association with igneous rock suites, alteration patterns, etc. This will lead to more efficient exploration models for their discovery and evaluation. Nevertheless, descriptive models are needed in practical terms to aid engineers in the evaluation of particular deposits: choice of exploration tool, elements to analyze in geochemical exploration, etc. (Herrington 2011).

2.6 Ore-Forming Processes

The list of captions in ore-forming processes is much larger than the list of geological processes found in any geology text explaining the origin of rocks. Thus, some mineral deposits are formed by magmatic processes, while other mineral deposits are produced by sedimentation or surface weathering. Probably, the main difference between both lists is the secondary importance of metamorphism in the enumeration of substantial ore-forming process compared to its fundamental role in generating rocks. Another major difference is the essential function of hydrothermal fluids (hot aqueous fluids) in the genesis of ore deposits. The circulation of this kind of fluids in the crust is usually cited as a factor that modifies locally the composition and texture of previous rocks. Ore-forming processes can be classified into four main categories (Evans 1993): internal, hydrothermal, metamorphic, and surficial processes. The former three processes are related to subsurface phenomena, while the last one covers those processes occurring at the Earth's surface. Hydrothermal should be further subdivided into magmatic, metamorphic, diagenetic, and surface to refine the nature of the hydrothermal process. Therefore, the first approach to ore-forming processes can be outlined according to the next four types described below: magmatic, metamorphic, sedimentary, and hydrothermal processes.

Whatever the ore-forming process, because of chemical and geological factors, some minerals/metals tend to occur together in mineral deposits, while others may be found associated with a particular rock type. Examples of the former are galena with sphalerite, copper sulfides with molybdenite, gold with arsenopyrite or pyrite, and silver with galena. Regarding the association of mineralization/host rock, examples are lead-zinc in carbonates, copper or copper-lead-zinc with volcanic rocks, tin and tungsten with granite intrusions, chromite in large ultramafic intrusions, and uranium in sandstone and shales.

2.6.1 Magmatic Processes

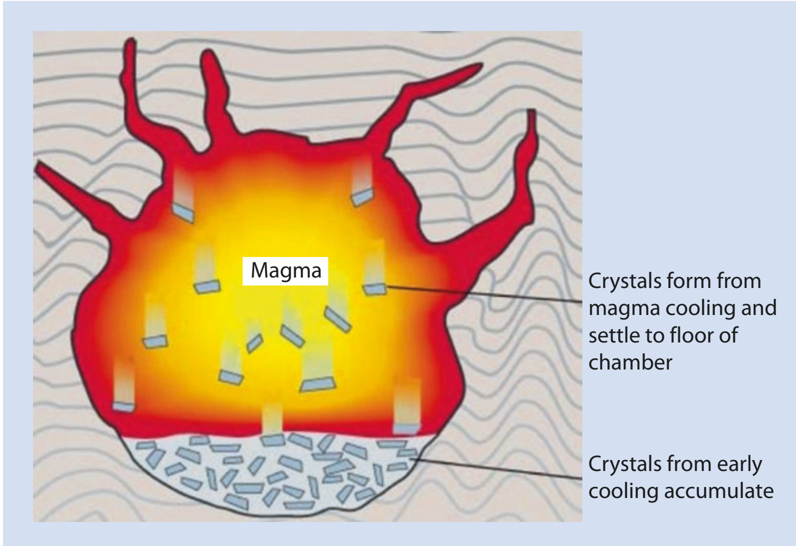
In a broad sense, ore-forming processes related to the evolution of magmas emplaced at crustal levels span a continuum. The two end members of this continuum are (a) orthomagmatic processes, concentration of mineralization as a direct result of magmatic crystallization dominated by silicate melt-crystal equilibria, and (b) (magmatic) hydrothermal processes, concentration of ore minerals from magmatic hydrothermal fluids by crystallization dominated by crystal-volatile equilibria (Misra 2000). The second possibility is considered here as totally controlled by the action of hydrothermal fluids, and, accordingly, it will be included in the group of hydrothermal processes. A large and diverse group of ore deposits originates by various processes during the formation, evolution, emplacement, and crystallization of silicate melts (magmas) in the upper mantle and in the Earth's crust. Magmatic deposits may form as a result of (1) solid phases crystallizing as a differentiate as the magma cools, (2) minerals crystallizing from the enriched residual fluids formed as magma cools and crystallizes, (3) the formation of a sulfide melt that developed by immiscibility from a coexisting silicate melt, or (4) where a magma transports xenolithic or xenocrystic phases that it has picked up on its passage through the Earth's crust (Herrington 2011).

The processes of magmatic ore formation are related to intrinsic properties of the magmas and are linked genetically to its cooling and solidification pattern. It is recognized that different mineral deposits are hosted in igneous rocks, and these deposits display different metal associations. This must be associated somehow to the environment in which magmas are originated and the compositional characteristics generated from specific settings. In this sense, it is broadly recognized that most of the chalcophile and siderophile elements (e.g., Ni, Co, Pt, Au) more likely to be linked with mafic rocks, while concentrations of most lithophile elements (e.g., Sn, U, and W) are classically located in association with felsic or alkaline rocks (Robb 2005). Essentially, this distribution was understood because of the geochemical fate of different metals during fractional crystallization (solid-liquid fractionation) of silicate melt bodies (Pohl 2011).

Where the magma enters the crust and crystallization starts, an immiscible sulfide liquid will divide from the silicate liquid if the concentration of sulfur exceeds the solubility. Experimental studies have shown that the solubility of sulfide depends on external parameters, such as temperature and pressure, and on the composition of the melt. During fractional crystallization of magma, the temperature drops, Fe content slightly changes, and Si content increases, which lead sometimes to sulfide saturation and the separation of sulfide liquid. Many parameters influence these processes, including depth of intrusion, tectonic activity, temperature gradient in space and time, fractional crystallization, dynamics of the melt body, repeated injection of fresh melt, assimilation of country rocks, sulfur or external fluids, liquid immiscibility of ore and silicate melts, and mixing or redissolution (Kerr and Leitch 2005).

Another mechanism to explain the formation of magmatic mineral deposits is the so-called fractional crystallization (◘ Fig. 2.8). In this model, dense minerals form a cooling magma chamber and settle to the bottom producing a sequence of layered rocks. The remaining liquid magma becomes saturated with sulfur, and sulfide minerals rich in some metals crystallize out of the magma and settle to the bottom. For instance, these layered rocks formed by sulfides host PGE deposits.

Fig. 2.8 Illustration of fractional crystallization



2.6.2 Metamorphic Processes

Ore deposits in metamorphosed rocks can be formed before, during, or after metamorphic processes. The first category is of premetamorphic origin independent from later metamorphic overprinting, and it is the class of metamorphosed ore deposits (Pohl 2011). Metamorphic deposits owe their origin to contact or regional metamorphism and involve recrystallization, commonly accompanied by mobilization of disseminated ore constituents by metamorphic fluids (Misra 2000). Metamorphic rocks host many ore deposits, and metamorphic fluids are thought to be a source for various mineral deposits. Thus, this type of fluids usually carries important metal content, although for chloride-complexed metals, maximum concentrations are commonly lower for magmatic fluids. For instance, gold ore is the type of mineralization usually linked to metamorphic fluids. Therefore, based on chemistry, it is possible to argue that in some circumstances, metamorphic fluids can contain high concentrations of metals and may therefore be potential ore fluids (Banks et al. 1994). According to Yardley and Cleverley (2014), there are three situations in which ore deposits are formed from metamorphic fluid processes: (a) where relatively metal-rich metamorphic fluids provide a medium for segregation, (b) where decarbonation reactions lead to focused fluid flow and skarn formation, and (c) where rapid uplift drives dehydration reactions despite falling temperature, so that the rate of fluid production is not limited by heat flow.

Since magmatic activity is common in certain metamorphic settings, it is reasonable to consider that some mineral deposits in metamorphic rocks were formed by combined metamorphic and magmatic processes. Skarn and contact metamorphism ore deposits are intimately related to thermal aureoles of magmatic intrusions. They can be envisaged as products of contact metamorphism, but the causal agent is the interaction with magmatic fluids and not simply change by heating (Pohl 2011). Because of the complications of describing skarns based on alteration minerals, which are a combined function of wall-rock chemistry and the superimposed system, mineralized skarns are best classified in terms of component of interest. Seven major types are recognized: iron, gold, tungsten, copper, zinc, molybdenum, and tin (Herrington 2011). The different metals found in skarn deposits are a product of the differing compositions, oxidation state, and metallogenic affinities of the igneous intrusion. For instance, Fe and Au skarn deposits are usually associated with intrusions of more mafic to intermediate compositions. Most of the large

Fig. 2.9 Skarn tungsten mine at Los Santos (Salamanca, Spain) (Image courtesy of Daytal Resources Spain, S.L.)

and economically viable skarn deposits are associated with calcic exoskarns, a limestone (calcic) being the host rock and the metasomatic assemblage external to the intruding pluton (exo – prefix). Thus, tungsten skarns produce the bulk of the world production of tungsten (Fig. 2.9) and are typically associated with calco-alkaline intrusions emplaced relatively deep in the crust.

2.6.3 Sedimentary Processes

Low-temperature surface processes can be responsible for the formation of economic ore deposits at or very near the Earth's surface. Under favorable conditions, sediments and sedimentary rocks become selectively enriched in some elements of potential economic value. Two main types of sedimentary processes can be outlined: sedimentation and weathering. Sedimentation may lead to the formation of mineral deposits through clastic accumulation (e.g., gold or diamond placer deposits) and chemical and/or biochemical precipitation of economically important constituents in lakes, coastal settings, or shallow to deep oceans, including evaporation processes. In clastic accumulation, physical processes such as physical erosion, transportation, and deposition lead directly to the redistribution and accumulation of specific minerals. Thus, these deposits are formed as a result of the differing physical and chemical behavior of the minerals forming the original rock, either hydraulic (water) or Aeolian (wind) being the physical processes. Examples of these deposits are the already mentioned diamond placer deposits (Fig. 2.10) in river sediments and deposits of heavy minerals in beach sands.

Regarding chemical and/or biochemical precipitation, metals and other valuable minerals are soluble in surface waters. They precipitate where they meet saturation levels (evaporation) or where the composition or physical conditions on the water shift. Examples of the latter are sediments enriched in iron or manganese resulting from mixing of waters with different composition or redox states. Evaporation is a surface phenomenon where dissolved salts precipitate as water is lost in an evaporating basin or by the evaporation of water from the ground's surface due to heat energy from the sun. Sedimentation is limited to the surface of the Earth, which is also the realm of life and its biochemical cycles; therefore, sedimentary ore formation will almost always show biogenic components (Southam and Saunders 2005). It is very obvious in phosphate deposits made of bones and coprolites and in lignite seams composed

Fig. 2.10 Diamond placer deposit in river gravels (South Africa) (Image courtesy of Rockwell Diamonds Inc.)

of fallen trees. Bacteria can enhance dissolution of rocks and minerals containing metals, aid in metal transport, affect porosity and permeability of rocks, and cause the precipitation of biogenic sulfur, sulfides, and carbonates. In particular, iron-reducing bacteria and sulfate-reducing bacteria may play important roles in low-temperature ore genesis. Thus, iron-reducing bacteria can cause reductive dissolution of Fe oxyhydroxides, such that it occurs in red beds, causing adsorbed and coprecipitated metals to be released to solution. Organic compounds produced by bacterial degradation of a more complex organic matter could enhance metal transport by formation of metal-organic complexes. Similarly, biogenic H_2S could form stable aqueous metal-sulfide complexes leading to transport of certain metals such as Ag at low temperature (Kyle and Saunders 1996).

Weathering may also lead to residual concentration of weathering-resistant minerals of the parent rock or of relatively insoluble elements reconstituted into stable minerals (Misra 2000). In this regard, weathering is a very important ore-forming process resulting in chemical change and redistribution of components in surface rocks by migrating solutions. The differential chemical properties of minerals at the Earth's surface and in the surface-crustal interface can lead to residual upgrades or chemical dissolution and reprecipitation mechanisms to concentrate the metal/mineral of interest. Under these conditions, ore formation is driven by the circulation of largely meteorically derived water at the Earth's surface, although similar analogous processes can take place on the seafloor. These subsurface waters can dissolve and reprecipitate components at favorable mineral sites or surface interfaces (Herrington 2011).

Supergene processes usually originate different types of raw materials such as iron, manganese, or aluminum ores. In supergene process, two basically different process types may lead to concentration: (1) the valued component is enriched in a residuum, while much of the rock mass is dissolved and carried away; an example are laterite deposits, in which iron or aluminum is enriched in the clayey-sandy soils of the tropics and subtropics; and (2) the valued component is dissolved, transported, and concentrated on reprecipitation; in this case, the transport distance is commonly very short, meters to ten of meters (Pohl 2011). A special case of weathering would be the so-called supergene enrichment process, which involves the leaching of ore-forming

Fig. 2.11 Gossan at VMS deposit (Fiji) (Image courtesy of Geonomics)

elements (e.g., copper) from surficial parts of a low-grade sulfide deposit and reprecipitation below the water table. The process involves the release of ore metals from unstable sulfide minerals to downward percolating meteoric water and precipitation of more stable secondary oxide and sulfide mineral assemblages in the subsurface environment. These deposits are usually called «gossan» (Fig. 2.11). In the nineteenth and twentieth centuries, gossans were important guides used by prospectors in their quest for buried ore deposits.

2.6.4 Hydrothermal Processes

A big problem dealing with the word hydrothermal is its meaning. Hydrothermal means hot water, which is an extremely lax sense of the word because hot water can range from 70 to 200 °C or even 400 °C. The former temperature can be attained in the sedimentary realm, during diagenesis, and the others are characteristic temperatures of endogenous conditions. Hydrothermal fluids generally travel along temperature or pressure gradients, from hot areas to cool areas or from high pressure to low pressure. They migrate until they reach a suitable site for metal deposition. For this deposition, the following is necessary: a rapid decrease in temperature such as where hot fluids exit at the seafloor, a rapid decrease in pressure such as where fluids enter a fault cavity, and/or a change in the chemical composition of the fluid such as where fluids react with a rock (Stevens 2010).

Hydrothermal processes can develop in almost all geological environments. The application of new technologies in geosciences in the last 50 years (e.g., fluid inclusions, trace element analysis, isotope geochemistry, among many others) has changed many of the geological concepts, including metallogenic thinking. For instance, expelled fluids in sedimentary basins during diagenesis can produce numerous metallic concentrations, excluding the participation of endogenous processes. In the past decades, many efforts are carried out toward a better understanding of the complexity of hydrothermal processes.

Although there are several natural processes that concentrate elements within the Earth's crust and form mineral deposits, the most important of which is the hydrothermal process. Hydrothermal ore-forming processes are ubiquitous, and many mineral deposits on Earth have been originated straightly from hot aqueous solutions flowing through the crust. Direct evidence for the presence of hydrothermal fluids in the Earth's crust is surface manifestations such as hot springs and fumaroles. In this sense: «the concept of hydrothermal mineralization can be extended to deposits related to fluids derived from sources other than magmatic solutions; such fluids include those formed from metamorphic dehydration reactions, from the expulsion of pore fluids during compaction of sediment (the release of

Fig. 2.12 Yellowstone (USA) hot springs

trapped water from sedimentary basins undergoing diagenetic change), and from meteoric waters; it also considers seawater as a hydrothermal fluid with specific reference to the formation of base metal deposits on the ocean floor» (Robb 2005).

Magmatic hydrothermal fluids form as a body of magma cools and then crystallizes. In some circumstances, the magmatic system can be a passive source of heat that drives the circulation of fluids exotic to the magma through adjacent fractured crust into which the magma is intruding. In other situations, the magmas, particularly felsic magmas that form granitic rocks, include very significant amounts of miscible water, which is carried in the magma itself. As the magma cools and crystallizes, it becomes more concentrated and eventually forms an immiscible fluid phase, which in the process collects other components that prefer to partition from a silicate melt into a hydrous fluid phase. Williams-Jones et al. (2002) suggest that these metal-rich fluid phases can then migrate away from the magma and interact with minerals and fluids in previously crystallized magma or outside rocks, which cause these to become altered by chemical reaction and lead to precipitation of new mineral phases, including the ore minerals.

Surface or seafloor hydrothermal fluids are generated as deeply penetrating meteoric- or seawater-derived waters descend and become heated deeper in the crust. This process is particularly apparent in regions where there is elevated crustal heat flow, often where the Earth's crust is being thinned. In the case of seafloor, this phenomenon is common where a new ocean is formed by the seafloor spreading through the formation of submarine volcanoes. On land, such hydrothermal fluids can be generated in zones of crustal attenuation, often associated with subaerial volcanism. Surface manifestations of this process are the presence of hot springs on land (Fig. 2.12) or seafloor hydrothermal vents.

The various stages of diagenesis that result in the transformation from uncompacted particles of sediment to lithified sedimentary rock produce aqueous solutions that evolve with time and depth; such type of fluids are often involved in the formation of ore deposits (Robb 2005). This process may develop on a large scale in a sedimentary basin undergoing burial and lithification and is a related process to hydrocarbon generation. The released water can pick up dissolved salts (becoming a brine; Table 2.2), which then has a greater ability to transport many cations and ligands to a point of deposition to form an ore deposit (Brimhall and Crerar 1987). In sedimentary basins, evaporite beds may be a specific source of salts that can be dissolved by the basinal water. Basins undergoing diagenesis become heated, and thus the basinal brine may be a highly effective solvent for dissolving large quantities of metals. These basinal brines can then migrate via crustal faults and permeable horizons to depositional environments.

Diagenetic process evolves to metamorphism as rocks are gradually buried and temperatures

2

Table 2.2 Terms for water with different salinities (Davis and DeWiest 1966)

Term	Concentration of total dissolved solids (TDS) in ppm (parts per million) and weight percent	
Fresh water	0–1000	<0.1%
Brackish water	1000–10,000	<1%
Seawater	31,000–38,000	3.1–3.8%
Saline, or salty water	10,000–100,000	<10%
Brine	>100,000	>10%

overcome approximately 200 °C. Thus, metamorphic-hydrothermal fluids form as metamorphism results in mineral-chemical processes that may release volatiles, often dominated by water but which may include gases such as CO_2. Metamorphism is induced in rocks by external heat or pressure or by a combination of both. Heat may be provided by the deep burial of a rock mass through time or alternatively by the intrusion of a magma body nearby. Pressure to cause metamorphism may be provided again during deep burial or else by tectonic processes.

2.7 Mineral Resources Commodities

Mineral deposits can be classified according to the valuable raw material being extracted. This classification finds some application in a purely economic context and gives raise to three main groups: (a) energy commodities, this group is formed by petroleum, natural gas, tar sands, bituminous shales, coal, and uranium; (b) metallic commodities, a very large group that includes many metal types, related to their uses, density, monetary value, etc.; and (c) nonmetallic commodities. In turn, the latter can be subdivided into two essential categories: industrial minerals and industrial rocks. The minerals used by their specific chemical and physical properties (e.g., sodium sulfate utilized as laundry detergent) fit in the first category; the second one, developed after the Second World War and of growing commercial interest, includes a wide variety of raw materials, mainly rocks, that are used preferably in construction (e.g., buildings, roads, or bridges). Typical examples are natural aggregates and building stone.

2.7.1 Energy

Energy commodities include mainly fossil energy raw materials and uranium to produce nuclear energy. Coal was the first fossil energy raw material used by man at the beginning of the Industrial Revolution, and this predominant role spanned until early in the twentieth century. Since then, oil has displaced coal to a second rank (here the terms oil and petroleum are used interchangeably, although some differences exist). In 2014, the world's primary energy supply was provided by 32.9% from oil, 23.7% natural gas, 23.8% coal, 6.8% hydroelectricity, 4.4% nuclear power, and 2.8 renewables (BP Statistical Review of World Energy 2016). Altogether they form the so-called primary energy. In the future, the Energy Outlook 2035 establishes that economic expansion in Asia will produce a continued growth in the world's demand for energy, rising by 37% from 2013 to 2035 or by an average of 1.4% a year.

Petroleum

Petroleum (Fig. 2.13) is derived from ancient fossilized organic materials such as zooplankton and algae (Box 2.3: Petroleum Formation). It is formed by hydrocarbons with the addition of some other substances. Thus, the main hydrocarbons commonly present in petroleum are the following: paraffins (15–60%), naphthenes (30–60%), and aromatics (3–30%), with asphaltics making up the remainder. The percentages for these hydrocarbons can vary greatly, depending upon the geographic region. Regarding the chemical composition, the basic components are carbon (93–97%), hydrogen (10–14%), nitrogen (up to 2%), oxygen (up to 1.5%), and sulfur (0.5–6%), with a few trace metals making up a very small percentage of the petroleum composition. The properties of each different petroleum source are defined by the percentage of the four main hydrocarbons found within petroleum as part of the petroleum composition.

Petroleum is generally measured in volume (a barrel is equal to 159 liters). The petroleum industry classifies the different oil types by the location where the petroleum is produced (West Texas

Fig. 2.13 Petroleum platform (Image courtesy of Pedro Cámara)

Box 2.3

Petroleum Formation

Petroleum (also known as crude oil or simply oil) is a fossil fuel that was formed from the remains of ancient marine organisms. Coal, natural gas, and petroleum are all fossil fuels that formed under similar conditions. In fact, petroleum is frequently found in reservoirs along with natural gas. In the past, natural gas was either burned or allowed to escape into the atmosphere. Now, technology has been developed to capture the natural gas and either reinject it into the well or compress it into liquefied natural gas (LNG), which is easily transportable and has versatile uses.

Formation of naturally occurring raw petroleum takes millions of years. Large amount of the organisms sourcing the petroleum remains settled to sea or lake bottom, mixed with sediments and buried under anoxic conditions. As the microscopic algae and phytoplankton died, they sank to the bottom and accumulated in large quantities in the oxygen-free sediments. Over time, they were buried deeper and subjected to a long process of chemical conversion by bacterial decomposition followed by the effects of high temperatures. This caused the formation of liquid and gaseous hydrocarbons in the source rock (hydrocarbons are simply chemicals made up of hydrogen and carbon). Petroleum source beds are fine-grained, clay-rich siliciclastic rocks (mudstones, shales) or dark-colored carbonate rocks (limestones, marlstones), which have generated and effectively expelled hydrocarbons. Most of the economically useful petroleum deposits were deposited during the Phanerozoic. This is thought to reflect the lower rate of organic carbon production and burial in the earlier eons.

Increasing heat and pressure cause the organic matter to change, first into kerogen, one of the products of anaerobic decomposition of organic matter (it is found in various oil shales around the world) and then into liquid and gaseous hydrocarbons in a process called catagenesis. Thus, catagenesis comprises all processes that act on rock matrix and organic matter after considerable burial and that result in petroleum generation; higher pressure and temperature are essential factors of change. The main result of catagenesis is the generation of oil and wet gas while kerogen «matures. » At about 60 °C, oil begins to form in the source rock due to the thermogenic breakdown (cracking) of organic matter (kerogen). There is a temperature range in which oil forms. It is called the «oil window» (often found in

2

the 60–120°C interval – approx. 2–4 km in depth). Below the minimum temperature, oil remains trapped in the form of kerogen, while above the maximum temperature is converted to natural gas through thermal cracking (about 160°C). The gas produced in this way is often separated from the petroleum. If temperature reaches high value (>250°C), the original biomass will be destroyed and no gas or petroleum is formed. Typically lower temperatures during petroleum formation will result in thicker, darker raw petroleum deposits, the most solid of which being a bitumen substance.

After expulsion from the source rock (Fig. 2.14), both oil and gas, lighter than water, migrate upward through permeable rocks (e.g., sandstones) or fractures until they are stopped by a non-permeable layer of rock (e.g., shale). The production of petroleum increases pressure within the rock because oils and gases are less dense than solids and, hence, take up more volume. The overpressure fractures the source bed, enabling migration of the gas and oil into adjacent permeable rocks. Migration occurs vertically and laterally through the fractures and faults until an impermeable barrier is reached. Oil and gas migration takes thousands or millions of years and may extend over tens of kilometers. Gravity forces the oil to move out of the source rock and upward toward the surface, looking for a reservoir. Reservoir is a rock that has the ability to store fluid such as sandstone where oil or gas can be between grains of sandstone. Porous limestone is also a good reservoir rock since many cavities can be connected with each other. Thus, reservoir rocks are porous and always saturated with water, oil, and gas in various combinations. Petroleum reservoirs can be found beneath the land or the ocean floor.

In addition, impermeable rock has to be present to stop petroleum escaping from reservoir rock. Impermeable rock that forms a seal over reservoir rocks is called cap rock. Cap rocks of most petroleum fields are fine-grained, clay-rich sediments like shales or mudstones. Due to their low permeabilities and very small-diameter pores, capillary entry pressures are so high that they cannot be overcome by the buoyancy of a high oil or gas column. The most ideal and best sealing cap rocks are, however, evaporite strata like anhydrite or rock salt. Such good-quality cap rocks hold many of the large petroleum accumulations in the Middle East in place.

If there is a suitable combination of source rock, reservoir rock, and cap rock and a trap in an area, recoverable oil and gas deposits may be discovered there. If there is no cap rock, the oil and gas will slowly continue to migrate toward the surface. In certain geological locations, as the oil migrated and came closer to the Earth's surface, microorganisms slowly consumed the hydrocarbons, beginning with the lightest. The heavy oil and bitumen now being produced are the remnants of that migration. Heavy oil deposits (e.g., tar sands) are the world's largest known liquid hydrocarbon resources and comprise about 65% of all the liquid petroleum in the world. Very large deposits of tar sands occur in northern Canada (Athabasca tar sands) and eastern Venezuela.

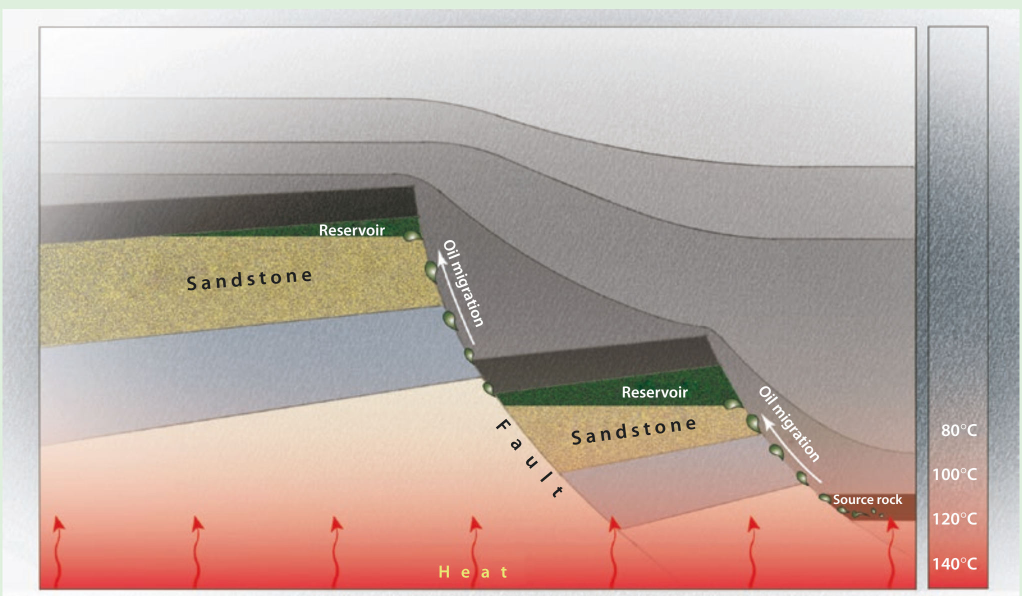


Fig. 2.14 Formation of petroleum reservoirs (Illustration courtesy of The Norwegian Petroleum Directorate)

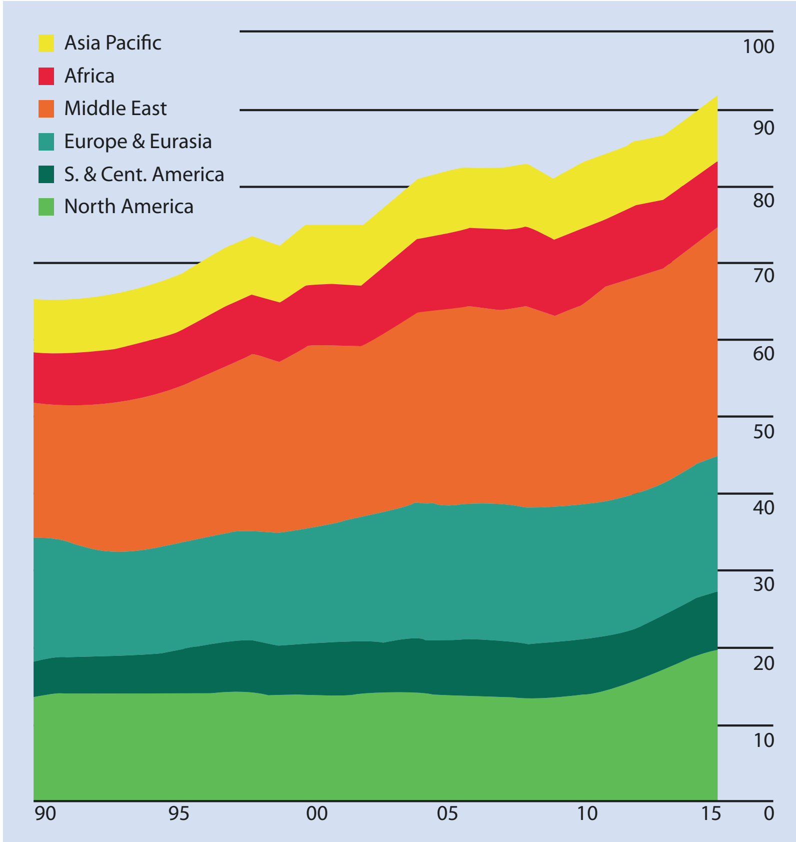


Fig. 2.15 Oil production by region in million barrels daily (BP Statistical Review of World Energy 2016)

Intermediate or Brent), the density in API gravity (if the API gravity of a crude oil is greater than 10, it floats on water; if less than 10, it sinks), and its sulfur content. The major oil-producing regions around the world are located in Kuwait and Saudi Arabia, although other countries in the Middle East region also make up a significant part of world production (e.g., Iran and Iraq) (Fig. 2.15). The North Sea crude oil fields are the second most influential oil field in economic terms.

The main application of petroleum is for power combustion engines (gasoline or petrol). Other applications include manufacturing of plastics and synthetics, paving road, and roofing. In general, a lighter raw petroleum composition is more useful as a fuel source, while denser petroleum composition is more suitable for plastic manufacturing and other uses. Broadly, a barrel of crude oil produce the following components: 43% gasoline, 21% diesel, 10% jet fuel, 4% fuel oil, 4% liquefied petroleum gases, and 18% other products.

Natural Gas

Natural gas plays a vital role in the world's supply of energy. In its pure form, natural gas is colorless, shapeless, and odorless. Compared to other fossil fuels, natural gas is cleaner and emits lower values of harmful components to the air. Although it is a blend of different hydrocarbon gases, natural gas is formed mainly of methane (70–90%), ethane, propane, butane (all three together up to 20%), and other components (e.g., pentane, carbon dioxide, oxygen, nitrogen, or hydrogen sulfide). Temperature and pressure determine the composition of the gas phase because several higher hydrocarbons are gaseous in the reservoir but condense if the pressure is lowered. Natural gas is called «dry» if it is almost pure methane and «wet» if other hydrocarbons are present. The «dryness» of gas can be characterized by the percent methane/percent ethane ratio. Other denominations are «sour» gas if it has elevated fractions of sulfur and CO_2 and «sweet» gas if it contains less than

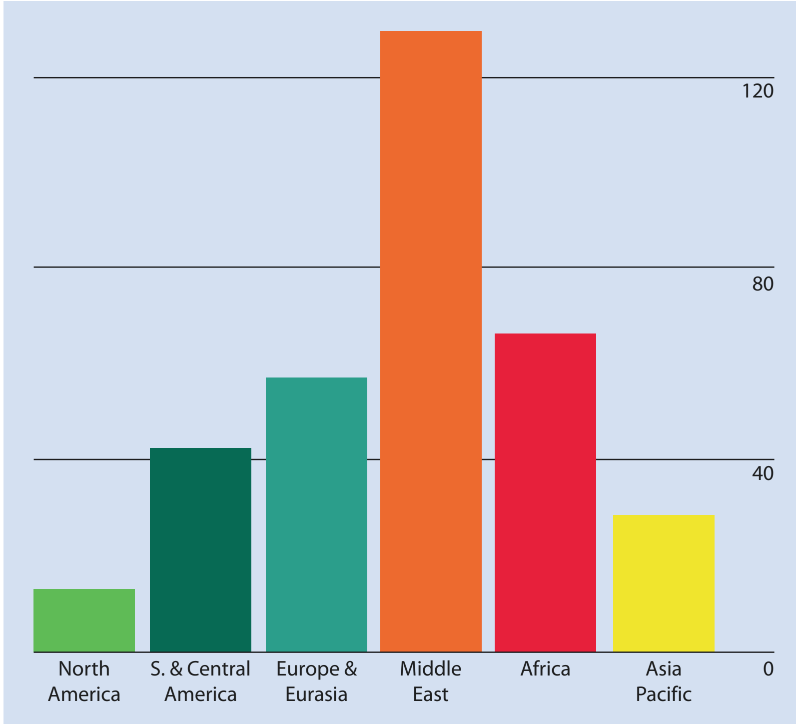


Fig. 2.16 Natural gas reserves-to-production (R/P) ratios in 2015 by region (BP Statistical Review of World Energy 2016)

2% of CO_2 and no H_2S. Only «sweet» gas can be directly used, and the rest must be first refined.

Natural gas is found in reservoirs often associated with oil deposits. Since gas is dissolved in oil, a free gas cap forms on top of the oil pool where saturation is reached. When natural gas is formed, it rises toward the surface because it has a low density. Some of this methane will dissipate into the air, but it will also rise up into geological formations that trap the gas under the ground. These formations are mainly composed of layers of porous sedimentary rocks with an impermeable layer of sediment on top to prevent the migration of the natural gas until the surface. The obtained natural gas is then refined to remove impurities (e.g., water, other gases, or sand). After refining, the natural gas is usually transmitted through a network of pipelines to its point of use. Natural gas can be measured in cubic feet or, like other forms of energy, in British thermal units (Btu). The definition of a Btu is the following: 1 Btu is the quantity of natural gas that will generate sufficient energy to heat 1 pound of water by 1 degree at normal pressure. Regarding the production and reserves of natural gas, Fig. 2.16 shows the reserves-to-production ratios in 2015 by region.

Fig. 2.17 Tar sands sample

Tar Sands

Tar sands (Fig. 2.17), sometimes referred to as oil sands, are a combination of bitumen, water, clay, and sand, the bitumen being a heavy black viscous oil. Tar is a term for heavy and extra-heavy oils (6–12°API) that are highly viscous and sulfur-rich. It is the residuum of a degradation or normal petroleum; degradation is essentially the loss of light hydrocarbons and an increase of N-S-O compounds. Deposits of tar sands may be mined to obtain the oil-rich bitumen, which is later refined to produce oil. Because the bitumen in tar sands cannot be pumped in its natural state, tar sand deposits are commonly mined using open-pit mining. In other cases, the oil is extracted by underground heating with additional upgrading. This process involves injecting steam into the ground to melt the bitumen from the sands and pumping the bitumen up to the surface.

During many decades, the oil industry clearly ignored tar sands oil since the exploitation of this energy source is much more expensive, difficult, and, more important from an environmental view point, dirty than conventional oil. Theoretically, much of the world's oil reserves (e.g., 2 trillion barrels) are in tar sands form, although obviously it is not all mineable. The largest deposits in the world of tar sands are found in Canada (Athabasca deposit) and Venezuela, although various countries in the Middle East and Russia have also important reserves. In this sense, only Canada has a large-scale commercial tar sands industry. Exploitation of tar sands produces actually a strong dispute in Canada, essentially for the environmental impacts of this kind of mining.

Bituminous Shales

Oil shale or bituminous shale is a sedimentary rock that contains up to 50% of organic matter. In fact, it represents certainly an old petroleum parent rock. Once extracted from the ground, the rock can either be used directly as fuel for a power plant or be processed to produce shale oil and other chemicals and materials. With a few exceptions (e.g., fracking), these deposits are yet little exploited. Because environmental considerations and other factors make extraction of these raw materials relatively unattractive, the strategical character of oil shales as a resource of oil and gas depends on a number of criteria such as the ultimate destination of the raw material, the basic cost of extraction and processing, and the environmental costs, among many others. The heating value of bituminous shale is low and similar, for example, to that of brown coal or average forest residues and less than half of that of the average bituminous coal. This is drastically changing with the introduction of fracking or hydraulic fracture techniques (Box 2.4: Hydraulic Fracking).

Box 2.4

Hydraulic Fracturing

Natural gas produced from shale is often referred to as «unconventional gas» by contrast to «conventional gas» produced from other kinds of rock usually sandstones or limestones. Conventional gas is found in reservoirs in sandstone or limestone where gas has migrated up from source rocks. In these rocks, organic matter becomes gas or oil through the action of heat and pressure over time. According to the International Energy Agency (IEA), the volume of unconventional gas resources (including shale gas, tight gas, and coalbed methane) is currently estimated at 340 trillion cubic meters, equivalent to about 40% of global gas resources. In this statistics, shale gas accounts for the biggest share of these resources. Thus, the emergence of shale gas and shale oil has quickly changed the landscape of opportunities for energy provision and security in different regions of the world.

As the reserves of conventional natural gas and oil falling inexorably and could be nearly exhausted, the extraction of unconventional oil and gas trapped in shale appears to be an attractive alternative for several countries, especially the USA. Because shale is a fine-grained, sedimentary rock, the gas and oil it contains do not easily flow and therefore must be released before it can be pumped from the ground. The technique used to extract shale gas is called hydraulic fracturing or colloquially «fracking.» It consists of injecting water, proppant (e.g., granules of sand), and chemicals at high pressure into a shale or sandstone formation. The buildup in pressure causes the formation to fracture, and the proppant fills the fractures to keep them from resealing. This allows the natural gas impounded in the formation to rush into the well for extraction. A combination of factors, including technological advance, desire to decrease dependence from foreign energy, new geopolitical realities, and high oil prices, have made unconventional gas and subsequently hydraulic fracturing particularly attractive.

Hydraulic fracturing is most often performed in horizontally drilled wells (Fig. 2.18). A typical horizontal well has an average lateral extension of 1400 m (maximum of 3000 m). After a period of vertical drilling in order to reach shale deposits (most of unconventional gas is trapped deep inside of shale formations at depths between 1500 and 3000 m), a lateral extension of up to 2000 m is drilled parallel to the rock layer containing the shale. In the next step, fracking fluids are injected into the recently bored hole in order to release the hydrocarbons that are trapped; the fluid is injected under high pressure with the intent of fracturing the soft shale. The rock is hydraulically fractured multiple

2

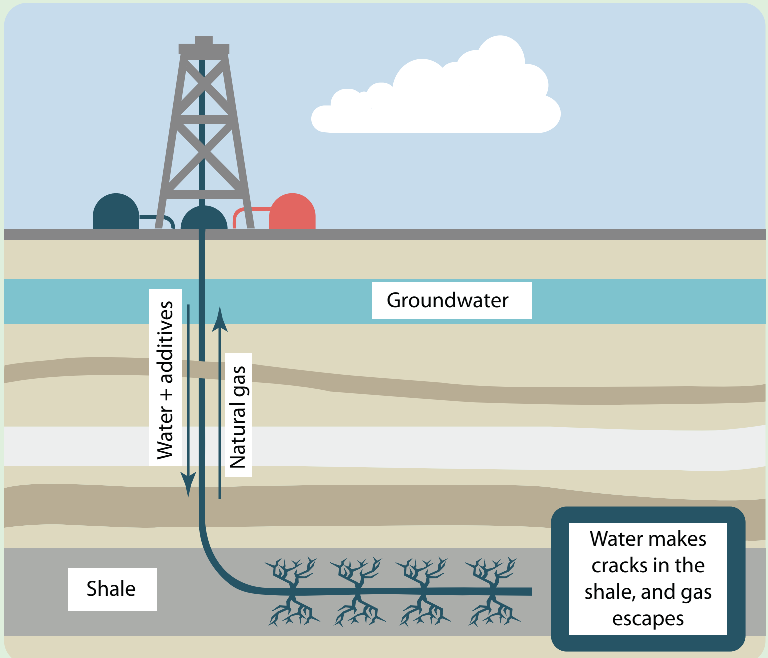


Fig. 2.18 Hydraulic fracking (Illustration courtesy of National Aeronautics and Space Administration)

times every 100 m along this horizontal extent. Occasionally, other substances such as gels, foams, compressed gases, and even air are injected. Chemical mixtures are usually included in the injection, and their objectives are to increase the permeability of the rock by dissolving various components.

Regarding the fracking fluid, it can be injected at various pressures and reach up to 100 MPa (1000 bar) with flow rates of up to 265 liters/second, the cracks being produced typically less than 1 mm wide. The fracking fluid contains around 20 percent of sand, and this helps to open and keep open the tiny cracks, allowing gas to flow into the well. Fracturing fluid consists of about 98–99.5% water and proppant. The rest (0.5–2% by volume) is composed of a blend of chemicals, often proprietary, that enhance the fluid's properties. The concentration varies depending on the geology and other water characteristics. These chemicals typically include acids to «clean» the shale to improve gas flow, biocides to prevent organisms from growing and clogging the shale fractures, corrosion and scale inhibitors to protect the integrity of the well, gels or gums that add viscosity to the fluid and suspend the proppant, and friction reducers that enhance flow and improve the ability of the fluid to infiltrate and carry the proppant into small fractures in the shale.

Coal

Coal is a solid, black mineral made up of carbon, hydrogen, oxygen, and nitrogen in varying proportions. In addition, it contains impurities such as ash and sulfur. In the Industrial Revolution, coal was a major fuel competing with charcoal and wood. Coal is an essential fuel for steel and cement production and other industrial activities as well as to provide electricity.

Coal commonly contains altered remains of prehistoric vegetation because it is of vegetable origin, with components growing in swamps and lagoons and going through a peat stage, all with the combined effect of pressure and heat over millions of years to form coal seams. The change from plant debris to coal involves biochemical action, preservation of the material from further decay, and pressure under accumulated plant materials and other later sediments. This caused physical and chemical changes in the organic remains transforming them into peat and then into coal. Coal formation began during the Carboniferous period, called the first coal age, which spanned 360–290 million years ago. However, coal occurs in all post-Devonian periods. For instance, Cenozoic yields most of the lignite of the world.

The types of vegetation, depth of burial, temperature and pressure and this depth, and length of the time forming the deposits are factors to define the quality of a coal deposit. The degree of change undergone by a coal deposit as it matures from peat to anthracite is known as coalification. Coalification has an important bearing on coal's physical and chemical properties and is referred to as the «rank» of the coal. Ranking is determined by the degree of transformation of the original plant material to carbon. The ranks of coals according to the carbon content are lignite, subbituminous, bituminous, and anthracite (Fig. 2.19).

The use of carbon as an energy source causes bad effects on both humans and the environment. Examples of these issues are acid rain, waste products, high levels of carbon dioxide, contaminated water, poisonous emissions, and increased risks of lung cancer for coal plant workers. Despite this fact, nearly 70% of China's electricity comes from coal, and around 40% of the world's electricity is produced after this energy source. Coal seam extraction can be carried out by surficial or underground mining, depending on the depth and quality of the seams and the geological and environmental factors. In addition, lignite can be broken down chemically through a process called coal gasification to create synthetic natural gas. Coal reserves in 1995, 2005, and 2015 by region are shown in Fig. 2.20.

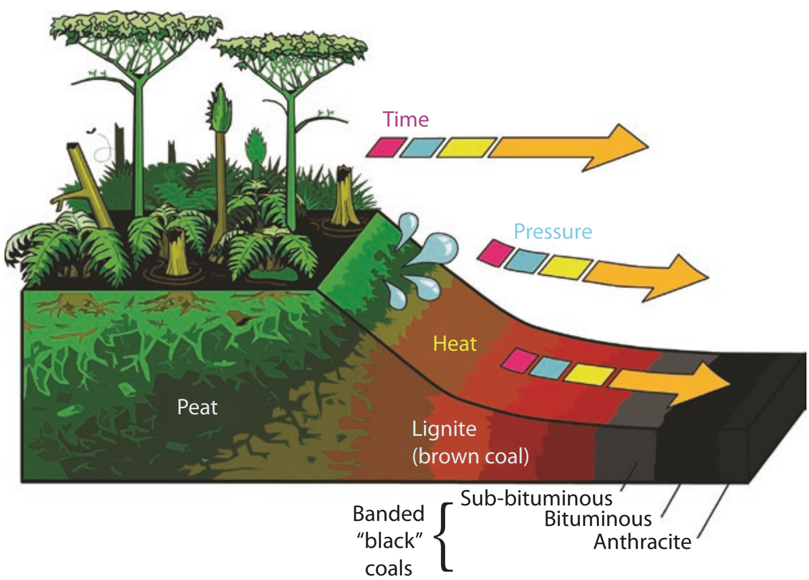


Fig. 2.19 Transformation of the original plant material to carbon and types of coal according their rank (Illustration courtesy of Kentucky Geological Survey)

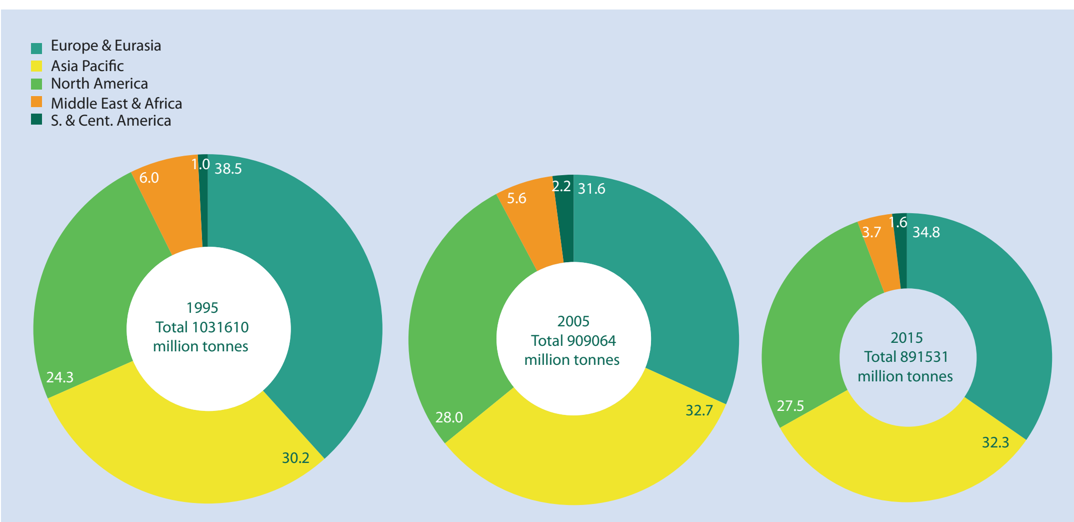


Fig. 2.20 Distribution in percentage by region of coal-proved reserves in 1995, 2005, and 2015 (BP Statistical Review of World Energy 2016)

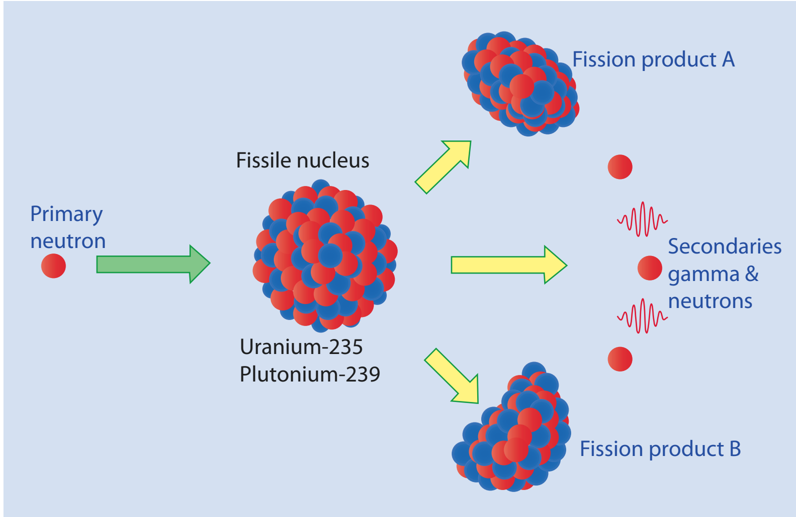


Fig. 2.21 Fission process

Uranium

Uranium is the raw material for nuclear power, a radioactive metal being present on the crust of the Earth. It is important to bear in mind that nuclear power actually originates about 16% of electricity of the world. Uranium can come from mining directly uranium-rich ore bodies or as a by-product from mining other minerals such as copper, phosphate, or gold. In this sense, the uranium concentration in the mineralization can range from 0.03% up to 20%. The most important uranium-rich ore producers in the world are Kazakhstan, Canada, and Australia. There are three methods to obtain uranium in the mine: classical open-pit and/or underground methods and in situ leaching. In the latter, uranium is leached directly from the ore. It is the leading method to produce uranium today in a process called in situ leaching (ISL). The WNA (World Nuclear Association) reports that ISL mining accounted for approximately 49% of world production in 2014. ISL processing implies that mining solution is passed through the underground ore body using several bores or wells. The uranium then is brought to the surface in a dissolved state for further purification. After the chemical treatment to separate uranium, the product is the so-called yellow cake, which is a yellow powder of uranium oxide (U_3O_8) where the uranium concentration is reaching more than 80%.

Natural uranium includes mainly two isotopes: U-238 (99.3%) and U-235 (0.7%). The fission process in the nuclear reactor is carried out using preferably U-235 (Fig. 2.21). Therefore, because nuclear power plants need fuel with U-235 enriched to a level of 3–5%, the material must be enriched to achieve this concentration. Since enrichment process is produced in gaseous form, the «yellow cake» is turn to uranium hexafluoride gas (UF_6). Enriched uranium (UF6) cannot be directly used in reactors so that it must be converted into uranium oxide (UO_2). Fuel pellets are formed by pressing UO_2, which is sintered (baked) at temperatures of over 1400 °C to achieve high density and stability. The pellets are packed in long metal tubes to form fuel rods, which are grouped in «fuel assemblies» for introduction into a reactor. As the spent fuel assemblies are very hot and radioactive, they must be removed from the reactor and are stored under water, which provides both cooling and radiation shielding. After a few years, spent fuel can be transferred to an interim storage facility. After 40 years in storage, the fuel's radioactivity will be about a thousand times lower than where it was removed from the reactor. Some countries chemically reprocess usable uranium and plutonium to separate them from unusable waste.

2.7.2 Metals

Despite some limitations such as their low specific strength or corrosion processes, metals are still one of the most important components of our way of life. This situation will continue in the future, thanks

Fig. 2.22 Metallic mercury from Almadén (Spain)

to unique properties that make them irreplaceable. According to Lu (2010): «metals possess much higher fracture toughness than other materials: steels are the toughest known materials; secondly, the properties of metals are uniform in all directions (their strength is the same in tension and compression and it is usually predictable), being these features critically important for predicting fracture in engineering structures; third, most metals are more conductive than ceramics and polymers; and fourth, they have the best overall mechanical properties at temperatures up to a few hundred degrees; moreover, most metals are recyclable, making them more competitive for quantity applications».

In order to separate the metals in groups, there is a general consensus that five clusters can be outlined: (1) iron and steel metals; (2) base metals, copper, lead, zinc, and tin; (3) precious metals, gold, silver, and PGM; (4) light metals, aluminum and magnesium; and (5) minor and specialty metals, this group is formed by numerous metals (e.g., mercury (Fig. 2.22), antimony, arsenic, bismuth, titanium, cobalt, tungsten, molybdenum, and many others).

Iron and Steel Metals

Iron ore is destined to the production of pig iron in the blast furnace. High iron concentration in ore, low content of SiO_2 and alumina, and coarse grain size are favorable properties. The basic materials for pig iron production are iron ore, coal and coke (also used as energy input to the process), and alternative reducing agents such as limestone and dolomite. The main application of this raw material is to produce steel, the toughest of all construction materials, which is an alloy made of low-carbon iron (steel production requires iron, steel scrap, and lime). Non-metallurgical uses of iron ore, such as chemical applications, pigments, and abrasives, consume a very small share of total iron ore production. Steel is obtained by blowing oxygen through molten iron, thereby reducing its carbon content up to 2%. The properties of steel can be adapted by alloying it with other metals such as manganese, chromium, nickel, cobalt, molybdenum, tungsten, and vanadium, the so-called steel metals.

The most famous and used alloy steel is stainless steel. Iron and the most common iron alloy, steel, are relatively poor materials from a corrosion viewpoint. In spite of this, there is a group of iron-base alloys, the iron-chromium (Fe-Cr) alloys, often with nickel (Ni) additions, known as stainless steels, which do not rust in seawater, which are resistant to concentrated acids, and which do not scale at temperatures up to 1100 °C. The combined effect of the alloying elements, heat treatment and, to some extent, the impurities, establishes the property profile of a certain steel type (Outokumpu 2013). Applications of stainless steel include food handling/processing, medical instruments, and structural/architectural uses, among many others (Fig. 2.23).

Fig. 2.23 Santander (Spain) Sport Hall; the external cover is made with stainless steel

Base Metals

Base metals such as copper and zinc are widely used in communication and information technology. Copper and its alloys exhibit many desirable properties. It is ductile, malleable, hard, tough, strong, wear, and corrosion resistant. It also has high-tensile strength, fatigue strength, and thermal and electrical conductivity. The production of copper is mainly utilized by the wire and cable markets, taking advantage of properties such as the electrical conductivity, corrosion resistance, and thermal conductivity. Excellent malleability, ductility, and resistance against atmospheric attack distinguish copper metal and its alloys (e.g., tin or zinc); copper shows also strong antibacterial properties. Other applications include structural and aesthetic uses.

With regard to zinc and lead, there are few ore deposits that contain only lead or zinc, and most mines produced both metals. Zinc is used predominantly in galvanizing and alloys. Steel coated with zinc (galvanized steel) exhibits high levels of corrosion resistance. This application is responsible for around 50% of total demand. Zinc-based alloys are also used in die casting, ranging from automotive components to toys and models. Lead is a heavy metal, soft, and malleable. Lead is commonly utilized in alloyed form, which increases its low-tensile strength. When added to metal alloy, lead improves their machinability. Regarding tin, it is a soft, weak, malleable, and ductile metal and has many important uses as an alloy. It can be alloyed with lead and with copper to produce bronze. The most important properties of tin-based alloys are their high resistance to corrosion, low-fatigue strength, and compressive strength. For its part, nickel, also a base metal, is hard and ductile (Fig. 2.24), and the main application is in steel alloys.

Fig. 2.24 Nickel briquettes (Image courtesy of Sherritt International Corporation)

Precious Metals

Apart from copper, gold is one of the earliest metals intentionally looked for by humans. Gold was always a metal valued for wealth, adornment, and strong currency. There is little difference today, and only 10% are consumed by industry (e.g., electronics and dental applications). For the future, an increasing role of nano-sized gold particles as catalysts in chemical production, in pollution control, and in medical applications is predicted (Pohl 2011). In respect of silver, it is obtained mainly as a by-product from copper, lead, zinc, and gold ores. In fact, the economic viability of many base metal and gold deposits relies on by-product silver. The use of silver is basically in industrial applications, but nearly 40% is consumed in jewelry, coins, and silverware.

The platinum group metals (PGM) are used in several industrial applications as well as in jewelry. The six chemical elements normally referred to as the platinum group elements (PGE) are ruthenium (Ru), rhodium (Rh), palladium (Pd), osmium (Os), iridium (Ir), and platinum (Pt). Platinum and palladium are actually the most commercially important of the PGM with largest utilization in the automotive industry where they are applied to decrease harmful emissions from vehicle systems (Gunn 2014). Rhodium is the third more important PGM. It is also used in autocatalysts, although its consumption is an order of magnitude less than platinum and palladium.

Light Metals

Aluminum is the most important of the non-iron metals, and it is commonly produced from bauxite, which is a loose soil or a hard rock with 30–65% Al_2O_3. About 95% of bauxite produced is processed into aluminum metal. The remaining 5% serves as an industrial raw material for numerous special products such as abrasives, Portland cement, technical ceramics, glass, chemicals, paints, and refractories. Favorable attributes of aluminum metal such as lightweight, strength, and excellent corrosion resistance (Lu 2010) allow its use in many applications from building air frames to food packaging.

The other light metal is magnesium. Very diverse raw materials, natural and industrial brines, and seawater are used for the production of magnesium and magnesium compounds. For instance, harvesting salts on the shores of the Great Salt Lake is a source of magnesium. Applications of the extremely light magnesium metal (density 1.74 g/cm^3) employ the pure metal or aluminum alloys. Magnesium-aluminum alloys are mainly consumed for beverage container making. About 40% of magnesium is used for die casting in the car industry in order to reduce weight and fuel consumption. Other sectors include the space, aircraft, and chemical industry. Magnesium is mainly utilized as magnesium oxide in applications such as refractory material (e.g., furnace linings for the production of iron and steel), glass, and cement.

Minor and Specialty Metals

The term minor metals encompasses a vast array of metals, including tungsten, titanium, cobalt, and molybdenum, to name just a few. These metals are crucial to the global economy, and many of them are by-products of the major exchange metals. Only the precious metals are more valuable than many of the minor and specialty metals. Minor metals show relatively low annual production volume, compared to base metals, and they have commonly high-technology applications. Uses include filaments in lightbulbs, electronic pastes, components in mobile phones and tablet Pc's, agriculture, and flat panel screens as well as alloying agents in specialist steels for the automotive and aerospace sectors, among many others; as technology progresses, new applications are found which will create new supply and demand patterns, as demonstrated by the growth in renewables technology (Minor Metals Trade Association – MMTA).

Nowadays, one of the most important groups of these metals is rare earth elements (REE) because their chemical properties make them indispensable and non-replaceable in many high-technology applications. For this reason, REE consumption is growing due to their daily contribution to our lives in products like hybrid cars, catalytic converters, wind power generators, household appliances, industrial motors, MRI machines, iPods and computer hard drives, and green energy technology.

2.7.3 Industrial Minerals

The use of the term «industrial minerals and rocks» is very common in the literature (e.g., Kuzvart 1984; Carr and Herz 1989; Jeffrey 2006), and it cover both types of raw materials. In this section, industrial minerals are described separately from industrial rocks because the characteristics of the materials and applications are wholly different. The economically usable minerals automatically classify themselves into four broad groups based on the stages of processing required for conversion to finally usable products (Chatterjee 2009): (1) those that are mainly used directly in consumer product industries, (2) those that are not used without first extracting metals from them, (3) those that are used in both ways but mainly valued for their metal content, and (4) those that are used in both ways, but their direct uses are of importance and their metal values are of minor significance. It has become a convention to refer to the first and fourth groups as «industrial minerals» (earlier called «nonmetallic minerals»), while the second and third groups are considered as «metallic minerals. »

Industrial minerals are valuable economic raw materials that are not used in the production of metals or energy. Compared with metals and other nonmetallic resources, they are mainly processed by physical methods. Both definitions, however, are not without exceptions, and some attributions to the group are rather by tradition (Pohl 2011). Typical examples of industrial minerals are talc (◘ Fig. 2.25), mica, and fluorite. Several ore minerals such as chromite, bauxite, and rutile also have industrial applications, but the bulk of production feeds metallurgy. Because of multiple and even changing uses and a wide genetic variety, the most common classification of industrial minerals is based in the alphabetical order. Occasionally, final applications of the industrial minerals are used as a basis for their classification.

Although the industrial mineral deposits are generally exploited for single minerals, a significant number are worked together as by-products such as fluorite and barite from Mississippi Valley-type lead-zinc deposits or quartz, feldspar, and mica from pegmatites. Most industrial minerals and rock commodities also have multiple uses. For instance, a pure limestone deposit could supply material for lime, aggregate, and cement production, in granular form for flue gas desulfurization, and in a range of powders for fillers, soil stabilization, and agricultural uses. Each of these applications can command very different prices per ton, so evaluating the overall value of the deposit is difficult and involves assessing for multiple quality requirements and variable product splits. In many cases, the evaluation process for an industrial mineral resource is considerably more technically complex than that for metal deposits (Jeffrey 2006).

Globalization is an important economic driver in the industrial mineral sector. Large international corporations (e.g., Sibelco in Belgium) have formed by consolidation and acquisition of smaller companies. In some cases, this process has led to one or two corporations having dominant control over individual mineral commodities such as borates, nepheline syenite, garnet, and talc. As the technical demands on specific minerals increase or supplies are restricted, companies explore the possibilities of making synthetic mineral products. This is especially true for gemstones, but

◘ **Fig. 2.25** Talc mine at León (Spain) major industries are making synthetic zeolites for use in washing powders, as intermediates such as synthetic rutile for TiO_2 manufacture and in pigments, and as bulk materials such as magnesite, gypsum, and soda ash (Jeffrey 2006).

In some applications, the boundaries with material science become blurred such as in industries making synthetic corundum and silica for laser, military, and electronic applications. Here the mineral structure has been perfected to a point not found in nature. Some of these synthetic minerals are also produced as by-products of upgrading other mineral products, but all affect the demand for primary industrial minerals from new or existing deposits. More often, a shortage of suitable mineral supplies, or the possibility of cost savings, leads to substitution by function. Other minerals that can perform the same role in a product are then used instead. The increased use of fine-ground or precipitated calcium carbonate at the expense of kaolin in paper coating is a good example.

As a tool to assist in teaching about industrial minerals, a classification that defined seven groups of commodities based on the relative importance of physical and chemical applications or a combination of the two can be established (Smith 1999). The classification is constructed using a matrix of commodities and uses that are grouped according to applications. Clustering of commodities reveals the following groupings: (1) principal abrasives (diamond, alumina, garnet, and pumice), (2) principal refractories (pyrophyllite, sillimanite group, magnesite, and graphite), (3) principal fillers (wollastonite, titanium minerals, mica, barite, and iron oxide), (4) principal physical and chemical minerals (feldspar and zeolite), (5) mixed-application physical minerals (silica, perlite, clay (◘ Fig. 2.26), and talc), (6) principal chemical minerals (phosphate, salt, and sulfur), and (7) mixed-application physical and chemical minerals (olivine, chromite, fluorspar, gypsum, and limestone).

Regarding the trade value of industrial minerals, most of them are essentially high-volume, low-value commodities, while metals are the opposite, mainly precious metals. Beyond the difference in scale of value between the two groups of commodities, a key issue is the fact that industrial minerals do not have markets whose prices are set by an exchange system (e.g., London Metal Exchange in metals). Some attempts have been made by various organizations in recent years, especially with

◘ **Fig. 2.26** Clay (bentonite) quarry from Milos (Greece) (Image courtesy of José Pedro Calvo) the advent of the Internet and e-commerce. For the reasons outlined here, it is unlikely that any kind of industrial mineral pricing exchange will be created in the foreseeable future.

Industrial Mineral Applications

The industries in which industrial minerals are utilized are cover paint, electronic, metal casting, paper, plastic, glass, ceramic, detergent, pharmaceutical and cosmetic, environmental engineering, and construction (IMA Europe). For instance, glass in buildings is manufactured with industrial minerals, mainly silica. The following descriptions are a brief resume of the application of industrial minerals in these sectors.

The glazes that cover ceramics are largely composed of minerals, mainly borates, silicates, and metallic pigments. Ceramics and refractory articles are indispensable in buildings: pipes, tiles, and refractory bricks are all 100% industrial minerals. Even if some ceramics are being replaced by resins, these also contain important amounts of industrial minerals. Technological developments in the ceramic sector represent an area in which industrial minerals are at the forefront of progress. For instance, ceramic tiles protect space shuttles in order to support the high temperatures of the Earth's atmosphere.

Industrial minerals such as clays, sand, feldspar, kaolin, and other minerals are basic to all construction materials, from bricks to tiles and from cement to limes and plastics. Apart from the basic structure, industrial minerals are also present in all parts of the building as a constituent or during their manufacturing. For instance, all ceramic compounds of a house (e.g., tiles, tubing, etc.) include industrial minerals. Even the wallpaper, paints, and carpet lining contain important amounts of industrial minerals.

Detergents such as the powder ones utilized for laundry and dishwashers include a «bleaching system. » Two systems are currently in use: perborate and percarbonate. Both rely on industrial minerals, borates, or calcium carbonate, respectively, which are chemically processed up to the required properties. Detergents are a major consumer of silica, which makes a whole family of detergents based on sodium silicate. Other industrial minerals (e.g., bentonite and sepiolite) are also used in detergent applications because of their adsorption properties. In this sense, sepiolite is the main component in making cat litter (Fig. 2.27).

The nervous system of a computer is made of silicon, this component being extracted from silica sand or massive quartz rocks. This quartz crystals also pace actually the functioning of most of clocks. After extraction, the silicon is delivered to the electronic manufacturers in the form of «wafers» a few centimeters wide.

Industrial minerals are crucial in water management, whether considering drinking water preparation or wastewater treatment. Thus, silica sands are used as filters, perlite, zeolites, or talc as flocculants or adsorbents, bentonite as

Fig. 2.27 Cat litter manufactured with sepiolite (Image courtesy of SAMCA)

a degreasing agent, and calcium carbonate as a neutralizing agent, to mention but a few. Industrial mineral-based liners and geosynthetic liners, either basal or superficial, are increasingly used to avoid escape of leachate from landfill sites. Air treatment of industrial effluents also largely relies on minerals. Activated carbon is the best-known technique, but other minerals are used as well. For instance, flue gas desulfurization of power station fumes is achieved with calcium carbonate.

The glass industry is one of the primary consuming markets for industrial minerals with the highest demand in terms of volume for silica sand, limestone, feldspar, and soda ash. Fiberglass and glass wool are also members of this group. The mineral blend is a determinant to the glass properties during manufacture and use.

Historically, fillers and extenders were used to furnish low-cost bulk to paint solid content. Today, the range of extenders available is extremely wide and determines many of the paint's properties: gloss, opacity, flow, film toughness, permeability, rheology, resistance, etc. Waterborne systems, low solvent paints, powder coatings, high-solid coatings, and industrial minerals are crucial to all the environment-friendly developments of paint technology. The paper industry, particularly printing and writing paper, is by far the largest volume user of industrial minerals (Box 2.5: Papermaking Additives).

Box 2.5

Papermaking Additives

Papermaking starts with the production of the most important raw material: wood. The pulping process then converts the wood into the most appropriate type of pulp. Pulping of wood can be done in two ways: mechanically or chemically. In the case of mechanical pulp, the wood is processed into fiber form by grinding it against a quickly rotating stone under addition of water. In chemical pulp, the pure fiber has to be set free, the wood chips being cooked in a chemical solution. The next step is pulp bleaching. It is a complex process consisting of several chemical process steps with washing taking place between the various chemical treatments. The paper machine then converts the pulp into a thin base paper, which, at the end of the production process, is coated to give it a superb flat surface and bright shade. Coating a paper enhances its optical and tactile characteristics (whiteness and shade, gloss, and smoothness), but it also improves its printing behavior, allowing the use of very fine screens, yielding more color in thinner ink layers, and producing more contrast in printed images.

In all the previous processes, many types of additives (fillers, binders, and many others) are used to improve the efficiency and quality of the final product. In papermaking, minerals are used either as fillers or as a coating on paper. Some minerals, like talc, are also used in pitch control (absorption of wood resins that tend to obstruct the machines). The use of minerals in paper production increases the speed of the machine performance and fluidity. The final characteristics of the paper (strength, whiteness, gloss, ink retention, etc.) are largely determined by the blend of minerals used. High-quality, glossy paper is obtained by applying a thin layer of industrial minerals on the surface of the paper. As for fillers, the final characteristics of the coating and its fitness for use are governed by the nature of the mineral blend.

The list of minerals used as additives in papermaking is impressive. Soda ash dissolves out the noncellulose parts without weakening the finished paper. Titanium dioxide is a strong white pigment which makes paper whiter and more opaque, acting as a filler and giving a smoother surface to the paper. The filling effect is much stronger than with calcium carbonate, but it does not have the ability to neutralize paper acids. Titanium dioxide is also used to tint-colored pulps. China clay is a fine white powder, also known as opal gamma kaolin, which is used to make paper more opaque and smooth and reduce shrinkage. It is especially useful in paper casting and will appeal to papermakers and model makers alike. Calcium carbonate provides an alkaline reserve in paper which promotes acid-free archival qualities, being also used as filler and in coating. It retards shrinkage in paper castings and makes a smoother surface. In paper sheets, it improves opacity and whiteness. Talc gives paper a greasy or soapy feel and enables it to take a high finish. Kaolin is one of the most used filler. Lime is used in alkaline pulping process. Magnesite is a common component of cigarette paper as filler, being also considered as an excellent ingredient for harmless smoking; hydromagnesite and huntite are used to control the burning rate of cigarette papers. Sodium silicate is utilized in waste paper deinking for wetting, ink dispersion, and peroxide stabilization. Finally, many pigments and dyes used in papermaking come from industrial minerals such as iron oxide, titanium oxide, zinc compounds (e.g., zinc sulfide or zinc oxide), lead compounds, cadmium sulfide, etc. A type of mixture containing coprecipitates of titanium and mica (or other minerals) is used to make a pearlescent, which is transparent and highly light refractive, imparting to the ink film the luster characteristic of mother-of-pearl.

The role of industrial minerals in pharmaceuticals falls into one of two main categories: excipient or active substances. The excipients have no intrinsic health benefit on their own; they are used solely as carriers, allowing the intake of minute amount of active substances. pH regulation or adsorbents are the kind of applications for which some minerals are used as active ingredients. Thus, antacid pills are composed mainly of calcium carbonate, lithium used in antidepressants is derived from industrial minerals, and many excipients are minerals such as talc, magnesium carbonate, or silica. Many cosmetics incorporate important amount of industrial minerals such as talc, although others like mica, silica, or borates are utilized for their abrasive, visual, or stabilizing properties. It is necessary to remember that earliest civilizations (e.g., the Romans) already made use of earth pigments for body painting.

Finally, polymeric resins such as PVC and PP are generally filled and/or reinforced with industrial minerals (e.g., talc and calcium carbonate). They are also used in polyamide, unsaturated polyesters, HDPE (high-density polyethylene), and LDPE (low-density polyethylene). Small amounts of minerals, particularly talc and silica, are used in the compounding and manufacturing of rubber goods. Thus, the new-generation car tire relies its energy saving on the silica content of the polymer.

2.7.4 Industrial Rocks

This term encompass a group of rocks (single mineral species are excluded of this group) whose main application is addressed to construction market. Industrial rocks typically comprise of multi-mineral hard and unconsolidated rocks and sediments. Aggregates (sand and gravel) for road construction, limestone for cement, and dimensional stone (granite, marble or slate) for building material are all well-known examples of industrial rocks.

The main characteristics of industrial rocks can be outlined as follows. Firstly, the price of the raw material is low to very low, and sometimes the price of the finished product is low as well. One of the principal industrial rocks, namely, aggregates, displays the lowest price for a raw material in the industry (e.g., 7 dollars per ton of concrete sand). For this reason, exploration, exploitation, and mineral processing costs must be very low. Secondly, the very high prices of transport resulted in a proximity to consumption center. Consequently, the markets for the product are commonly local markets, especially in aggregate industry. Finally, resources and reserves of industrial rocks are almost infinite. Moreover, one particular type can be substituted by another one, for example, if the price of the product suddenly increases, and even a finished product can also be replaced by another one of similar specifications. According to the main markets of construction, industrial rocks can be grouped in to five main types: (1) aggregates, (2) ornamental rocks, (3) limestone for cement and lime, (4) gypsum, and (5) clay for bricks and tiles.

Aggregates

Aggregates are granular materials used in construction formed of natural or crushed, hard, sound, and durable particles of nonreactive minerals. Sand, gravel, and crushed rock are typically the most common natural aggregates in the market. While aggregate is used primarily in asphalt and concrete (asphalt pavement includes 94% aggregate and concrete is formed by 80% aggregate), all construction worldwide involves the use of this raw material. In fact, aggregates are the second natural resource more used by the humankind, after water. Aggregates are mainly obtained by mining quarries and gravel pits and in some countries from sea-dredged materials (marine aggregates). Recycled aggregates (see ► Chap. 1) are derived from reprocessing materials previously used in construction such as demolition debris.

The production and consumption of this raw material are impressive. According to the Union Européenne des Producteurs de Granulats (UEPG – European Aggregates Association), the European aggregate consumption is 2.8 billion tons per year, the average aggregate consumption being 5.2 tons per person per year; about 90% of all aggregate produced are from quarries and pits (25,000 quarries and pits in Europe) and the remaining 10% from recycled aggregates (6%) and marine and manufactured aggregates (2% each). US aggregate consumption includes approximately 1 billion tons of sand and gravel in 2016 and similar amount of crushed rock (USGS).

Aggregates are indeed the main component in all homes, offices, social buildings, and infrastructures. For instance, the construction of a common new home uses up to 400 tons of aggregates, from the foundations through to the roof tiles. Other

Fig. 2.28 Aggregates forming a breakwater or armour stone

examples are roads or railways: the construction of 1 km of motorway consumes up to 25,000 tons of aggregates, and the construction of 1 m of railway for a high-speed train (TGV) uses up to 10 tons of aggregates. Drainage, dams, and breakwaters (Fig. 2.28) are a few more of other important construction items involving aggregate.

Specifications for the most important applications of aggregates such as concrete and ballast are closely regulated and subject to industrial standards (e.g., ASTM in the USA, EN in Europe, ISO worldwide) and concern petrographical composition, geometrical properties such as particle size and grain shape, mechanical and physical properties (e.g., resistance to wear or resistance to fragmentation), thermal and weathering properties (e.g., boiling test for Sonnenbrand basalt), and chemical properties (e.g., determination of acid-soluble chloride salts).

Ornamental Rocks

For centuries, natural stone has been used by nearly all civilizations, being applied mainly in architecture. Ornamental rocks are the main economic component of the natural stone industry. The market is shaped by three rock types: granites (Fig. 2.29), marbles, and shales (Fig. 2.30), although they do not always represent the same typology of geological rock. Thus, limestone is a marble in natural stone industry, although obviously the limestones need a metamorphism to become a marble. Other example is basalt, which is defined as granite in natural stone industry, although the former is a volcanic rock and the latter a plutonic one.

Ornamental rock blocks are exploited in quarries. Currently, the most common method to extract the blocks is by using diamond wire. Diamond wires are cutting tools for rocks (marble, granite, or slates). The wires are composed of a stainless steel cable over which are assembled diamond-sintered pearls, 10–12 mm in diameter and spaced 25 mm along the wire. The utilization of this slabbing technology has expanded all over the world due to its advantages facing other techniques such as explosives or thermal lance. After extraction in quarries, the blocks are manufactured using different techniques, which depend on the size of the products and the type of rock. Marble or granite is commonly polished to perform products for interior paving. Granite is also processed to obtain flamed granite, most used in pavements.

Carbonate Rocks for Cement and Lime

Carbonate rocks are extremely important raw materials for industry, construction, agriculture, forestry, and environmental engineering. The most representative application of these rocks is in cement and lime industry. Cement is a fine powder that sets after a few hours when mixed with water. It then hardens in a few days into a solid and strong

Fig. 2.29 Granite quarry for ornamental rock at Cadalso de los Vidrios (Spain) (Image courtesy of Marcelino Martínez)

Fig. 2.30 Underground shale quarry for ornamental rock (roofing) (León, Spain)

material. The so-called Portland cement is the most classical type of cement although there are many types of common cement. This product is manufactured in a controlled chemical combination of mainly calcium, silicon, aluminum, and iron (Box 2.6: Manufacture of Cement). The main use of cement is to make concrete, the most important construction material in the last century.

Box 2.6

Manufacture of Cement

Cement is a fine gray powder that, when reacted with water, hardens to form a rigid chemical mineral structure that gives concrete its high strengths. The credit for its discovery is given to the Romans, who mixed lime ($CaCO_3$) with volcanic ash, producing a cement mortar that was used during construction of such impressive structures as the Colosseum. When the Roman Empire fell, information on how to make cement was lost and not rediscovered until many centuries later. Roman cement was not improved upon until 1758, when Smeaton noticed that using a limestone that was 20–25% clay and heating the mixture resulted in a cement that could harden under water.

Portland cement (the most common type of cement in common use today) is manufactured in a four-step process: (a) quarrying, (b) raw material preparation, (c) clinkering, and (d) cement milling and mixing. The name Portland was given owing to the resemblance of this hardened cement paste to the natural stone available at a place called Portland in England. Most common way to manufacture Portland cement is through the so-called dry method. The raw material for cement manufacture is a rock mixture of about 80% limestone (which is rich in $CaCO_3$) and 20% clay or shale (a source of SiO_2, Al_2O_3, and Fe_2O_3). Lime and silica provide the main strength of the cement, while iron reduces the reaction temperature and gives the cement its characteristic gray color.

Raw material preparation includes a variety of blending and sizing operations that are designed to provide a feed with appropriate chemical and physical properties. Thus, quarried clay and limestone are crushed separately, and samples of both rocks are then sent off to the laboratory for mineral analysis. If necessary, minerals are then added to either the clay or the limestone to ensure that the correct amounts of aluminum, iron, etc. are present. Since the four basic oxides must be present in exact proportions (calcium oxide, 65%; silicon oxide, 20%; alumina oxide, 10%; and iron oxide, 5%), limestone and clay are mixed together with many other raw materials such as slate, marl, blast furnace slag, silica sand, iron ore, and much more. These are called correctors because they must define the final proportions of all oxides. The clay and limestone and correctors are then fed together into a mill where the rock is ground until the material is less than 100–200 μm in diameter.

In the third step of manufacturing, the fine-grained raw materials are then dried, heated, and fed into a rotating kiln (◘ Fig. 2.31). Here the raw materials react at very high temperatures to form $3CaO{\bullet}SiO_2$ (tricalcium silicate), $2CaO{\bullet}SiO_2$ (dicalcium silicate), $3CaO{\bullet}Al_2O_3$ (tricalcium aluminate), and $4CaO{\bullet}Al_2O_3{\bullet}Fe_2O_3$ (tetracalcium aluminoferrite). Minor compounds such as MgO, TiO_2, Mn_2O_3, K_2O, P_2O_5, and Na_2O are also present in clinker. The cement kiln heats all the raw materials to about 1500 °C in huge cylindrical steel rotary kilns (60 m long). The materials are continuously and slowly moved to the lower end by rotation of the kiln. A burner is located at one end of the kiln, and the ground raw materials are introduced at the other end. As the material moves through the kiln, some elements are driven off

◘ **Fig. 2.31** Rotating kiln to manufacture cement (Image courtesy of Grupo Cementos Portland Valderrivas) in the form of gases. The remaining elements are joined to form a new substance called clinker and formed by gray balls. They are discharged red-hot from the lower end of the kiln and commonly are brought down to handling temperature in various types of coolers.

The final step includes clinker milling and mixing with other components to obtain the so-called Portland cement. Thus, after the clinker is cooled, cement plants grind it in large ball mills to obtain a very fine powder (e.g., 20 µm). Finally, it is then mixed with small amount of either gypsum or anhydrite, both of which are forms of calcium sulfate (as setting retardant) and other materials. It is essential to note that cement manufacture is an energy-intensive process.

One of the most significant challenges facing the industry into the twenty-first century is a requirement to reduce CO_2 emissions. CO_2 is produced during the calcination phase of the manufacturing process and also as a result of burning fossil fuels. Opportunity to reduce emissions through increased energy efficiency is only possible on the latter of the CO_2 emissions. In this sense, due to the characteristics of the production process, the cement industry is capable of coprocessing (a) alternative fuels, which have significant calorific value (e.g., waste oils); (b) alternative raw materials, the mineral components of which mean they are suitable for the production of clinker or cement (e.g., contaminated soil); and (c) materials that have both a calorific value and provide mineral components (e.g., paper sludge, used tires). Without coprocessing, the wastes and by-products that make up these materials would have to be incinerated or landfilled with corresponding greenhouse gas emissions.

Regarding the other main product obtained from carbonate rocks, lime is a term specifically used to refer high-quality products such as quicklime (CaO) and calcium hydroxide, also known as hydrated lime ($Ca(OH)_2$). The raw material for all this type of products is limestone, commonly formed by almost exclusively calcium carbonate ($CaCO_3$). Limestone is processed to form lime, being heated in a specially designed kiln to over 900 °C. In this process, called calcination, a chemical reaction occurs and creates calcium oxide. The applications of lime are huge, but those in environmental engineering are the most widely consumed (e.g., soil conditioning or to neutralize the acidic effluents).

Clays for Bricks and Tiles

Clay rocks are cohesive unconsolidated or indurated clastic sedimentary rocks where size fraction lower than 0.002 mm is dominant. They vary considerably in physical properties, color, and mineralogical content. Clay rocks mainly consist of clay minerals such as kaolinite, illite, montmorillonite, chlorite, and mixed-layer clay minerals. Besides clay minerals, clay and claystone contain fine-grained clastic silicates (quartz, mica, and feldspar), biogenic matter (carbonate microfossils, kerogen, and coaly particles), and diagenetic minerals (marcasite, pyrite, carbonate, and phosphate). These clays are mainly used for the production of bricks, roof tiles, ceramic tiles, and other fired and sintered products. Ceramic materials are one of the most important components of the construction industries, and they are primarily utilized as building materials. These include two big groups: (a) bricks and roof tiles and (b) ceramic tiles. Clay for bricks and roof tiles is used in a wide range of buildings from housing to factories as well as in the construction of tunnels, bridges, etc. In brick- and roof tile-making terms, clay includes a range of naturally occurring raw materials. In manufacturing process, clay must possess some specific properties and characteristics. It usually shows the most important property to obtain these products: plasticity. This property permits clay to be shaped and molded when mixed with water.

All types of clays used for bricks and roof tiles contain some percentage of silica and alumina sand, silt, and clay with varying amount of metallic oxides. Metallic oxides act as fluxes promoting fusion of the particles at lower temperatures (950 °C). In geological terms, the key in the manufacturing process is the mineral content of the raw material. Due to variances in the age of the deposits, depositional conditions, and impurities involved, there are variations between different clay types even in the same deposit. These variations may affect the brickmaking process and the properties of the finished product.

Regarding the second group, ceramic (wall and floor) tiles (Fig. 2.32), they are made from clay and other inorganic raw materials that are ground and/or mixed and then molded before drying and firing at sufficiently high temperatures (1400 °C)

Fig. 2.32 Glazed ceramic tiles (Image courtesy of José Pedro Calvo)

to acquire the necessary stable properties. The raw materials that make up the ceramic tile are essentially clays, feldspars, sand, carbonates, and kaolin. From a glazed point of view, ceramic tiles can be unglazed or glazed. The former is fired only once, whereas glazed tiles include a vitrified coating between the firing. The manufacturing of glaze and frit is a complex process involving many different raw materials, such as carbonates, silicates, borates, and many others.

2.8 Genetic Classification of Mineral Deposits

According to the main ore-forming processes, a simple genetic classification of mineral deposits encompasses four main groups: (1) magmatic, (2) hydrothermal, (3) sedimentary, and (4) metamorphic/metamorphosed. The following is a description of the main classes included in these groups. However, it is not obviously an exhaustive overview of all types of mineral deposits existing in the Earth's crust.

2.8.1 Magmatic Ore Deposits

A magmatic ore deposit is formed by an accumulation of magmatic minerals. Some of them are extremely rare and almost never encountered in common rocks (e.g., alloys of the platinum metals). However, other minerals such as magnetite are common. A very large and diverse group of ore deposits originates by magmatic processes. According to Ardnt and Ganino (2012), many magmatic ore deposits are hosted by granites, but the ore results from precipitation of ore minerals from aqueous fluids and not from the granitic magma itself. The type of ore mineral in magmatic deposits is directly linked to the composition of the host rock. For instance, deposits of nickel, chromium, and platinum group elements are founded in mafic-ultramafic hosts. By contrast, felsic rocks generate ores from the elements confined that concentrate in evolved magmatic liquids. Some of these elements are present in late-crystallizing phases such as ilmenite, which contains Ti and cassiterite; the ore of Sn and others enter the water-rich fluid that separates from the silicate liquid to be redeposited in pegmatites or in hydrothermal ore bodies. Pegmatites are also an important source of rare but increasingly important metals such as lithium and beryllium.

Crystallization of economic minerals normally occurs from mafic to ultramafic magmas that are low in viscosity and have important content of nickel, copper, and platinum group elements. Magmatic ore deposits associated with ultramafic and mafic rocks span most of the history of the Earth, being well represented in all continents. «Currently, these deposits are estimated to account for approximately 7% of the total value of annual global metal and mineral mining and they include the world's greatest concentration of metals: the Bushveld Complex, which has an estimated total metal endowment value, representing past production and current reserves and resources, of US $3.6 trillion» (Peck y Huminicki 2016).

The description of magmatic deposits can be carried out according to the host rock association or related to the commodity. The latter is easier and allows to summarize the main groups of deposits present in the Earth's crust from a magmatic viewpoint. On this basis, four types of magmatic ore deposits can be defined: (1) chromite deposits, (2) nickel (copper) sulfide deposits, (3) platinum group element (PGE) deposits, and (4) diamond deposits. Since the four types can be considered as orthomagmatic deposits, a fifth type related to granitic pegmatites can be added.

Chromite Deposits

Chromite (Mg, Fe^{2+}) (Cr^{3+}, Al, Fe^{3+})$_2O_4$ is the only commercial source of chromium. The source to obtain this metal comes mainly from four different mineral deposit types: podiform deposits, stratiform deposits, placer deposits, and laterites. The latter are derived from weathering of ultramafic rocks that contain chromite. In particular, most of the world's resources are located in stratiform chromite deposits such as the Bushveld Complex (South Africa) (◘ Fig. 2.33) and the Great Dyke (Zimbabwe). The Bushveld Complex contains the main type examples of ore deposits in a large layered intrusion. Important podiform chromite deposits are located in Kazakhstan, Turkey, the Philippines, New Caledonia, and Russia. Known resources of alluvial and eluvial placer deposits derived by erosion of such rocks are low in grade and of very minor importance (Misra 2000). The major stratiform chromite deposits also contain important contents of platinum, palladium, rhodium, osmium, iridium, and ruthenium.

Regarding the genesis of the deposits, little consensus has been reached about the magma chamber processes responsible for chromite segregation and crystallization although extensive studies have been carried out. In general, the most widely accepted explanation involves the mixing of primitive and fractionated magmas. Thus, the commonly cited hypotheses include: «(1) mixing of a parent magma with a more primitive magma during magma

◘ Fig. 2.33 Stratiform chromite at South Africa (Image courtesy of Roland Oberhänsli) chamber recharge; and (2) contamination of the parent magma by localized assimilation of country rock at the roof of the magma chamber; the mixing of magmas would produce a partially differentiated magma, which could then be forced into the chromite stability field and result in the massive chromitite layers found in stratiform complexes» (Schulte et al. 2012). In this sense, chromitite is a term used for massive chromite containing 50% to more than 95% of cumulus chromite.

The sequences of massive chromitite layers (>90% chromite) or seams of disseminated chromite (>60% chromite) are commonly found in the lower ultramafic parts of the layered intrusions. These intrusions were emplaced in stable cratonic settings or during rift-related events throughout the Archean or early Proterozoic, although a few younger deposits exist. The intrusions extend anywhere from 2 to 180 km in diameter and can reach thicknesses of as much as 15 km. As a rule, the individual seams included in the intrusions range from less than 1 cm to 5–8 m thick. The mineral occurs in layers that reach a meter or more in thickness alternating with layers composed of other magmatic minerals (Arndt and Ganino 2012). In some cases, the chromite deposit is not economic due to the low grade of the mineralization or the low tonnage of chromite available for mining.

Podiform chromite deposits, another important source for chromite, are small magmatic chromite mineralization originated in the ultramafic part of an ophiolite complex in the oceanic crust. Most podiform chromite deposits are located in dunite or peridotite close to the contact of the cumulate and tectonite zones in ophiolites (Mosier et al. 2012). Accordingly, chromite that occurs in podiform deposits has a geotectonic environment distinctly different from the model in stratiform chromite deposits. In podiform deposits, chromite shows different textures such as massive aggregates and banded, nodular, net, or graded layers, which indicate relict cumulate features. Nodular texture is probably the most important feature to distinguish podiform chromite deposits from stratiform deposits.

Nickel (Copper) Deposits

These deposits are referred as magmatic sulfide-rich Ni-Cu ± PGE deposits related to mafic and/or ultramafic dyke-sill complexes. The name of the deposits emphasizes the relation of these Ni-Cu sulfide-rich deposits to mafic and ultramafic rocks and to mostly small- to medium-sized dykes and sills, as opposed to the generally much larger layered mafic-ultramafic intrusive complexes that typically host sulfide-poor PGE-enriched deposits such as Stillwater Complex in Montana (USA). Nickel sulfide deposits can be classified into two principal classes based on the petrology of the host rocks: peridotite-dunite class (komatiitic association) and gabbroid class (tholeiitic association) (◘ Fig. 2.34).

According to Schulz et al. (2014): «sulfide deposits containing nickel and copper with or without (±) platinum-group elements (PGE) account for approximately 60% of the world's nickel production and they form where mantle-derived, sulfur-undersaturated picrite or tholeiitic basalt magma becomes sulfide-saturated, commonly following interaction with continental crustal rocks; sulfur saturation results in formation of an immiscible sulfide liquid, which tends to segregate into physical depressions in the lower parts of dike- and/or sill-like intrusions because of changes in the magma flow dynamics; such dynamic systems appear to promote the interaction of sulfide liquid with a sufficiently large amount of silicate magma to concentrate chalcophile elements to economic levels». The ore metals nickel, copper, and the PGE are all chalcophile and show a tendency to partition more or less strongly into the sulfide. Nickel is lithophile as well as chalcophile, and in normal ultramafic rocks, it is distributed between olivine and sulfide. Copper is moderately chalcophile, but the PGEs are enormously chalcophile. This means that any droplet of sulfide will extract most of the copper and nickel and effectively all of the PGE from the surrounding silicate liquid. In this sense, if the sulfide droplets can then be concentrated effectively, for instance, by gravitational processes, then an ore deposit is formed (Arndt and Ganino 2012).

Deposits of magmatic Ni-Cu sulfides occur with mafic and/or ultramafic bodies emplaced in diverse geological settings. They generally are found in penetrating faults, which permit the efficient transport of magma undersaturated in sulfur from the mantle to relatively shallow crustal depths. For this explanation, sulfur-bearing crustal rocks such as black shales, evaporites, or paragneisses are near to many deposits and a potential source of sulfur. These deposits range in age from Archean to Cenozoic, but the largest number of deposits are Archean and Paleoproterozoic. Although the

Fig. 2.34 Aguablanca mine (Spain), a gabbroid class sulfide-rich nickel-copper deposit (Image courtesy of Lundin Mining Corporation)

deposits occur in most continents, the biggest ones are located in Russia, China, Australia, Canada, and Southern Africa. The major Ni-Cu sulfide mineralogy typically consists of an intergrowth of pyrrhotite, pentlandite, and chalcopyrite. In most cases, the massive and matrix ore is zoned, with copper-rich zones relatively enriched in gold, palladium, and platinum. Those zones, as footwall dykes and veins, either overlie or are separated from Cu-poor zones relatively enriched in osmium, iridium, ruthenium, and rhodium. The compositional zonation is attributed to fractionation of monosulfide solid solution from a sulfide liquid. Cobalt, PGE, and gold are extracted from most magmatic Ni-Cu ores as by-products, although such elements can have a significant impact on the economics in some deposits, the Noril'sk-Talnakh deposits being a good example, which produce much of the world's palladium; in addition, these deposits may contain between 1 and 15% magnetite associated with the sulfides (Schulz et al. 2014).

The sulfide-rich Ni-Cu ± PGE deposits contain ore grades of between 0.5% and 3% of nickel and between 0.2% and 2% of copper. Tonnages of individual deposits range from a few tens of thousands to tens of millions of tons bulk ore. Two giant Ni-Cu districts, with ≥10 Mt nickel, dominate world nickel sulfide resources and production. These are the Sudbury district in Ontario (Canada) where sulfide ore deposits are at the lower margins of a meteorite impact-generated igneous complex and contain 19.8 Mt of nickel and the Noril'sk-Talnakh district in Siberia (Russia) where the ore deposits are in subvolcanic mafic intrusions and contain 23.1 Mt of nickel. Three other Ni-Cu sulfide deposits in the world are also important: Voisey's Bay in Newfoundland, Kambalda in Australia, and Jinchuan in China.

PGE Deposits

The concentration of PGE in terrestrial environments ranges from sub-ppb level in rocks of felsic and intermediate composition to generally 1–100 ppb in mafic and ultramafic rocks. Economic deposits typically contain 5–10 ppm PGE and involve concentration factors in the order of

1000, similar to those for gold deposits. Anomalous concentrations of PGE are known from high-temperature magmatic to low-temperature hydrothermal and sedimentary environments, but significant concentrations of PGE are virtually restricted to ultramafic rocks. Two types of deposits, both intimately associated with Ni-Cu sulfides, account for about 98% of the world's identified PGE resources: (a) stratabound deposits in large, layered complexes (e.g., Bushveld, Stillwater, and Great Dyke) mined primarily for PGE and (b) Ni-Cu sulfide deposits mined primarily for Ni-Cu sulfides, but containing recoverable amounts of PGE as by-products (e.g., Sudbury, Noril'sk-Talnakh, Jinchuan, and Karnbalda deposits) (Misra 2000). The large layered intrusions contain about 90% of the world's PGE resources, with the Bushveld Complex accounting for about 80%.

In general, the deposits generally occur as sparsely dispersed sulfide minerals in basal units or stratabound layers or reefs in very large- to medium-sized, typically layered mafic and/or ultramafic intrusions. In the Bushveld Complex, there are in the lower part important deposits of the platinum group elements mainly at two specific horizons. The upper layer is the Merensky Reef, a thin (1–10 m) layer of pegmatoid pyroxenite. The second principal mineralized layer, termed UG2, is a group of thick chromite reefs that, in addition to high PGE concentration, are also extracted for their chromium contents (Arndt and Ganino 2012). There is no consensus regarding the origin of these types of PGE deposits: one line of thinking argues that these deposits formed through magmatic processes, whereas the opposing view ascribes an important role to the migration of volatile-rich fluids. Arndt and Ganino (2012) also said that: «in the first case, a plume of primitive magmatic liquid was injected into the base of the chamber and then mixed with evolved liquid to produce a hybrid magma that became saturated in sulfide. The other view propose that volatile-rich fluids migrated up through the cumulus pile, leaching out the PGE from the cumulus minerals then redepositing them at favorable horizons.»

Diamond Deposits

Diamonds form under extreme high pressures and temperatures at depths greater than 150 km below the surface, predominantly though not exclusively, in the Earth's lithospheric upper mantle. They are transported into the crust either rapidly in explosively emplaced volatile-rich kimberlite, lamproite, or related magmas or more slowly by tectonic processes in rocks that have undergone ultrahigh-pressure metamorphism (Fig. 2.35).

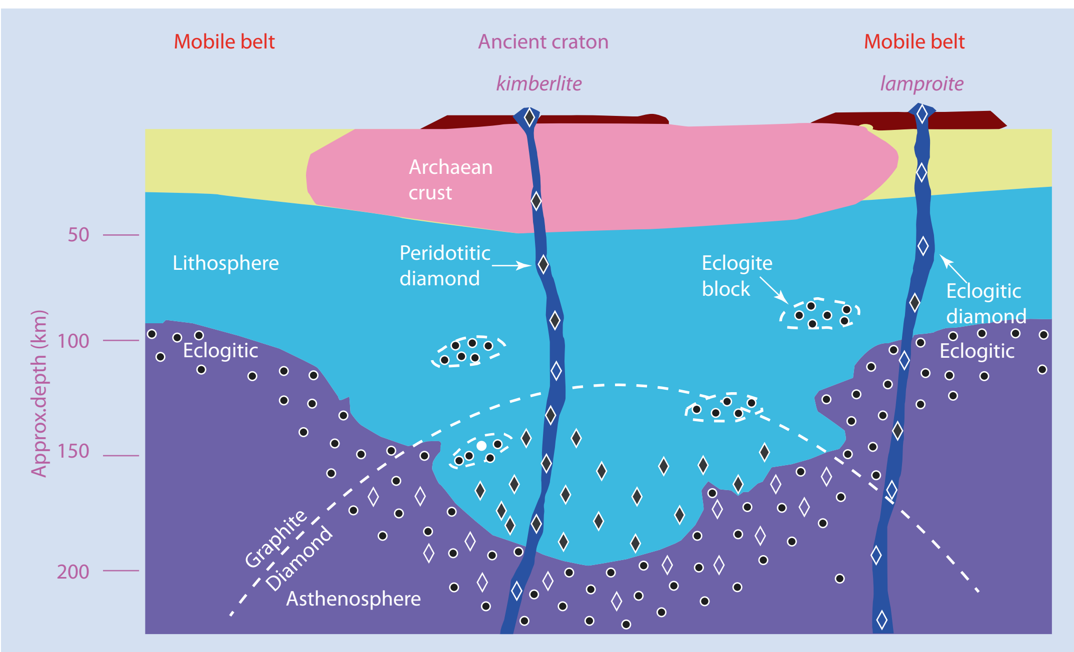


Fig. 2.35 Origin of diamonds (Illustration courtesy of De Beers)

Diamond ore deposits are confined to a minority of the volcanic sources and to secondary deposits derived from them (Gurney et al. 2010). Since diamonds only form beneath old, stable, and thick parts of the Earth's crust, this greatly restricts the global distribution of primary deposits.

Although diamond deposits are often very low grade, the value of the individual diamonds makes the overall deposit highly valuable. Thus, diamond deposits represent some of the highest value mines globally. Grade values in diamond deposits commonly range from 0.25 to 1.5 carats/t, carat being the measurement of weight in diamonds (1 carat equals 0.2 g). The value of the diamonds can be very variable depending on its size, shape, color, and quality. Large, equidimensional, colorless, and clear diamonds without defects are most highly valued. Therefore, the common measure used to assess the economic potential of a deposit is a combination of the grade of the deposit (carats per ton) and the dollar value per carat of the diamonds in the deposit (Stevens 2010). Diamond in kimberlite is probably the best-known type of magmatic mineral deposit (◘ Box 2.7: Diamond in Kimberlite).

Box 2.7

Diamond in Kimberlite

Diamond is one of the most sought-after gemstones on Earth. They are formed mainly in the Earth's lithosphere where pressure conditions are appropriate for carbon to crystallize as diamond, and they are brought to the surface, mostly through the eruption of alkaline igneous rocks. Following the discovery of diamonds in river deposits in central South Africa in the mid-nineteenth century, it was at Kimberley where the volcanic origin of diamonds was first recognized. These volcanic rocks, that were named «kimberlite,» were to become the cornerstone of the economic and industrial development of Southern Africa. Thus, the name of the rock comes from the town of Kimberley in South Africa, where the discovery of a diamond called «the Star of South Africa» in 1869 spawned a diamond rush and creating the Big Hole (◘ Fig. 2.36). It is claimed to be the largest hole excavated by hand. Early mining of the kimberlites around Kimberley was a chaotic business with many claim holders digging small individual claims of 10 by 10 m. Later, as mining reached deeper levels and became more difficult, claims were consolidated into numerous companies. In 1888 De Beers Consolidated Mining Company was created, and this company consolidated all mining operations under the one company, thereby creating the leading diamond producer in the world for the next 90 years.

A variety of mantle-derived igneous rocks comprise the primary sources of diamond, with the principal hosts being kimberlite and lamproite. Kimberlite is a special type of ultramafic magma and derived from the Earth's mantle at more than 140 km depth. Lamproite, a rock type similar to kimberlite, can also contain commercial diamond deposits. All kimberlite-hosted diamond mines which exist in the world, like in south-central and Southern Africa, western Africa, Canada, China, Russia, and the USA, are located in Archean continental blocks. They are virtually restricted to ancient (>2.4 Ga) cratons and the younger (>1.0 Ga) accreted belts of cratonized regions that are underlain by cratons (the ages of kimberlites range from Proterozoic to Tertiary). Diamonds in economic deposits are estimated to be mainly (99%) derived from subcontinental lithospheric mantle (Gurney et al. 2010). In both kimberlites and lamproites, diamonds range in size from microcrystals smaller than 50 microns to macrocrystals occasionally over l cm in size. It is important to remember that most kimberlites and lamproites contain no diamonds. In fact, diamonds are a very minor xenocryst component (<5 ppm) in even the richest ore bodies. Of the approximately 1000 individual kimberlite intrusions known in South Africa, only about 50 carry significant quantities of diamonds. Of these, many are considered subeconomic either because the quantity or quality of the diamonds or the quantity of ore is insufficient. The presence and quality of diamonds in a kimberlite can only be determined with confidence by the collection and processing of a large and representative sample.

The typical diamond deposit is pipe- or carrot-shaped with a circular surface diameter of 50–500 m and a depth extent of several hundred to one thousand meters or more. The distribution of the diamond xenocrysts is variable in the whole host rocks, and the concentration has a level of less than 0.01–2.0 ppm. Strictly speaking, diamonds in kimberlites are not truly magmatic. Kimberlite magma is merely a vehicle that transports the diamonds rapidly to the surface under conditions that prevent them from reverting to graphite, their unattractive low-pressure polymorph. Diamonds remain hidden, unless they are picked up by «younger» kimberlites, lamproites, or other magmatic rocks originated either within or below the mantle source region and intruding fast enough for the diamonds to survive transport to the surface or near-surface emplacement site. Probably, kimberlites move to the surface through the mantle at velocities of 10–30 km/h by crack propagation processes.

Fig. 2.36 Kimberley mine in South Africa (The Big Hole) (Image courtesy of De Beers)

The famous diamond deposits at India and Borneo were the only diamond producers until the eighteenth century. Some big and famous diamonds such as Koh-i-Noor or the Great Mogul Diamond were obtained from these countries. Several decades ago, almost all diamond mines were located in Southern Africa, but large and important deposits have been found and mined in Russia, Australia, and Canada. Examples of these deposits are the Mir pipe in Yakutia (Russia), perhaps the most diamond-bearing kimberlite pipe in the world, which contains only one part of diamond per every one and half million parts of kimberlite; the diamond-bearing kimberlites of Africa (Angola, Botswana, Lesotho, Sierra Leone, South Africa, Swaziland, Tanzania); the diamond deposits in Australia (Western Australia); and the kimberlite pipes in Canada (NWT). Secondary diamond deposits such as placer deposits are formed from these primary source kimberlite rocks by weathering and transportation. The resulting deposits are commonly very rich in high-quality diamonds. Examples include those of the Ural Mountains, the marine deposits of Namibia, and the alluvial deposits of West Africa, Brazil, and Venezuela. These deposits have supplied about 90% of the world's diamond output.

Fig. 2.37 Lithium pegmatites (Salamanca) (Image courtesy of Carlos Villaseca)

Pegmatite Deposits

Another type of magmatic ore deposit is found in pegmatites. In these rocks, metals like lithium, beryllium, boron, tin, niobium, thallium, and the rare earth elements are mined. Pegmatites are formed by the crystallization of melts expelled from granitic magmas. Pegmatitic rocks are very coarse-grained basement rocks abundant in quartz, feldspar, and/or mica, in places endowed either with megacrystals of the aforementioned rock-forming minerals or rare-element minerals. «Apart from the size of their crystals, it is the varied spectrum of rare elements and the significant number of extraordinary minerals resultant from these elements, which renders these crystalline rocks so different from granitic rocks» (Dill 2015).

Most pegmatites show a paragenesis of orthoclase, microcline, albite, mica, quartz, and common minor minerals including topaz, tourmaline, cassiterite, beryl, and lithium. Granite pegmatites occur in the form of dikes, oval, and lenticular bodies, being homogeneous (without a change of mineralogy or texture from wall to wall) and isotropic or strikingly inhomogeneous and anisotropic (zoned or complex pegmatites). Most pegmatite bodies are relatively small with a thickness that rarely surpass tens of meters and a length of a few hundred meters (Pohl 2011), but with increasing industrial request for high-technology metals such as lithium and the rare earth elements (Arndt and Ganino 2012).

Pegmatites may host many useful raw materials: ores of Be, Li (Fig. 2.37), Rb, Cs, Ta > Nb, U, Th, REE, Mo, Bi, Sn, and W; the industrial minerals muscovite, feldspar, kaolin, quartz, spodumene, fluorite, and gemstones; as well as rare mineral specimens (emerald, topaz, tourmaline, ruby, etc.) (Linnen et al. 2012), the complex-type pegmatites of the lithium-cesium-thallium (LCT) family being an important class of rare-element pegmatites. The NYF pegmatites are a different family of pegmatites and are enriched in niobium, yttrium, and fluoride. Their current economic importance is much less than that of the LCT family, but these pegmatites could be a source in the future for rare earth elements and other strategic metals.

Pegmatites of the LCT family were emplaced in orogenic hinterlands intruding metasedimentary rocks, typically at low-pressure amphibolite to upper greenschist facies, the largest deposits being Archean in age. Giant deposits of these pegmatites include Tanco in Canada (2.1 Mt at 0.215% Ta_2O_5), Greenbushes in Australia (70.4 Mt at 2.6% Li_2O), and Bikita in Zimbabwe (12 Mt at 1.4% Li_2O) (Bradley and McCauley 2013). On the other hand, NYF pegmatites are also sometimes REE-enriched pegmatites. Traditionally, the vast majority of this kind of pegmatites has been exploited for their major mineral content: feldspar, quartz, and muscovite. Studies of REE-enriched granitic pegmatites as a whole lag severely behind those of LCT pegmatites in terms of classification, numbers and detail of field descriptions, and mineral compositional data (Ercit 2005).

2.8.2 Hydrothermal Ore Deposits

They represent an essential group of ore deposits because they are the source for most of the metal production of the world. Hydrothermal deposits provide almost 100% of lead, zinc, molybdenum, and silver and 60–90% of copper, gold, and uranium as well as gemstones and industrial materials such as clay minerals and quartz. Hydrothermal deposits are quite different, being located in a broad rank of geological and tectonic settings: some of them are closely linked with granitic rocks, others form on the ocean floor, and still others are in sedimentary basins; all the deposits have common origin via the precipitation of metals or ore minerals from hot aqueous fluids (Arndt and Ganino 2012). The main examples of hydrothermal ore deposits include (1) porphyry deposits, (2) volcanogenic massive sulfide (VMS) deposits, (3) sedimentary exhalative (SEDEX) deposits, (4) iron oxide-copper-gold deposits, and (5) gold deposits.

Porphyry Deposits

Porphyry copper deposits (PCD) are large, low- to medium-grade, Cu ± Au ± Mo hydrothermal deposits related to igneous intrusions, being the largest source of the world in copper and a major source of molybdenum, gold, and silver. Despite relatively low grades, PCDs have significant economic impact due to their large size (commonly hundreds of millions to billions of metric tons), long mine lives (decades), and high production rates (billions of kilograms of copper per year). With incrementing molybdenum/copper ratio, these deposits are transitional to low fluorine (quartz monzonite type) porphyry molybdenum deposits; with incrementing gold/copper ratio, they are transitional to porphyry gold deposits (John et al. 2010). Thus, it is common to describe several subtypes of porphyry deposits according to the dominant metal: porphyry Cu, porphyry Cu-Au, and porphyry Cu-Mo.

Porphyry copper deposits are constituted by disseminated copper minerals in veins and breccias and form high tonnage (greater than 100 million tons) and low- to moderate-grade (0.3–2.0% copper) mineral deposits. In contrast to VMS deposits (see next section), which normally are small (1–5 Mt) but of high grade (3–10% ore metals), porphyry deposits are enormous but of low grade. These deposits were the first group of metallic mineral deposits mined by large-scale, open-pit methods in the early twentieth century. The best-known deposits are in the cordilleras of North and South America, the Bingham mine in the USA (2.7 billion tons of ore grading, 0.7% Cu and 0.05% Mo), and the Chuquicamata mine in Chile (11 billion tons of 0.56% Cu and 0.06% Mo) (Arndt and Ganino 2012) (Box 2.8: Chuquicamata Copper Mine (Chile)).

Box 2.8

Chuquicamata Copper Mine (Chile)

Chile is known worldwide as the site of one of the largest copper concentrations on Earth. Thus, Chuquicamata mine (Fig. 2.38), property of Codelco, is one of the largest open-pit copper mines and the second deepest open-pit mine in the world (popularly known as Chuqui). The name comes from indigenous communities, «Chuquis, » who lived in the area and obtained native copper. The open-pit measures are 5 km large, 3 km width, and 1 km deep forming an ellipse. Chuquicamata mine began open-pit mining in the year 1915 although its mining properties had been known for centuries by the pre-Hispanic cultures. In 1971, the mine was nationalized, and management and operation were taken over by the Corporación Nacional del Cobre-Chile (Codelco). At the end of the year 2005, it had mined out about 2.6 billion tons of copper ore with a mean grade of 1.53%, reaching a pit depth of 850 m. The Chuquicamata mine complex is located 1650 km north of the Chilean Capital city (Santiago), at 2870 m above sea level. Chuquicamata produces electrorefined and electrowon cathodes having a purity of 99.99% copper. It also produces fine molybdenum, as well as other by-products, such as anode slimes and sulfuric acid.

The Chilean cordillera contains 9 of the 16 giant porphyries along the circum-Pacific belt. Chuquicamata lies in the Precordillera of northern Chile, which is parallel and west of the volcanoes that form the modern continental arc of the Andean Cordillera. The Chuquicamata mine lies on the Chuqui porphyry complex, a north-northeast trending, elongated, tabular, intrusive complex

Fig. 2.38 Chuquicamata mine (Image courtesy of Codelco)

that measures 14 km × 1.5 km. Virtually the entire ore deposit at Chuquicamata is hosted by and related to this 36–33 Ma porphyry complex that comprises a number of phases, many of which do not have well-defined contacts. The porphyry copper ore body is rectangular in plan and dips vertically, being the zone's porphyries largely affected by potassic alteration. The great majority of the mineralization at Chuquicamata occurs in veins and veinlets, the earliest of which are quartz and K-feldspar veinlets with little or no sulfides. These are cut by more continuous quartz veins ranging up to 5 cm in width with molybdenite and traces of chalcopyrite. The next generation is the pyritic main stage veins which carry pyrite, chalcopyrite, bornite, and digenite. The final phase of mineralization is represented by a partly preserved leached cap and extensive oxide ore that replaces an upper chalcocite blanket which overlies a high-grade supergene blanket that persists to nearly 800 m below surface in the zone of fault brecciation and pervasive pyritic main stage quartz-sericite alteration.

Finally, Chuquicamata underground mine is a structural and strategic project that represents an important part of Codelco's future and which considers transforming the world's largest open-pit mine into a gigantic underground operation. This new underground mine is being developed to access the ore body situated beneath the present open-pit mine because currently the mine is producing 400,000 tons of waste rock, and since it increases the cost and distance that must be reached to find mineralization, it generates higher costs. The geological data from drillholes indicate that below the final pit bottom, there are about 2.3 billion tons of ore with a mean copper grade of 0.81%. The project involves ore extraction by macro blocks and block caving in an underground mine at depths of 1300–1800 m. The underground mine, scheduled to begin operations in 2020, will comprise four production levels, a 7.5 km main access tunnel, five clean air injection ramps, and two air-extraction shafts.

The mineralization in the porphyry deposits consists of disseminated small concentrations of sulfide minerals in the highly altered upper portions of the intrusion and in surrounding rocks. Closely associated with the mineralization is a moderate to intense alteration that shows a zoning concentric about the intrusion. This alteration also span outside the zone of mineralization, and it is commonly utilized as a guide during the exploration of this type of deposits. Most PCD deposits are located within felsic to intermediate igneous intrusions and in the country rocks that surround the intrusion. Original sulfide minerals are pyrite, chalcopyrite, bornite, chalcocite, and molybdenite.

Gold is often in native form and is found as tiny blobs along borders of sulfide crystals, or it occurs in sulfosalts like tetrahedrite. Molybdenite distribution is variable, and radial fracture zones outside the pyrite halo may contain lead-zinc veins with significant gold and silver contents. In deposits with an extensive supergene enrichment zone developed in the upper parts of the deposit, copper oxide minerals and native copper may be present.

In many districts, plutons and batholiths that host the mineralization are older and not related to the ore-forming system, although they can be part of long-lived magmatic successions. In other districts, they are only slightly older and range from multiple large stocks to composite batholiths (John et al. 2010). The regional, deposit-scale, and local-scale environments of porphyry copper can be very varied. They are widespread but mostly localized in time and space through the evolution of magmatic arcs along convergent plate margins where subduction of oceanic crust and arc-type magmatism generates hydrous, oxidized upper crustal granitoids genetically related to ores. It is possible that many porphyry copper deposits are formed during unusual periods of subduction. Deposits have formed throughout most of Earth's history, but because they generally form in the upper crust (less than 5–10 km depth) in tectonically unstable convergent plate margins and are prone to erosion, more than 90% of known deposits are Cenozoic or Mesozoic in age.

PCDs are thought to derive from hydrothermal fluids generated near the top of a cooling magma body at depths between 1 and 5 km (Stevens 2010). The close spatial and temporal association between the ore bodies and granitic intrusions is clearly indicative that magmas are directly linked to the ore-forming process. Porphyry copper systems are mainly formed by magmatic fluids that were released during shallow emplacement of porphyritic granitoid stocks. The fluids create a fracture network in the rocks as they travel, thereby producing the characteristic stockwork texture of this type of deposits. The ore minerals crystallize out of the hydrothermal fluids as a result of cooling of the fluid as it moves away from the magma body. Thus, formation of porphyry copper deposits, as John et al. published in 2010, «involves a complex series of processes including magma generation, differentiation, emplacement, crystallization and degassing, high-temperature reactions between degassed fluids and meteoric and other non-magmatic waters, and near-surface reactions between low-temperature meteoric water and earlier formed, high-temperature minerals; external saline waters such as sedimentary brines were probably involved in the earlier stages of evolution of some porphyry copper systems, resulting in sodic and sodic-calcic alteration».

Volcanogenic Massive Sulfide (VMS) Deposits

This type of deposits is referred to as volcanogenic massive sulfide (VMS) although similar terms have been used: volcanic massive sulfide, submarine exhalative massive sulfide, and volcanic-hosted massive sulfide, among many others. More recently, the term polymetallic massive sulfide deposit has been also applied by many authors to VMS mineralization on the modern seafloor that contains significant quantities of base metals. Volcanic massive sulfide deposits are small- to medium-sized, moderate- to high-grade Cu ± Zn ± Pb ± Au ± Ag hydrothermal deposits hosted in volcanic and/or sedimentary rocks. They are significant sources of copper and zinc and, to a lesser extent, lead, silver, gold, cadmium, selenium, tin, bismuth, and minor amount of other metals. The polymetallic and sometimes high-grade character of the deposits make them a preferential target for exploration. As in the case of the porphyry deposits, there are several subtypes of VMS deposits depending on the dominant metal and host rocks.

VMSs are among the best understood of all ore deposits due to the ore bodies that are relatively simple, both in their structure and their composition and mineralogy, and they have been studied intensively over the last decades. They are one of very few deposits whose formation, by way of precipitation of sulfides at or just below the ocean floor, can be observed directly – black smokers. VMS deposits were among the first ever to be mined because this mineralization was mined in Cyprus and in Spain more than 2000 years ago, providing much of the copper utilized in the weapons of Roman centurions. The old Rio Tinto mine in southwestern Spain has one of the world's longest known mining histories with copper having been mined there even before Roman times (Box 2.9: Rio Tinto Copper Mine (Spain)). This mine was the foundation stone for the mega mining company that still bears its name. Rio Tinto has subsequently gone on to become one of the world's biggest diversified mining companies.

2

Box 2.9

Rio Tinto Copper Mine (Spain)

The Iberian Pyrite Belt is located in the SW of the Iberian Peninsula, comprising part of Portugal and of the provinces of Huelva and Sevilla in Spain, being one of the most important volcanogenic massive sulfide districts in the world. Río Tinto mine is located at the eastern end of the Iberian Pyrite Belt. Within the Pyrite Belt, there are eight major mining areas, each thought to contain more than 100 million tons or ore. These are from east to the west: Aznalcóllar-Los Frailes, Río Tinto, Sotiel-Migollas, La Zarza, Tharsis, Masa Valverde, Neves Corvo, and Aljustrel. Río Tinto mining area is the largest of these and includes two big open-pit mines: Cerro Colorado (◘ Fig. 2.39) and Corta Atalaya. The high geological interest of this mining district is because it is most probably the biggest sulfur anomaly on the Earth's crust, with original tonnages around the 2500 million tons of mineralized rock in different degrees. In fact, the Cerro Colorado deposit contained one of the largest known concentrations of sulfides in the world. The name of Río Tinto mines comes from Río Tinto river; in turn, it takes the name from its red color.

Río Tinto mines have a very long history, dating back to pre- Iberian times; then came the Iberians, including Tartessian, the Phoenicians, the Carthaginians, the Romans, the Moors, the Spaniards, and the British. It is believed that copper was first recovered from the ores in the third millennium BC and that silver was mined in the late Bronze Age, ninth to twelfth centuries BC onward. From Tartessian to Romans, mineralizations were mined actively, but little mining was done after the departure of Romans. After several centuries of some mining activities in the region, the British arrived to Río Tinto at 1873 (The Rio Tinto Company Limited was registered in London in March of this year). It seems that the purchase price of the mines was 92,800,000 pesetas equal to 3,600,000 sterling pounds. The Rio Tinto Company continued mining and smelting in Spain through two world wars and a civil war, until 20 June 1954 when two-thirds of its Spanish assets were sold to a Spanish group of bankers. Then, different owners lead Río Mines to closure, and in the last 10 years different efforts have been made to reopen the mines. Thus, open-pit mine and processing facilities have been on care and maintenance since mining ceased in 2000 due to low copper prices of less than $1.00/lb at the time. EMED Mining (actually Atalaya Mining) was granted an option to acquire the operation in May 2007, and it was subsequently acquired in October 2008. New commercial production will begin shortly. In summary, few mines in the world have such a history as Río Tinto mines.

Most of the mineral deposits in this area consist of massive sulfides within the Volcano-Sedimentary Complex. The Río Tinto massive sulfide (pyrite-chalcopyrite) deposit occurs on the transitional contact between a lower mafic volcanic unit composed of andesitic and spilitic pillow lavas and dolerite sills intercalated with bands of slate and chert of Lower Carboniferous age and the overlying felsic volcanic unit. It is composed of rhyolite lavas and pyroclastic rocks formed by submarine volcanic activity in the Lower Carboniferous period about 320 million years ago. Río Tinto is a textbook example of the volcanogenic massive sulfide (VMS) type of deposits.

Overall, massive sulfides display the typical structure of volcanogenic massive sulfide deposits: a lens of massive sulfides overlays a wide zone with rocks affected by an important hydrothermal alteration. In its core, there is a network («stockwork») of sulfide-rich veins considered as the zone that channeled hydrothermal fluids on their way out to exhalation at the

◘ **Fig. 2.39** Cerro Colorado open-pit (Spain) sea bottom or a favorable level. The mineralization in Río Tinto is found either as dissemination or small veins in the stockwork areas within volcanic rocks and slates, or as massive sulfide lenses lying atop or included in the stockwork zones, or in gossan areas representing the supergenic alteration of massive sulfides, sometimes up to 70 m thick. What makes Rio Tinto different from other districts in the Pyrite Belt is the fact that the massive sulfides seem to be formed in two different environments. On one side, the mineralizations in Southern Lode and Planes – San Antonio – are hosted in slates and have sedimentary structures, suggesting they were formed by exhalative processes in the sea bottom. However, the mineralizations in the Northern Vein hosted by dacite have a coarser grain and always display replacement structures with the hosting dacite. The mineralization was supergenically altered and eroded during the Cenozoic. Originally, there was a gossan (Cerro Colorado) of 10–70 m deep mined between 1974 and 2002 together with the copper of the underlying stockwork. The gossan was rich in Au, Ag, Pb, Sb, and Bi and poor in Cu and Zn.

Volcanogenic massive sulfide deposits are stratabound concentrations of sulfide minerals precipitated from hydrothermal fluids in extensional seafloor environments. The term volcanogenic implies a genetic link between mineralization and volcanic activity, but siliciclastic rocks dominate the stratigraphic assemblage in some settings. Relation to volcanoes ranges from proximity to quite tenuous connections to volcanism, as in parts of the Southern Iberian Pyrite Belt (Pohl 2011). VMS deposits are hosted in volcanic rocks dominated by basalt. There are usually important felsic volcanic and sedimentary rock layers closely associated with the deposit, and small intrusive igneous rock bodies are often located beneath the deposits (Stevens 2010).

The deposits are formed by two parts: a concordant massive sulfide lens (>60% sulfide minerals) and discordant vein-type sulfide mineralization, commonly called the stringer or stockwork zone. Individual lenses are 2–20 m thick and extend for tens to hundreds of meters laterally. Large lenses can reach more than 100 m thick and extent for more than 1000 m laterally. They show different mineralization textures such as breccias, layering, and laminations. The deposits are characterized by abundant Fe sulfides (pyrite or pyrrhotite normally comprises more than 80% of the minerals in the massive sulfide bodies).

VMS deposits are derived from hydrothermal fluids that circulate through a sequence of volcanic rocks and exit on the seafloor as a plume of metal-rich fluids. They encompass a wide variety of geodynamic and more local genetic settings. Thus, «the main tectonic settings include mid-oceanic ridges, volcanic arcs (intraoceanic and continental margin), back-arc basins, rifted continental margins, and pull-apart basins; the composition of volcanic rocks hosting individual sulfide deposits range from felsic to mafic, but bimodal mixtures are not uncommon and the volcanic strata consist of massive and pillow lavas, sheet flows, hyaloclastites, lava breccias, pyroclastic deposits, and volcaniclastic sediments; a zonation of metals within the massive sulfide body from Fe + Cu at the base to Zn + Fe ± Pb ± Ba at the top and margins characterizes many deposits» (Koski and Mosier 2012). Deposits range in age from Early Archean (3.55 Ga) to the present, and significant occurrences of VMS mineralization are found in greenstone belts of almost all Precambrian shield areas. Of particular importance are the Archean and early Proterozoic greenstone belts of the Canadian Shield, the Lower Paleozoic volcanic belts of the Caledonides in Scandinavia and the northern Appalachians of Newfoundland (Canada), the Upper Paleozoic Iberian Pyrite Belt extending from southern Portugal to southern Spain, and the Miocene Green Tuff Belt of Japan (e.g., Kuroko sulfide deposits) (Misra 2000).

Sedimentary Exhalative (SEDEX) Deposits

Almost 100 million tons of sediment containing 2% Zn, 0.5% Cu, and significant amount of Au and Ag have precipitated from hot dense brine that accumulated in the «Atlantis II Deep, » a 10 km diameter depression on the floor of the Red Sea. It would constitute a very attractive ore body of the type referred to as a SEDEX or sedimentary exhalative deposit. SEDEX deposits are medium to large sizes, moderate to high grade, Zn ± Pb ± Ag

Fig. 2.40 McArthur River Mine (Australia) (Image courtesy of Glencore)

hydrothermal deposits hosted in sequences of sedimentary rocks. Another example is the Salton Sea, a big and shallow lake in southern California that originated in 1905 where a canal transporting water from the Colorado River breached and flooded a salt pan (Arndt and Ganino 2012). The two processes recorded in Red Sea and Salton Sea examples are essential elements to understand the formation of SEDEX deposits.

The term SEDEX, derived from «sedimentary exhalative» (Carne and Cathro 1982), is based on the interpretation that the finely laminated or bedded sulfide ores represent chemical sediments precipitated from hydrothermal fluids exhaled onto the seafloor. Examples of SEDEX deposits are Broken Hill, Mount Isa, and McArthur River in Australia (Fig. 2.40), Sullivan in Canada, and Navan in Ireland. These types of deposits are the world's largest source of zinc and a major source of lead. SEDEX deposits are on average an order of magnitude bigger than VMS deposits (Stevens 2010). The dominant economic minerals are sphalerite and galena, and in some deposits chalcopyrite, with silver primarily contained with galena. SEDEX deposits range in age from 2 billion to 300 million years old, deposits younger than 300 million years not being documented. The deposits are characterized by moderate to high grades of zinc and lead, and copper is an important by-product in some deposits.

SEDEX Pb-Zn-Ag deposits are hosted in marine sedimentary rocks in intracratonic or epicratonic rift basins. The distinguishing features of an idealized, unmetamorphosed, or mildly metamorphosed SEDEX deposit may be summarized as follows: (a) mineralization hosted mostly either by marine, clastic sediments of continental derivation, typically pyritic and carbonaceous shales, or by platform carbonate rocks, with thin (1–10 cm thick) tuff horizons in the sedimentary sequence in some cases; (b) a zone of stratiform mineralization consisting of stacked lens-like, concordant, tabular bodies of low-Cu massive sulfide ore; (c) a footwall zone of minor stockwork and vein-type sulfide mineralization associated with hydrothermal alteration (predominantly silicification); (d) a distinct (Cu)-Pb-Zn-(Ba) lateral zoning of hydrothermally precipitated minerals and a less systematic (Cu)-Zn-Pb-(Ba) vertical zoning; (e) spatial association with a synsedimentary fault system that was active at the time of mineralization and may have been reactivated during later deformation; and (f) a general lack of demonstrable magmatic affiliation of the host rocks or of mineralization (Misra, 2000).

The main ore minerals of SEDEX deposits, sphalerite and galena: «were probably precipitated on or just below the sea floor from warm 100–200 °C, saline -10–30% total dissolved solids- basin brines that ascended along basin-controlling synsedimentary faults; deposition and sequestration of metals occurred by precipitation of sulfide minerals as a result of mixing of metal-transporting brine and locally derived H_2S produced by bacterial and perhaps thermochemical reduction of local seawater sulfate» (Emsbo 2009). They are formed in a similar manner to VMS deposits, although there is little if any involvement of igneous rocks in their formation.

Iron Oxide-Copper-Gold (IOCG) Deposits

An iron oxide-copper-gold deposit can be defined as a breccia-hosted polymetallic mineral deposit in which mineralization is spatially and temporally associated with granite and with iron oxide alteration. The Olympic Dam deposit in Australia is probably the best example in the world for this group of deposits; other typical examples are Kiruna in Sweden and Bayan Obo in China. IOCG deposits range in age from recent to Precambrian, and such deposits commonly show (1) Cu, with or without Au, as economic metals; (2) hydrothermal ore styles and strong structural controls; (3) abundant magnetite and/or hematite; (4) Fe oxides with Fe/Ti greater than those in most igneous rocks and bulk crust; and (5) no clear spatial associations with igneous intrusions as, for example, displayed by porphyry and skarn ore deposits (Williams et al. 2005). IOCG deposits are found in a wide range of rock types (sedimentary, volcanic, and igneous), and common forms of mineralization include breccia zones, tabular bodies, veins, stockworks, volcanic pipes, and disseminated mineralization. Hydrothermal iron oxide-copper-gold deposits can include mainly various combinations of metals such as Fe, Cu, Au, Ag, U, Th, F, Co, Bi, W, and rare earth elements (REE).

The supergiant Olympic Dam Cu-U-Au-Ag ore deposit of South Australia has the largest uranium resource and the fourth largest copper and gold resource in the world. The tectonic setting includes a hydrothermal breccia complex surrounded by Mesoproterozoic granite, the breccia being composed mainly of granite clasts and minor amount of Mesoproterozoic volcanic clasts. Very thick (>350 m) sections of bedded sedimentary facies occurring in the breccia complex include laminated to very thin planar mudstone beds, thin to medium internally graded sandstone beds, and thick conglomerate beds. Lateral continuity, provenance characteristics, great thickness, below-wave-base lithofacies, and intracontinental setting suggest that these bedded sedimentary facies are remnants of a sedimentary basin that was present at Olympic Dam prior to formation of the breccia complex (McPhie et al. 2011).

Due to the very recent discovery of the deposit type, theories of ore formation are subject to continual revision. According to Williams et al. (2005): «most theories call on large-scale magmatic events that drive large-scale flow of oxidized probably magmatic hydrothermal fluids into mid to upper crustal levels along fault zones; mixing of these fluids with near surface meteoritic fluids or brines is commonly invoked as the ore-forming process and brines and metals may be sourced directly from underlying magmas, indirectly by interaction of magmatic fluids with country rocks or other fluids, or independently through modification of basinal or metamorphic fluids». However, although the Olympic Dam breccia complex and ore body are primarily hydrothermal in origin, the Olympic Dam hydrothermal system would have had access to and interacted with the overlying sedimentary succession, so this succession should be considered as an additional source of both fluids and metals (McPhie et al. 2011).

Gold Deposits

Trace amounts of gold are present in a wide variety of mineral deposits, ranging from <0.01 ppm Au in Mississippi Valley-type deposits to concentrations in some sulfide deposits high enough to be recoverable as a by-product; main types of ores that routinely produce by-product gold are Ni-Cu sulfide ores associated with mafic and ultramafic rocks, VMS ores, and Cu ores of porphyry copper deposits. However, most of the important gold deposits belong to one of the following seven types: (a) young placer deposits, (b) deposits hosted by quartz-pebble conglomerates (Witwatersrand type), (c) volcanic-associated epithermal deposits, (d) sediment-hosted, disseminated deposits (Carlin type), (e) deposits hosted by banded iron formations, (f) intrusion-related deposits, and (g) lode deposits (Misra 2000). Type (c) is possibly the most important gold type deposit of hydrothermal affiliation. Other essential types for gold extraction (e.g., Witwatersrand type) are also described below.

Volcanic-associated epithermal gold deposits got the name «epithermal» according to the classification of Lindgren (1913), who coined this term for deposits that form from hydrothermal fluids at shallow crustal levels, occurring in a variety of structural settings. They are commonly associated with subduction-related calc-alkaline to alkaline arc magmatism as well as back-arc continental rift magmatism. Because of their relatively higher grades and amenability to cheaper open-pit mining and heap-leach extraction of gold, epithermal deposits have been a favored target of exploration since the early 1970s.

The main distinguishing characteristics of epithermal gold deposits are the following:

1. Andesitic volcanic and pyroclastic rocks are the more common host to ore (early to late Tertiary).
2. The deposits formed in extensional tectonic settings, in zones with well-developed tension fracture systems, and in normal faults that could channel hydrothermal fluids and localize mineralization.
3. (c) The mineralization is epigenetic and occurs commonly in the form of quartz veins.
4. Ore and associated minerals occur dominantly as open-space fillings, producing characteristic banded and crustiform textures.
5. Gold and silver are the principal economic metals; main ore minerals in the veins are native gold and silver, electrum, argentite, Ag-bearing As-Sb sulfosalts, and Au-bearing pyrite; associated base metal sulfides, which are generally concentrated below the precious metal horizon, include sphalerite, galena, and chalcopyrite.
6. Quartz and calcite are the most abundant gangue minerals in the veins.
7. Hydrothermal alteration of wall rock is a characteristic feature of all epithermal precious metal deposits (Misra 2000).

Vein Deposits

The most convincing examples of hydrothermal deposits are vein systems discordant to stratification or lithologic boundaries in host rocks (Box 2.10: Panasqueira Tungsten Mine (Portugal)). They represent dominantly open-space filling of structurally controlled fractures and faults. Some vein-type deposits are believed to be genetically related to exposed or buried igneous (especially felsic) intrusions because fluid inclusion and isotopic data provide evidence for a major contribution of magmatic water in the ore-forming fluids. Ore fluids for other types of vein deposits may have been dominated by magmatic water, meteoric water, or basinal brines.

Box 2.10

Panasqueira Tungsten Mine (Portugal)

The first prospecting license was granted in 1886 and the first reference to wolframite mineralization in the Panasqueira area reportedly dated to 1888, but probably Panasqueira mines were first worked for tin by the Romans and next by the Moors. The mining company was founded in 1896 to mine tungsten at Panasqueira as the industrial uses of the commodity were first being developed throughout the world. All the individual concessions were grouped into one single mining area known as the «Couto Mineiro da Panasqueira.» During the period 1947–2014, a total of approximately 31 million tons of rock have been mined that has produced approximately 111,123 tons of tungsten concentrate, 5383 tons of tin concentrate, and 31,702 tons of copper concentrate. Today Panasqueira is still one of the biggest tungsten mines in the world. Mining at the Panasqueira mine has evolved from labor-intensive hand operations in the early 1900s through mechanized longwall methods to the mechanized room and pillar operation currently used and based on an analysis of geological and geomechanical characteristics of the rock mass. This mining method is possible in part due to the very competent host rock, and underground rock support is rare.

Panasqueira mine is a world-class W-Sn-Cu vein-type deposit, located in the Central Iberian Zone of the Palaeozoic Iberian Massif (Portugal), which is one of the most important metallogenic provinces of Europe. The Central Iberian Zone is composed of a thick sequence of flysch-type units primarily composed of graywackes, shales, and schists of late Precambrian to Cambrian age. Intruding this flysch sequence are the Epi-Hercynian synmetamorphic muscovite-biotite granites or post-metamorphic biotite-rich granites. The Panasqueira deposit consists of a series of stacked, sub-horizontal, hydrothermal quartz veins intruding into the Beira schists and shales. A second set of non-wolframite-bearing quartz veins (veins contain minor chalcopyrite, galena, and pyrite) also exists at the Panasqueira deposit and is aligned with the vertical foliation and cut by the later tungsten-bearing hydrothermal vein system. Intrusives are an important component of the mineralizing events at Panasqueira. The most dominant and important structural feature at the Panasqueira mine is a flat open joint system prevalent throughout the mine workings. The remobilized ore-bearing fluids migrated from the intrusive to these flat joints to form the stacked quartz vein system.

The dimensions of ore body are 2500 m (length), 400–2200 m (width), and 500 m (depth). The principal tungsten-bearing mineral is wolframite, and by-products include tin (cassiterite), copper (chalcopyrite), and silver. The mineralization is generally coarse grained and very erratically distributed in the quartz veins. Wolframite mineralization occurs as very large nugget-like crystals or large crystal aggregates, usually concentrated toward the margins of the quartz veins or, occasionally, close to the central portion of the quartz veins (Fig. 2.41). The Panasqueira mine is renowned throughout the world for the extraordinary size and quality of the minerals wolframite, apatite, arsenopyrite, cassiterite, and quartz crystals that occur in cavities in the quartz veins. Wolframite crystals of this size are reportedly rare in other tin-tungsten occurrences.

Fig. 2.41 Pillar with a quartz vein in Panasqueira tungsten deposit

Many veins in this type of deposit are developed upward into a fan of thinner veins and veinlets, which resemble a branching tree. Thickness, vertical extent, and horizontal length of veins vary widely. Less than 0.5 m in thickness may allow profitable mining of high-grade gold and silver ore veins (Fig. 2.42), whereas tin and tungsten require a width of 1 m and barite and fluorite a minimum of 2 m width. The distribution of veins in space ranges from horizontal to vertical, although steeply dipping veins are the majority. From a tectonic viewpoint, many veins are associated with large-scale tensional tectonics including rifting and late-orogenic relaxation of orogens. However, veins may also originate during convergent tectonics (Pohl 2011).

Fig. 2.42 Ore sample with visible gold (Image courtesy of Petropavlovsk)

The most important control on vein formation is related to the mechanical properties of the host rocks. Fractures form more readily in competent rocks than in ductile materials. Very brittle rocks such as dolomite or quartzite use to create a network of short fractures instead of spatially separated longer ones. In that case, hydrothermal activity may result in stockwork ore. Stockwork ore bodies consist of numerous short veins of three-dimensional orientation, which are so closely spaced (e.g., 30 veins/m) that the whole rock mass can be mined.

The distribution of ore in veins is usually inhomogeneous, and only a small part of the total vein fill is exploitable. Veins commonly consist of quartz (sometimes of several varieties, e.g., chalcedony). This quartz commonly occurs as interlocking crystals with a great variety of sizes or as finely laminated bands parallel to the walls of the vein. Minor amount of sulfide and other gangue minerals such as calcite and various clay minerals occur. Typical mineral associations in vein deposits are gold with pyrrhotite, gold with arsenopyrite, gold with pyrite, gold with chalcopyrite, gold with minor sulfides (free gold), silver with galena and galena-sphalerite, silver with tetrahedrite, or antimony or copper-arsenic sulfides.

2.8.3 Sedimentary Ore Deposits

Sedimentary mineral deposits are those that form by sedimentary processes. They include placers originated by erosion, transportation, and sedimentation processes as well as deposits related to water infiltration, supergene alteration, and diagenetic processes. The boundary between sedimentation and diagenesis is subtle. Moreover, diagenetic ore deposits can be many times considered as diagenetic/hydrothermal mineral deposits.

Supergene Enrichment Deposits

If the sulfide mineralizations are exposed at the surface of the Earth, it is very common that these minerals become oxidized, the ore metals being leached downward and usually concentrated at the top of the water table. Thus, supergene enrichment is a consequence of near-surface oxidation caused by meteoric water seeping downward through the unsaturated zone. This oxidation process can be very useful if the previous mineralization has a low-grade character. In some cases, the copper grade can increase from 0.8% Cu in the primary ore to 2–3% in the thick layer of supergene enrichment. Consequently, these layers of supergene enrichment contain two to five times more ore metals than the primary ore, and

■ Fig. 2.43 Ambatovy (Madagascar) nickel-cobalt laterite deposit (Image courtesy of Sherritt International Corporation)

they are conveniently located close to the surface where they can be recovered at the start of the mining operation. For sulfide copper and silver ores, iron oxides, and some uranium ore deposits, this process is of economic significance (Pohl 2011). The best-known examples of supergene enrichment zones are perhaps those overlying porphyry copper deposits.

Other example of supergene enrichment deposits is lateritic nickel ore deposits. Nickel-cobalt laterites (■ Fig. 2.43), an important source of nickel, are supergene deposits of Ni ± Co formed from pervasive chemical and mechanical (tropical) weathering of ultramafic rocks, which contain as much as 0.3 percent nickel. Marsh and Anderson (2011) suggest that: «the extreme weathering removes all elements except the least soluble ones from the protolith and the residual material can average as much as 5% nickel and 0.06% cobalt; thus, the enrichment of nickel in the weathering profile is controlled by several interplaying factors that include parent rock, climate, chemistry/rates of chemical weathering, drainage, and tectonics.» In some cases, these deposits can be later subsequently weathered, redeposited, reconcentrated, and probably covered by new sediments.

Residual Deposits

In this type of deposits, the economically interesting component is concentrated in situ while weathering removes diluting parts of the rock. Examples are residual and eluvial placers, bauxite, lateritic gold, platinum, iron and nickel ores, residual enrichment of subeconomic iron and manganese ores, and industrial minerals such as phosphate, magnesite, and kaolin (Pohl 2011). The fundamental geochemical principle of the enrichment is the steady activity of a reaction front in soil while the land surface is lowered by weathering and erosion. At the reaction front, the valuable component is immobilized, and the enrichment is due to retention and accumulation of the component of interest contained in the removed rock and soil volume. An example of this process is eluvial enrichment of phosphate from carbonatites by leaching of carbonate whereas apatite remains in place.

Probably, one of the most characteristic ores of this group is bauxite (■ Fig. 2.44). The purest bauxites form through a combination of processes: (1) the presence of Al-rich (and Fe-poor) parent rocks such as alkali granite, syenite, tuff, or clay-rich sediment and their metamorphosed equivalents; (2) an appropriate balance of temperature

Fig. 2.44 McCoy bauxite mine (Australia) (Image courtesy of Alcoa)

and rainfall (high temperatures favor Fe-rich laterites); and (3) a pronounced alternation of wet and dry seasons (Arndt and Ganino 2012). As a result of these restrictions, the most important bauxite deposits in the world are located mainly in equatorial countries with tropical climates such as Guinea, Australia, Brazil, and Jamaica. In parts of Africa, South and Central America, and Australia, the concentration of Al_2O_3 increases from about 15% in the source rock to close to 60%, the level in rich Al ore (Ardnt and Ganino 2012).

Infiltration Deposits

Infiltration mineral deposits are formed where meteoric water takes up a substance that is dissolved by weathering and it is concentrated after considerable transport by infiltration in a different geological setting. The so-called «roll-front» uranium deposits are the most typical example of this type of deposits. In this deposit, uranium is quickly dissolved from rocks such as granite, gneiss, and felsic tuff and then transported during hundreds of kilometers by rivers, until infiltrating into an aquifer where reduced conditions produce the precipitation and concentration of uranium minerals (uraninite (UO_2) or pitchblende, UO_3, U_2O_5). The critical aspect to the formation of uranium deposits is the great different solubility of uranium in oxidized and reduced fluids. Uranium occurs in two valence states, the reduced form U^{4+} and the oxidized form U^{6+}. The latter is highly soluble in oxidized fluids where it forms stable complexes with fluoride, phosphate, or carbonate ligands; under these conditions, uranium is readily transported in the fluids that circulate along sedimentary basins. Some deposits of metal such as copper, iron, vanadium, silver, and Pb-Z-Ba-F could have a similar genesis (Pohl 2011).

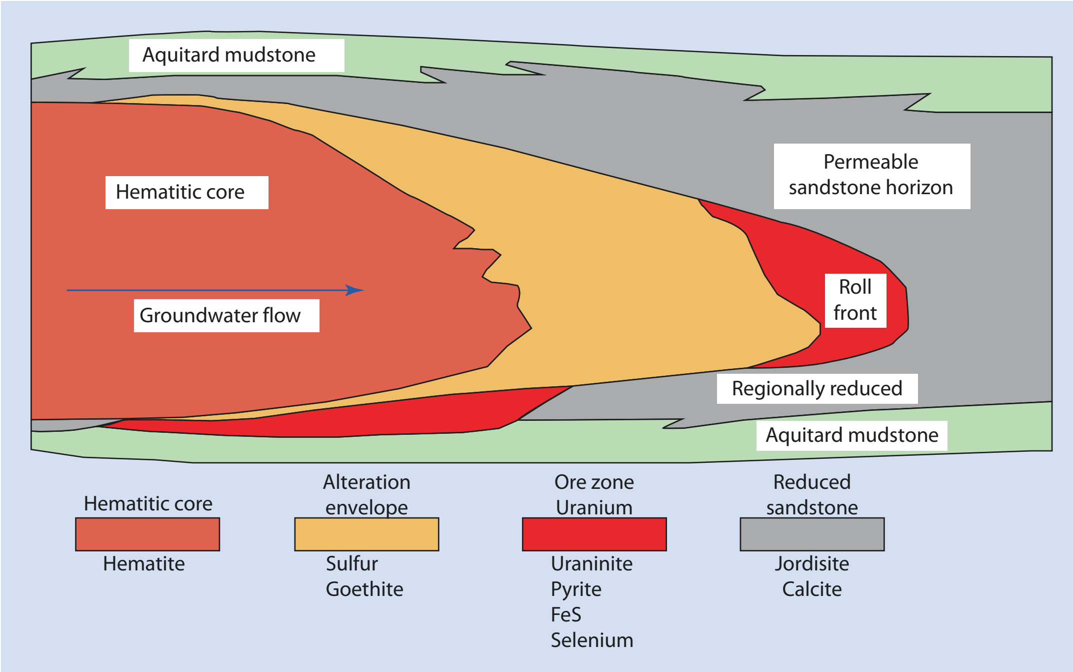


Fig. 2.45 Illustration of roll-front formation

From a geological viewpoint, roll-front uranium deposits host in coarse-grained permeable sandstones, which at depth contain a reduced array of pyrite, calcite, and organic matter. The age of this host sediment ranges from Upper Paleozoic to Cenozoic. In many cases, the sandstone bed is confined above and below by shale or other impermeable rocks (Fig. 2.45). This forces the groundwater to flow through the sandstone and provides a better opportunity to form an economic deposit. The Colorado Plateau region in the USA is the most famous place showing this type of uranium deposit.

Placer Deposits

A placer ore body is a deposit of sand, gravel, or soil containing eroded particles of valuable minerals. Due to the chemical and physical properties of the minerals, they can resist and become concentrated in the surface environment. Classical minerals in placer deposits are platinum metals; gold, present in the native or metallic form; many heavy minerals such as rutile, ilmenite, zircon, and monazite (they are sources of titanium, zirconium, niobium, and other high-technology metals); and gemstones such as diamond, garnet, or ruby.

Moreover, in this type of deposit the valuable minerals are clearly denser than other minerals that are transported at the Earth's surface. This allows minerals to be separated from detrital minerals or rock fragments that constitute the overall sediment load and finally to become concentrated in ore bodies. Therefore, a simple washing in a gold pan easily separates light minerals of valuable ones. There are many classifications of placer deposits of economic importance, but the most useful separate them as residual, eluvial, colluvial, fluviatile, and coastal; marine and beach placers are also terms used for coastal placer deposits. Other types include Aeolian placers and placers in glacial sediments, but they commonly have no economic significance.

Placer gold deposits have produced two-thirds of the gold mined over time. The fluvial placers of California, Australia, and elsewhere were mined out very rapidly in the gold rushes, usually over periods of only a few years. At present, production continues in the Witwatersrand ore bodies of South Africa, a hydrothermally reworked conglomeratic paleoplacer deposit that is the largest gold deposit in the

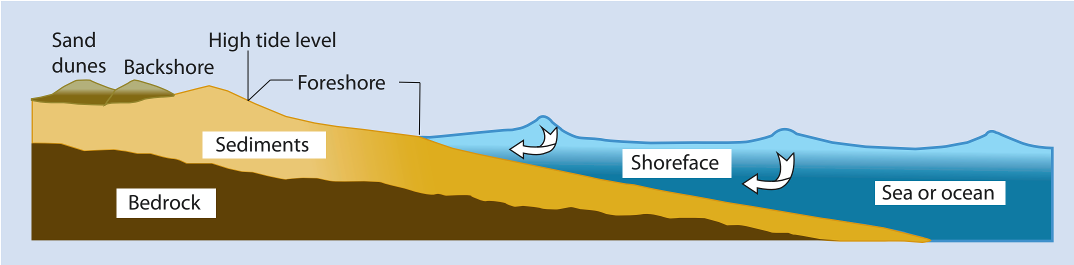


Fig. 2.46 Features commonly used to describe shoreline (strandline) depositional environments associated with deposits of heavy-mineral sands (not to scale) (Van Gosen et al. 2014)

world. An example of placer exploitation in the past is Las Médulas in Spain, mined by the Romans (see Box 3).

The processes that form coastal deposits of heavy-mineral sands usually begin inland and can be described in the following sequence (Van Gosen et al. 2014): high-grade metamorphic and igneous rocks that contain heavy minerals are weathered and eroded, contributing detritus composed of sand, silt, clay, and heavy minerals to fluvial systems; streams and rivers carry the detritus to the coast, where they are deposited in a variety of coastal environments such as deltas, the beach face (foreshore), the nearshore, the barrier islands or dunes, and the tidal lagoons as well as the channels and floodplains of streams and rivers in the coastal plain (Fig. 2.46); the sediments are later reworked by waves, tides, longshore currents, and wind, which are effective mechanisms for sorting the mineral grains on the basis of differences in their size and density. Regarding the age, most economic deposits of heavy-mineral sands are Paleogene, Neogene, and Quaternary in age.

Famous placer deposits also include diamond placers, which are the source of about 34% of global diamond production. The first diamonds discovered in South Africa were in gravels of the Orange River and its tributaries, and tracing these rivers back to their sources led first to the discovery of the primary diamond sources in kimberlites around the town of Kimberley in the center of South Africa and then to huge beach placers at the western coast of the continent in countries such as South Africa and Namibia (Fig. 2.47). Other interesting examples of placer deposits are those related to tin, platinum, and thorium-uranium metals. Regarding the tin placer deposits, Malaysia is the world's greatest producer of cassiterite, and

Fig. 2.47 The Debmar Atlantic is one of five deep-water mining vessels operating off the Namibian coast to extract diamonds (Image courtesy of De Beers)

about half of the deposits are located in placers in rivers, beach sands, and offshore deposits (the other half is related to granites). The same pattern can be applied to the platinum group elements.

Autochthonous Deposits

This type of deposits includes a large number of sedimentary ore deposits of varied characteristics. Sulfide deposits, mainly in black shales, conform the first group. Deposits focused in two metals, iron and manganese, form the second group, and phosphate and different types of salt deposits shape the third group. Autochthonous sulfide deposits are the second most important sources of copper in the world behind porphyry copper deposits and the most important sources of cobalt (Hayes et al. 2015).

Stratiform sediment-hosted copper deposits are hosted in black, gray, green, or white (reduced) sedimentary strata within or above a thick section of red (oxidized) beds. Mineralization consists of fine-grained copper and copper-iron-sulfide minerals that occur as stratabound to stratiform disseminations in siliciclastic or dolomitic sedimentary rocks. Regarding their tectonic setting, they are found in intracontinental rift-related sedimentary sequences and vary considerably in size, grade, and metal association. These deposits are characterized by zoning of ore minerals laterally along and across bedding, from pyrite and chalcopyrite to bornite and chalcocite.

Most famous deposits of this type are the Kupferschiefer in Central Europe and the Central African Copperbelt. The models proposed for the formation of these deposits fall under two main groups: synsedimentary (syngenetic) and diagenetic (syndiagenetic). According to the syngenetic model, sulfides precipitated in an anoxic water column containing H_2S from bacterial sulfate reduction as in the present Black Sea. In the diagenetic model, the ore emplacement occurred during early diagenesis or late diagenesis of the host sediments, which is a difficult question to answer, especially for deposits that have been subjected to metamorphism and deformation (Misra 2000). Taylor et al. (2013) and Hayes et al. (2015) suggested that sediment-hosted stratabound copper mineralization is derived from hydrothermal fluids generated during diagenesis and lithification in sedimentary basins.

With regard to iron and manganese, autochthonous ores are chemical, partly biogenic marine sediments. Although manganese nodules and crusts of the deep oceans may become an essential source of these metals, actually the most important raw materials of this group are enriched parts of marine-banded iron formations and manganese formations (predominantly formed in the Paleoproterozoic) and ooidal or massive iron and manganese ore beds that are of Phanerozoic in age. The so-called banded iron formations (BIF) constitute by far the most abundant and economically the most important iron-rich sediments.

The term BIF means bedded chemical sediments, which comprise alternating layers of iron minerals, commonly oxides or hydroxides, and fine-grained quartz (e.g., chert). The banding is manifested at different scales, not only centimeter-thick beds but also millimeter or submillimeter lamellae. In the major iron formations, the bedding has an impressive continuity: a single 2.5 cm-thick band has been traced over an area of 50,000 km^2, and varves at a microscopic scale are continuous for 300 km. Banded iron formations were deposited at three different time periods, all in the Precambrian, receiving different names for each type: Algoman, Superior, and Rapitan, respectively. Algoman-type deposits are usually small and are found in Archean greenstone belts in association with volcanic rocks. Superior-type deposits were the first iron-rich deposits mined, being located in marine shelf sediments. Finally, the Rapitan-type deposits are a relatively minor type, occurring in association with Neoproterozoic glacial deposits.

Oxides such as hematite or magnetite are the main phase in most banded iron formations although carbonate, silicates, or sulfide are the main minerals in other BIFs. Primary iron formations contain 20–30% Fe, but the ores mined in most countries contain grades ranging from 55% to 65% Fe. This is because enrichment processes act on the iron formations as they are exposed at or near the surface. Exposure under hot, humid climate conditions to circulating groundwater leaches silica from the rock and replaces it by iron oxides.

More autochthonous sedimentary deposits include manganese deposits (Fig. 2.48), phosphate deposits, and sodium and potassium nitrates and sulfates (Fig. 2.49). Regarding bedded manganese deposits, they are formed in a similar manner to iron formations, and the mineralogy assemblage is formed by pyrolusite (MnO_2) and rhodochrosite ($MnCO_3$), which precipitate from seawater as bedded sedimentary rocks. Manganese deposits occur in rocks of all ages, the largest deposits occurring in Proterozoic ore bodies of the Kalahari in South Africa. Phosphorites, which are mined to be used as fertilizers, form on shallow continent shelves either through direct precipitation from seawater or by diagenetic replacement of limestone.

2

Fig. 2.48 GEMCO sedimentary manganese oxide mine (Australia) (Image courtesy of BHP Billiton)

Fig. 2.49 Potassium sulfate underground mine (Brazil) (Image courtesy of Vale)

Brine Deposits

Although current global production and resources of potash are dominated by stratabound potash-bearing salt deposits, in some areas of the world, closed-basin potash-bearing brines are the main source for production of potash and potash-bearing brine resources. These brines may be alkaline or enriched in chloride, sulfate, or calcium, depending on the geological features of the drainage basin and the resultant chemistry of the inflows into the basin. Potash-bearing brines form in salt lakes and salars or playas in closed basins in arid environments, where high rates of near-surface evaporation concentrated the brine. The duration of this process is very variable, but it can range from hundreds of years to tens of thousands of years, even over a million years. From acidic to intermediate volcanic rocks and sometimes saline and continental sedimentary rocks are the main source rocks for this type of deposit (Orris 2011).

Fig. 2.50 Evaporation of brines to obtain common salt (Spain) (Image courtesy of José Pedro Calvo)

The evaporation of brines (Fig 2.50) produces chemical precipitates that are extracted to obtain common salt, sylvite (KCl), gypsum, and anhydrite. Evaporites including halite or gypsum can also form from seawater evaporation in broad inland seas where there are extensive water evaporations. Sodium and potassium nitrates and sodium sulfates are also evaporation deposits. In this sense, one of the driest regions in the world, the Atacama Desert of Chile, includes the world's largest natural deposits of sodium nitrate.

On the other hand, the process of evaporation can be induced artificially, as occurs in some lithium brine deposits. These deposits account for about three-fourths of the world's lithium production. Lithium brine deposits are accumulations of saline groundwater enriched in dissolved lithium. All producing lithium brine deposits share a number of first-order characteristics such as arid climate, closed basin including a playa or salar, tectonically driven subsidence, associated igneous or geothermal activity, adequate lithium source rocks, one or more suitable aquifers, and enough time to concentrate a brine (Bradley et al. 2013). All closed-basin lithium-brine deposits that are of present economic interest are of Quaternary age (e.g., Atacama Salar; Fig. 2.51). Brine, typically carrying 200–1400 milligrams per liter (mg/l) of lithium, is pumped to the surface and concentrated by evaporation in a succession of artificial ponds, each one in the chain having a greater lithium concentration. After a few months to about a year, a concentrate of 1–2% lithium is further processed in a chemical plant to yield various end products, such as lithium carbonate and lithium metal.

Diagenetic Deposits

As aforementioned, diagenetic deposits form a complex group of mineral deposits where the qualification of the ore-forming fluid as diagenetic or hydrothermal is almost impossible, since both are sometimes the same. The previous described stratiform sediment-hosted copper deposits are a good example of this controversy. The Mississippi Valley-type Pb-Zn-F-Ba deposits hosted in marine carbonates are probably the most representative mineral deposit type of this group (Box 2.11: Reocín Pb-Zn Mine (Spain)).

Fig. 2.51 Atacama Salar (Chile) (Image courtesy of SQM)

Box 2.11

Reocín Pb-Zn Mine (Spain)

Mesozoic basins in the north Iberian Peninsula contain Zn-Pb Mississippi Valley-type mineralization mainly in the Basque-Cantabrian basin. Thus, the Reocín zinc-lead (Zn-Pb) deposit in the Basque-Cantabrian basin of northern Spain is the largest known stratabound carbonate-hosted Zn-Pb deposit in Spain and one of the world's largest known Mississippi Valley-type (MVT) deposits. Prior to closure in 2003, the deposit yielded approximately 62 Mt of ore grading, 8.7% Zn and 1.0% Pb after 150 years of exploitation. This is a stratabound ore deposit 3300 m long and 800 m wide, formed by different mineralized and overlapped bodies with variable richness, locally reaching thicknesses up to 100 m included in barren intermediate zones. Previous geologic investigations on the genesis of this deposit have generated the typical confrontation between proponents of a syngenetic origin and supporters of an epigenetic origin, very common in Mississippi Valley-type deposits. The stratigraphic and structural setting, timing of epigenetic mineralization, mineralogy, and isotopic geochemistry of sulfide and gangue minerals of the Reocín deposit are consistent with the features of most of Mississippi Valley-type ore deposits.

Reocín was discovered in 1856 and first mined by the Compagnie Royale Asturienne des Mines and, since 1981, by its affiliated company, Asturiana de Zinc, S.A. Miners began the extraction of «calamines» (oxides, hydroxides, and carbonates of Zn, Pb, and Fe) in Reocín, and at the beginning of the twentieth century, as the exploitation got deeper, the sulfides started appearing, forcing a change in the calcination treatment and the installation of the first European plant of sulfide flotation (1922). Between 1943 and 1965, mining work focused in the interior, but a collapse caused the reactivation of open-pit mining. Since 1976, a mixed system was developed with both open-pit mining and interior works (Santa Amelia well). Peak production was reached between 1990 and 1995. The exhaustion of the deposit and the lack of new reserves caused the closure of Reocín in 2003.

The mineralization occurs within Lower Cretaceous-dolomitized Urgonian limestones (116 ± 1 Ma) on the southeastern flank of the Santillana syncline. The Urgonian Complex reaches a thickness of 4000 m of marine sediments. It is limited at the base by siliciclastic formations of saline and freshwater environments and, at the top, by a sandy complex. Its most characteristic facies are limestone with rudists and dolostones, the Reocín ores being always located in the dolostones. The geometry of the mineralized bodies is highly variable (stratabound), conditioned by synsedimentary faults contributing as paths for the circulation of dolomitizing and mineralizing fluids, which formed deposits along bedding planes and fractures. The Reocín ore bodies appear only slightly deformed, and few faults are observed in the mine. The most important mineralized level in Reocín, for its extension and grades (>25% Zn), is the so-called Southern Layer, mostly hosted in dolomite and even replacing it locally.

The mineralogic and paragenetic sequence of the ore minerals is simple and includes, in order of abundance, sphalerite; wurtzite; galena; marcasite; pyrite, accompanied by dolomite; and rare calcite as gangue minerals. Sphalerite is usually the major sulfide, commonly precipitating as colloform and banded growths (Fig. 2.52).

Galena is present as skeletal or dendritic growths, evidence of rapid precipitation. Carbonate gangue is usually dolomite; at Reocín, several precipitation stages of this carbonate have been recognized. Marcasite is locally very abundant in this deposit. The following minerals are found in the open-pit showing supergenic alteration: smithsonite ($ZnCO_3$), hydrozincite ($Zn_5((OH)_3CO_3)_2$), goethite (FeOOH), hemimorphite ($Zn_4(Si_2O_7)(OH)_2H_2O$), etc. It is important to bear in mind that the deposit was discovered in the eighteenth century due to the presence of a pervasive gossan, although there are evidences for the extraction of the oxidation area since Roman times.

Fig. 2.52 Typical mineralization from Reocín (Santander, Spain)

Mississippi Valley-type (MVT) deposits are a large and heterogeneous group that contains a substantial amount of the reserves of zinc and lead in the world. They are the main source of these metals in the USA and contribute significantly to the production of lead and zinc in Canada and Europe, usually occurring in districts (clusters) that may extend over hundreds of square kilometers and contain up to 500 million tons of ore. These deposits constituted a wide group of lead-zinc mineral deposits that occur mainly in carbonates of any age from the Proterozoic to the Cretaceous (no MVT deposits have been reported from the Archean). In spite of the abundance of appropriate carbonate rocks, the Proterozoic contains only a few MVT deposits. MVT deposits display their maximum presence from Devonian to Carboniferous. By that time, vast and permeable carbonate platforms and abundant evaporites are formed. According to Leach et al. (2010), the intense orogenic activity during the assembly of Pangea in relatively low latitudes created abundant opportunities for the migration of sedimentary brines into the interior carbonate platforms and within extensional domains landward of the orogenic belts to form MVT deposits.

This type of deposit is typically stratabound and takes place in dolostones, although limestone or sandstone can also include this mineralization, and always at shallow depths along the flanks of sedimentary basins. The most common depositional setting is represented by platform carbonate sequences, commonly reef facies, located in fairly undeformed foredeeps or in foreland thrust belts. MVT deposits are mineralogically simple, although considerable variation exists among districts in terms of the total ore-gangue assemblage. Thus, the most typical mineralogy includes sphalerite and galena as dominant minerals and lesser amount of pyrite, marcasite, dolomite, calcite, and quartz. The textures of the sulfide minerals are very varied, and examples are coarse and crystalline to fine-grained textures and/or massive to disseminated ones. One of the most characteristic structures in this type of deposit is banded and colloform structure, which is common as a result of deposition in open spaces. Other recognizable processes consist mainly of dolomitization, brecciation (mineralization in breccias is one of the most characteristic features of Mississippi Valley-type deposits), and host-rock dissolution.

Fluid inclusion studies invoke low-mineralization deposition temperatures ranging from 50 to 200 °C. However, these temperatures are higher than those attributable to normal thermal gradients within the sedimentary pile. Regarding the composition of the ore fluids, they were dense basinal brines, commonly containing 10–30 wt. % dissolved salts. Classical examples of this type of deposit are Viburnum Trend (Southeast Missouri, USA) and Pine Point (Canada). Regarding the origin, the general framework of genetic models for typical MVT deposits is constrained by two important common factors: the ore fluids were moderately hot, highly saline brines, and the mineralization was epigenetic. Controversies are centered in the origin and migration of ore fluids, the source(s) of the mineralization constituents, and the mechanisms of mineral precipitation.

2.8.4 Metamorphic and Metamorphosed Mineral Deposits

As commented previously, mineral deposits in metamorphosed rocks can have been originated before, during, or after metamorphic processes. The first category, which is of premetamorphic origin independent from later metamorphic overprinting, is the class of metamorphosed ore deposits. Some authors consider that the skarn-type deposits can be included in the magmatic domain but here are considered as metamorphic ore deposits because they are a product of contact metamorphism. On the other hand, the formation of ore deposits by regional metamorphism is now generally accepted (Pohl 2011), and examples of these deposits are orogenic gold, graphite veins, and several large talc deposits, among others.

The most important ore deposit type is undoubtedly the skarn deposits. They represent a very diverse class in terms of geological setting and ore metals, which range from Precambrian to late Cenozoic in age, and constitute the world's premier source of tungsten and important sources of copper, iron, molybdenum, and zinc. A continuum exists between the porphyry-type and the skarn-type ore deposits, and at least some skarn deposits appear to be mineralized in carbonate wall rocks within porphyry systems. Nevertheless, skarn deposits do possess enough special characteristics to be treated as a distinct class (Misra 2000).

The term skarn, an old Swedish mining term, encompasses a large variety of generally coarse-grained calc-silicate rocks enriched in calcium, iron, magnesium, aluminum, and manganese, regardless of their association with minerals of potential economic value. They were formed by replacement of originally carbonate-rich rocks by metasomatic processes (Einaudi et al. 1981). Carbonate rocks such as limestone and dolostone are by far the most common protoliths of skarns, although occurrences of skarns in shales, quartzite, and igneous rocks have been reported. A diagnostic feature of typical skarns is their mineral assemblages; the primary assemblage varies with the compositions of the skarn-forming fluids and the invaded rocks but is characterized by anhydrous Ca-Fe-Mg silicates and pyroxenes (including pyroxenoids), and garnets are of special importance.

Skarn deposits can be classified on the basis of the dominant economic metal(s): iron, copper, molybdenum, gold, tungsten, tin, and zinc-lead. The main ore minerals of these skarn types are, respectively, magnetite (Fig. 2.53), chalcopyrite ± bornite, molybdenite, electrum, scheelite, cassiterite, and sphalerite-galena (e.g., Misra 2000). In addition to distinctive metal associations, the skarn deposits exhibit a systematic variation in skarn mineralogy, especially in terms of pyroxene and garnet compositions, for instance, a decrease in diopside component of pyroxenes through the sequence Cu ≥ Fe ≥ W≥ Zn-Pb skarn deposits. From an economic point of view, seven major skarn types can be distinguished: iron (calcic), iron (magnesian), copper (calcic), molybdenum (calcic), tungsten (calcic), tin (calcic), and zinc-lead (calcic). Skarn ore bodies are also a major source of many industrial minerals, including wollastonite, graphite, asbestos, magnesite, talc, boron, and fluorite.

As an example of this type of deposits, skarn gold deposits «consist of disseminated to massive sulfide lenses and crosscutting veins in carbonate platform sequences superimposed by volcanic and/or plutonic arcs; mineralization is associated with Al-rich garnet-pyroxene skarn assemblages replacing limestone, calcareous siltstone, and carbonatized volcanic rocks adjacent to diorite or granodiorite stocks, dykes, or sills» (Robert

Fig. 2.53 Mineralization of magnetite and sulfides in Cala mine (Spain) (Image courtesy of César Casquet)

et al. 1997). Sometimes, the deposits occur in districts along with porphyry Cu-Mo mineralization, tending to be linked with more mafic, hotter intrusions. Mineralogy includes the following minerals: pyrrhotite, pyrite, arsenopyrite, and lesser amount of telluride minerals, presenting also wide variations in their gold-to-silver ratios (Au/Ag = 1:10 to 10:1).

2.9 Questions

Short Questions

- List the four basic geological requirements for any ore deposit to form.
- What is a metallogenic province?
- List several criteria used to classify mineral deposits.
- What are the main ore-forming processes?
- What «gossan» means? Explain its importance.
- What is the definition of a Btu?
- Explain the term «tar sands.» What are they used for?
- What is the hydraulic fracturing?
- List the rank of coals according to the carbon content.
- What are the industrial rocks? List some examples.
- List the four main types of magmatic ore deposits related to the commodities.
- What are the main examples of hydrothermal ore deposits?
- Describe very briefly the genesis of the so-called «roll-front» uranium deposits.
- Explain why Mississippi Valley-type deposits are typically stratabound.

Long Questions

- Identify relationship between mineral deposits and plate tectonic settings.
- Explain the industrial minerals applications.

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Mineral Resource Exploration

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Summary
This chapter is concerned with the process of analyzing an area to find mineral deposits, which is termed mineral resource exploration. The information collected during exploration is utilized to evaluate the size and quality of an ore deposit and to establish there is an option for it to be mined. Two main phases can be broadly outlined in mineral resource exploration: reconnaissance exploration and detailed exploration. The geological, geophysical, and geochemical methods applied at different stages of mineral resource exploration are described. The methods are organized in order of scale and stage, from remote sensing to drilling, through photogeology, geophysical, and geochemical surveys. Previously, mineral deposit models are applied to predict how and where mineral deposits might occur. Since large databases are generated in geochemical exploration, the main statistical techniques (univariate, bivariate, and multivariate methods) are commented in this heading. Finally, several exploration case studies are summarized to show the main items of mineral resource exploration.

3

3.1 Introduction

Mineral exploration can be defined as the process of analyzing an area of land to find mineral deposits (◘ Fig. 3.1). Therefore, mineral exploration covers all the processes that reflect information about the presence of ore deposits. The information collected during exploration is utilized to evaluate the size and quality of an ore deposit and to establish there is an option for it to be mined. Metal prices mainly define exploration expenditures and, in the long run, by demand of metals. Where metal's demand peaks so does exploration expenditure. Most mineral exploration is carried out by companies with a capital base produced either from existing mineral production or from investors. The company size can vary from small venture capital companies (the so-called juniors) with one or two geologists to great multinational mining companies such as Glencore, BHP Billiton, Rio Tinto, Anglo American, or De Beers with operations on several continents (the so-called majors). A junior exploration company can be defined as a company that focuses solely on the exploration and discovery of mineral deposits and does not operate a mine (Stevens 2010). Although the mining industry includes about 6000 companies, the majority are the 4000–5000 junior exploration companies that do not have a mine in operation.

Junior companies, registered principally on stock exchanges in Canada, Australia, and London, carry out most of exploration, especially metals. They have made almost all of the major new discoveries in the past several decades. These junior do not present any cash flow and mainly depend on funding from the stock exchanges. On the opposite, the majors are those companies with annual revenue over USD 500 million and the financial strength to develop a new mine on their own (SNL Metals & Mining). They are often more selective in their choice of exploration properties. Where a junior company can be happy to discover a relatively small deposit, majors are interested in the world-class deposits that could be developed into a large mining operation. Although majors have the largest exploration budgets, they tend to be less successful than juniors at discovering new deposits. Some of the reasons for this include the following: (1) majors spend many of their exploration budgets drilling around deposits that have already been discovered with the aim of expanding the reserves; (2) majors become too focused on the search for large deposits and thus miss opportunities; and (3) majors buy into deposits or junior companies after the discovery has been made; they leave the high-risk discovery stage to the junior (Stevens 2010). The federal state government, Bureau of Mines, and geological surveys also participate in exploration. In general, the role of the geological surveys commonly includes some mineral exploration information to the government, and the private sector presented as a reconnaissance work.

The main features of the mineral exploration process can be summarized as follows:

1. It is a time-consuming process, ranging from 2 years up to 5 years or more.
2. It is also expensive (2 or 3 millions of dollars per year) and high-risk investment, unlike ordinary businesses investments.

◘ **Fig. 3.1** Electromagnetic survey in the field for mineral exploration (Image courtesy of Alrosa)

3. It is undertaken in various stages of investigation, each phase conditioned by the results of the previous step.
4. It starts at the broad scale and narrows down the work area to settle on a target or a set of targets.
5. The methods used vary in the different phases of the process, and this variation is defined by the size of the prospect as well as the type density of information needed.
6. Rarely results in a mine are being developed; the rate for finding new profitable mining operations commonly ranges from a high of 4% to less than 1% and even sometimes as low as 1‰.

Exploration field activities take place as part of strategies to locate and define a particular economically mineable mineral commodity in a mineral province. In this sense, the prospect could be an ancient mine, an outcrop including mineralization, an area elected based on geological items, or simply some anomalous feature of the environment such as a geophysical or geochemical result that can be interpreted as showing close spatial relation to a mineralization. Thus, mineral exploration companies usually classify exploration programs into two categories: greenfield or brownfield, a terminology originally used in construction and development. Greenfield exploration means unknown territories where ore deposits are not already known to be present (◘ Fig. 3.2). On the contrary, brownfield exploration refers to prospecting in areas where mineral deposits were previously discovered. Obviously, the risk in brownfield exploration is considerably lower than in greenfield exploration because of the lack of geological information available in the latter.

Historically, discoveries have taken place in waves, after the introduction of new methods or advances in the understanding of ore genesis (Paterson 2003). For instance, discovery rates jumped sharply between 1950 and 1975, following the development of new methods and instruments in exploration geophysics and geochemistry.

Very often, the terms prospection and exploration are used in a misleading way. For some authors, exploration sounds similar to prospecting, but other authors consider prospecting simply as the search for ores or other valuable minerals (first stage) while exploration (second stage) estimates as faithfully as possible the size and value of an ore deposit, by using techniques very similar to but more intensive than those used in the previous phase of prospecting. Thus, the line to differentiate between prospecting and exploration usually is not possible. In this chapter, with the exception of the section devoted to mineral exploration stages, the terms prospection and exploration are used indistinctly to avoid problems of interpretation.

■ **Fig. 3.2** Amulsar region (Armenia): example of a greenfield mineral exploration territory (epithermal-style gold mineralization) (Image courtesy of Lydian International)

The mineral deposits to explore now for the mining companies are mainly hidden by leached and weathered outcrops, with soil or other cover. For this reason, very sophisticated exploration techniques are actually needed to find them since most mineral deposits located at or near the Earth's surface have probably been discovered. As a general rule, the first stage of prospection/exploration involves locating prospective deposits using knowledge of ore genesis and occurrence models. Thus, geological environments associated with the wanted type of mineral deposit are target of investigation. Methods such as geological mapping and sampling, geophysical surveys, and geochemical analysis are commonly used at an early stage of exploration to define potential ore deposits. Thus, the goal of geophysical/geochemical exploration is to find an anomaly something different from the normal or expected; anomalies can indicate the presence of minerals and could be a target for drilling. An anomaly is a geological incongruity that has the possibility of being an ore deposit. Obviously, an anomaly does not necessarily imply a mineral deposit, but every mineral deposit was first an anomaly, that is, something out of the ordinary (Hartman and Mutmansky 2002). Where a mineral deposit has been identified, the next step is to map it more extensively to obtain a first evaluation of the grade and tonnage of the mineral deposit. The target is later drilled to study the mineralization in depth; drilling is undertaken only in advanced mineral exploration. In increasing order of cost per km^2, the main methods used in mineral exploration are remote sensing, geological mapping, geophysical surveys, geochemical surveys, and drilling.

Regarding the exploration trends in the world, mining companies reacted to the poor market conditions of the last years with a strong decrease in their exploration expenditures. The result was a 19% decline in worldwide nonferrous metal exploration budgets in 2015, compared with the previous year, with final investment of about USD 9.2 billion (SNL Metals &Mining). ■ Figure 3.3 shows the main destinations for nonferrous exploration in 2015. Nonferrous exploration means to look for precious and base metals, uranium, diamonds, and several industrial minerals; it particularly precludes exploration for commodities such as iron ore, coal, aluminum, or oil and gas. Regarding allocation of exploration, «Latin America has been considered the leading region for mineral exploration by many companies for the past decade owing to its promising geology, its long history of world-class discoveries, the perception of its mineral policies and its successful historical record of mineral production and development» (Wilburn and Karl 2016).

Fig. 3.3 Top destinations for nonferrous exploration in 2015 (SNL Metals & Mining)

3.2 Mineral Resource Exploration Stages

It is quite difficult to define exactly the number of stages in mineral exploration processes since it depends of several factors such as the commodity to investigate, the region to explore, the overall costs of the different steps, and others. Up to five stages in mineral exploration, the so-called mineral exploration cycle, are usually found in literature: program design, reconnaissance exploration, detailed exploration, prospect evaluation, and preproduction. However, there is consensus that two main phases can be broadly outlined: reconnaissance exploration and detailed exploration (or prospecting and exploration). Commonly, prospecting is the very first stage in the search for mineral deposits, and permits tend to cover large areas in an attempt to see if mineral deposits are present, whereas exploration involves more detailed data gathering over smaller and specific areas. The complete sequence of mineral activity is carried out for only a very low number of mineral projects, being the initial stages abbreviated if the information acquired in those stages is already accessible to the mining company. Thus, a project can be quickly abandoned at any phase if the results obtained are not clearly hopeful. In other words, as commented above, very few discovered mineral deposits become producing mines.

The time required for exploration of a mining project depends on its size and location. The following time requirements can provide a broad approximation: (1) small deposits, from 2 to 4 years; (2) medium-sized deposits, from 4 to 6 years; and (3) large deposits, from 6 to 10 years of exploration. Actually, the process of mineral discovery and its development to production mine can take up to 25 years, because of the large size of the modern mines.

3.2.1 Program Design

At the program design step (generative stage or project generation, or simply planning stage), the management staff of the company, with considerable experience of exploration, defines the economic parameters for mineral targets.

Technicians, usually geologists and/or geophysicists, design the exploration program that promises the best results in the search for such target. According to Sillitoe (2000), the keystone to prospecting organization is to have the best forthcoming staff and appropriate finance in order to generate confidence throughout the organization. The economic parameters vary widely depending on the expected exploration and development of the type of mineral deposit sought and on the economic factors and mine life. The conduct of a good prospecting program is aimed at the discovery of a maximum number of mineral deposits at minimum cost. In this searching process, geologists decide the types of deposits to explore and which geological and exploration models should be applied. Previously, the management staff chooses the commodity or commodities.

The intensive use of mineral deposit models is a defining feature at this stage. This is because the first step in a new program is to acquire information about the study areas to be investigated. Thus, favorable regions are selected, either on the basis of known potential as expressed by existing mines or mineral occurrences or on the basis of general knowledge of their geological characteristics. In summary, the area to be explored is identified based on literature search, looking at history, reports and maps, and thesis works, among many others; these are called desktop studies. At the end of this stage, exploration procedures are recommended to the management by the geological staff, and a time schedule and general budget are established. Regarding the exploration costs, the exploration manager commonly considers them as an expenditure within an organization while it is as a geologist on a specific exploration project that one becomes involved in the exploration costs within the context of the project (Moon and Whateley 2006). Prospection can be commodity- or site-specific. In other words, the search can be limited to a particular mineral or metal or to a particular geographic area.

3.2.2 Reconnaissance Exploration

Since a prospect has been identified, a progressive series of definable exploration stages can be carried out. As a rule, positive results in any stage of exploration will originate an advance to the next stage and an increase of the exploration effort. On the contrary, negative results reveal that the prospect will be commonly abandoned, although further follow-up is possible if the economic conditions have changed. The first stage of mineral exploration is the reconnaissance exploration, although it can be named also in a variety of ways: simply prospection, target identification, early and extended reconnaissance, and many others. In turn, it typically includes two steps: regional appraisal and reconnaissance of region. The main goal of the process is to identify an ore deposit that can be the target for subsequent exploration; the quantities estimated for the deposits are with a low level of confidence, and these estimates are inferred, that is, based on interpretation of geological, geophysical, and geochemical results.

Reconnaissance exploration aims at rapid and low-cost sorting out of prospective parts of an area. Regions ranging from 2000 to 200,000 km^2 are evaluated with an analysis of accessible information, and parts of a region that cover 100–5000 km^2 are studied through field examination, spaced geochemical sampling with wide grids, and geophysical exploration. In this sense, an invaluable information to surface regional geology is that obtained with regional geophysics. Airborne magnetic, radiometric, and regional gravity data are available in a great part of the developed world (Fig. 3.4), and these techniques lead to refining geological interpretation. Regional geochemical surveys also provide much information in areas of poor outcrop.

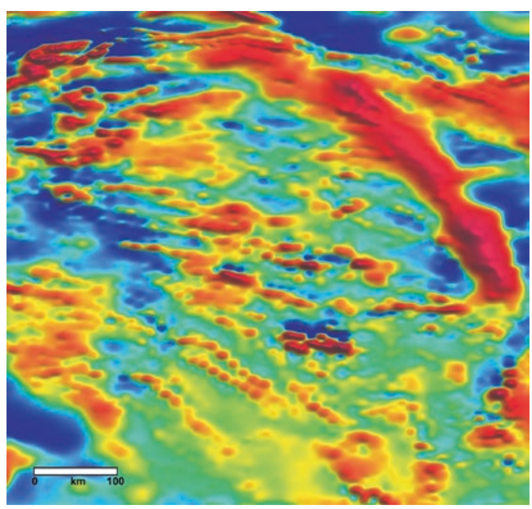


Fig. 3.4 Regional magnetic map in mainland Europe (Image courtesy of Getech)

The results are brought together on maps on 1:50,000 to 1:25,000 or smaller scale. They are geologically analyzed in view of the characteristics of known occurrences of the type of ore deposit being explored. The next step deals with selection of smaller target areas for detailed investigation. In general, the targets are not clearly defined until the first stage has been accomplished: in fact, target identification is the main goal of reconnaissance exploration. It can cost from several thousand to one million or more USD, commonly spending from a several months to 2 or 3 years to complete. Once field studies such as rock and soil sampling have been carried out, the results will be collected and models for the mineralization will be created using specialized computer software.

In the first phase of reconnaissance exploration (regional appraisal), the following procedures are usually performed:

1. Review of all information on the target such as government geological information as well as geophysical and geochemical surveys in the area, the results of previous exploration data and the known occurrence of minerals, and other previous bibliographies
2. Photogeological study of available air photographs
3. Study of accessible remote-sensing information
4. Air and ground field inspection
5. Petrographic and mineralogical studies to determine main rock types, mineral assemblages, and identification of minerals of interest

In the second phase of reconnaissance exploration (reconnaissance of region), techniques are:

1. Geological mapping and sampling
2. Geochemical surveys and indicator mineral studies
3. Geophysical surveys, airborne or ground
4. Shallow pattern drilling for regolith or bedrock geochemistry, including geophysical borehole logging and drilling aimed at increasing geological knowledge
5. Field inspection of outcrops and anomalous areas
6. Petrographic and mineralogical studies, including study of host rock of the deposits and alteration zone, mineralogical studies (ore microscopy, X-ray diffraction, among others), identification of oxidized and primary zones, etc.

3.2.3 Detailed Exploration

If the goal of the previous stage is to locate anomalies due to the presence of a mineral deposit, the objective of detailed exploration is to define and evaluate this deposit in detail. The exploration will focus to determine the geological setting, depth, geometry, grade, tonnage, extent, and worth of the ore deposit identified. Similar techniques than those applied in reconnaissance exploration will be used though in a more comprehensively manner over a much smaller area. Exploration culminates in preparation of a pre-feasibility study that either accepts or rejects the deposit for further consideration. Detailed exploration is restricted to relatively small areas and is intensive and expensive, especially where drilling is carried out. For this reason, it is essential to protect the investment and potential revenue from the prospect by obtaining exclusive exploration or mining rights and to enter in negotiations with owners of surface property in preparation for later mine development (Gocht et al. 1988).

In the final stage of exploration, the target that ranges initially from 2 to 25 or more km^2 is investigated through detailed field inspections, geochemical sampling, and ground and airborne geophysical surveys. It generally begins with establishing a regular grid on interesting areas serving as a base for more detailed geochemical and geophysical studies as well as geological mapping, generally undertaken at 1:10,000 to 1:2500 scales. In this step, it is common to carry out limited trenching, drilling, and systematic sampling as a guideline to developing geological conceptions. In this way, the target is later reduced to a smaller one ranging from 1 to several km^2 for further drilling to establish if the hypothetical valuable mineral deposit really is present. It is clear that investigating if a discovery displays a sufficient size and quality inevitably includes a subsurface investigation. In this case, the geologist usually faces the task of generating a target for drilling. This stage can cost from several tens of thousands to tens of millions of USD, and they will usually take 1 to several years to complete, assuming that there are not disrupts. Once the existence of a valuable ore deposit is determined, perhaps 1 or 2 years after the initial discovery of economic ore, the exploration is considered finished and at that moment the development process of the mine begins.

Classical techniques for this stage are comprehensive geological mapping and sampling; detailed geochemical surveys, with an elaborated grid pattern sampling and analysis; detailed geophysical surveys, usually on the ground; drilling, logging, trenching, and geophysical survey in the holes; and bulk sampling. Drilling involves various types, initially with a relatively wide spacing of holes. In areas of poor outcropping, trenching or pitting is essential (Fig. 3.5) to verify the bedrock source of a geological, geochemical, or geophysical anomaly. Once the samples have been obtained, they must be sent to a laboratory for their analysis (Fig. 3.6). Cost should not be the main factor to select the laboratory. For this decision, accuracy, precision, and an effective proceeding are also requested (Moon and Whateley 2006). Before samples are submitted to the laboratory, it must be ensured that all the elements that can be associated with the explored ore deposit are incorporated in the analysis and very important that this analysis comprises possible pathfinder elements.

The further decision to carry out a feasibility study can be obtained from the information

Fig. 3.5 Trenching in progress (Image courtesy of Petropavlovsk)

Fig. 3.6 Preparing samples for analysis in the laboratory (Image courtesy of Anglo American plc.)

provided by detailed exploration, since resource/reserve estimations for the deposits are with a high level of confidence. This is probably the most critical stage of exploration because decisions involving high costs and potential costs have to be made in view of the results. If a decision is taken that a potential ore deposit has been delineated, the costs of subsequent exploration will drastically increase, usually at the expense of other prospects. At this stage, it is essential to consider that if it is decided to make the decision to close prospection of a mineral deposit after this stage, there is always the option that an ore body has been lost (Marjoribanks 2010).

3.2.4 Pre-feasibility/Feasibility Study

The final step in mineral exploration process is the preliminary feasibility study that analyzes all components (geological, mining, environmental, sociopolitical, and economical) relevant to the determination to develop a mine. In very large projects, the costs involved in evaluation are high so that a pre-feasibility study is almost always carried out during the previous step. Thus, the main goal of this type of study is to assess the various possibilities and possible combinations of technical and business issues, to evaluate the project sensitivity to changes in the individual parameters, and to rank various scenarios prior to selecting the most likely for further and more accurate study. Upon completion of a pre-feasibility study, geological confidence is such that it should be possible to publicly declare ore reserves (from measured and indicated resources) (Table 3.1) and any other mineral resources that can become mineable in the future with further study (Scott and Whateley 2006). The results of the pre-feasibility study determine whether the increasingly large expense derived from full geological, technical, and economic evaluation of a prospect is justified. In other words, this study will detect if the costs involved in exploration are suitable for the earnings that logically can be expected.

The feasibility study is the final evaluation of the profitability of a mining venture in light of the results of exhaustive geological exploration; assessment of mining and processing cost; environmental factors, including mine reclamation; and market

Table 3.1 Example of mineral resource and reserve data presented in a pre-feasibility study of a mining project

Mineral resource table

Category	Tonnage (million tonnes)	Cu Grade (%)	Au Grade (g/t)	Ag Grade (g/t)	Contained Cu (billion pounds)	Contained Au (million ounces)	Contained Ag (million ounces)
Measured	39.5	0.25	0.39	2.58	0.22	0.50	3.27
Indicated	247.2	0.34	0.26	3.81	1.85	2.04	30.26
Total measured and indicated	286.7	0.33	0.27	3.64	2.07	2.53	33.54
Inferred	346.6	0.42	0.24	4.28	3.23	2.70	47.73

Mineral reserve table

	Tonnes (Mt)	Diluted grade			Contained Cu (billion pounds)	Contained Au (million ounces)	Contained Ag (million ounces)
		Cu (%)	Au (g/t)	Ag (g/t)			
Proven probable	69.0	0.606	0.520	4.94	0.9	1.15	11.0
Probable	459.1	0.582	0.291	6.18	5.9	4.30	91.2
Total proven and probable	*528.0*	*0.585*	*0.321*	*6.02*	*6.8*	*5.45*	*102.1*

3

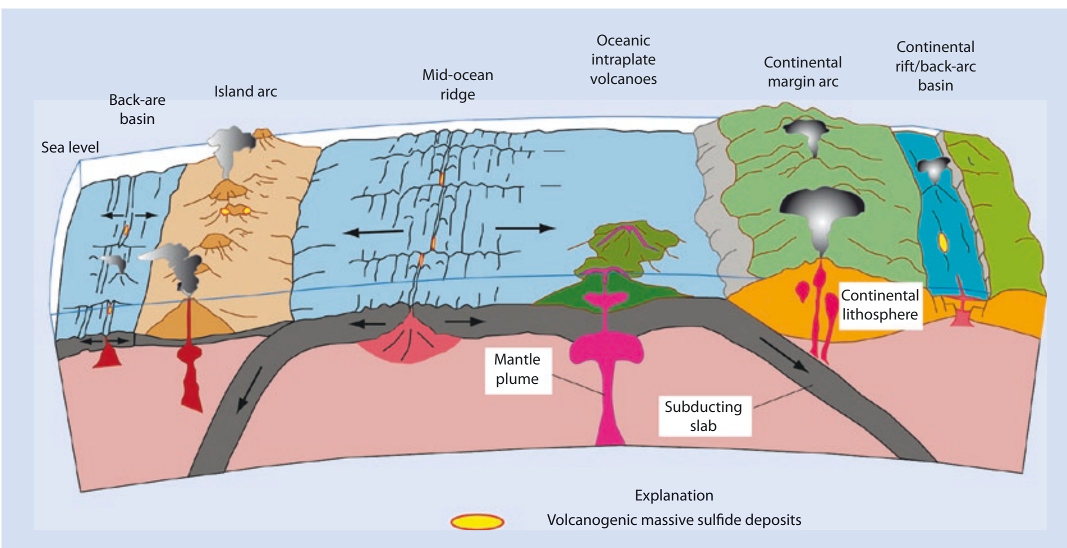
analysis. This study usually forms the basis for the «go/no go» decision on developing a mine (Gocht et al. 1988), that is, it is the basis for an investment decision or decision to proceed to the next stage of development. Obviously, feasibility studies are of higher level of rigor than pre-feasibility studies. Thus, in feasibility studies, social, environmental, and governmental approvals, permits, and agreements, commenced during the pre-feasibility study, will be in place or will be approaching finalization.

A feasibility study incorporates all types of detailed information obtained in previous stages of mineral exploration such as geology, mining, environmental, infrastructure and service, financial data, marketing, economic viability, and many other factors. Moreover, sufficient sample collection and test work have taken place during a feasibility study for more of the resource estimate to be reported in the measured category. Several million dollars are commonly spent in large projects, to bring the project to feasibility study level and sensitivity analyses. They will have been established to analyze the main factors that can have a definitive impact upon the reserve estimation. This will help to calculate the risk associated with the reserve data, which at this stage will enter within the acceptable risk category of the company. It is very common that financial institutes utilize independent consultants to audit the resource and reserve estimations.

3.3 Mineral Deposit Models

To predict and have a better knowledge of how and where an ore deposits can be present, scientists developed mineral deposit models (◘ Fig. 3.7). A working definition of «model» in the context of mineral deposits is «the systematically arranged information describing the essential attributes (properties) of a class of mineral deposits» (Cox and Singer 1986). Models are very useful to organize the information about a mineral deposit because they are simplifications and abstractions based on a large number of individual observations. As such, they need refinement as new data are acquired and have to be set as exploration is carried out. In fact, it is very difficult to find a paper in the contemporary literature on economic geology of a mineral deposit that does not utilize the expression «mineral deposit model.»

Mineral deposit models are developed from the information of a particular important deposit or the combined information of several equivalent deposits. Thus, the grouping of deposits based on common characteristics forms the basis for a classification, but the specification of the features needed for being included in the group is the basis for a model (Barton 1993). Consequently, models contain an element of prediction, particularly where certain physical attributes are characteristic



◘ **Fig. 3.7** Model showing volcanogenic massive sulfide deposits in different tectonic settings (Schulz 2012) of ores of a well-defined deposit type. Models try to be constructed as much as possible «independent of site-specific attributes and therefore contain only those features that are transferable from one deposit to another; this goal is difficult to attain, because it is not always known which features are site-specific» (Cox and Singer 1986).

According the definition of a mineral deposit model, it can aid in identifying areas favorable for finding valuable deposits since they describe all of the essential features of a selected group of mineral deposits (Singer 1995). Obviously, there are a great number of mineral deposit models, new models being created as new types of deposits are identified. The scale of the models can vary from regional size (regional-scale models are constructed through metallogenic studies) to smaller local ore bodies, or even refer to some highlighted part of an ore body.

The application of a particular deposit model will depend on the quality of the database. Some deposit types (e.g., placer gold) are easy to understand and supported by well-developed models while other deposits such as the Olympic Dam mineral deposit model are not still well established and can be represented only by a single deposit. In these cases, the information about the deposit is very difficult to obtain. Thus, the models should be used with caution and with understanding of their limitations. The current trend in exploration and mineral deposit modeling is to incorporate every possible component of individual metal deposits in a database and carry out correlative analyses using computers. This approach is simply a continuation of the mindset that created the descriptive model and the availability of a new tool: the computer. Nonetheless, this model is in reality a simulation, with its inherent case-specific limitations, and as well can give misleading results with limited utility for an emergent phenomenon (Robinson 2007).

The geological surveys of Canada and the USA have originated the vast majority of mineral deposit models as well as a great number of publications describing various mineral deposit types. They are the main source to obtain a complete information about the topic. Interactions between the constructors of published models and the explorationists who use them are critical to the evolution of more accurate and useable models. In this regard, the deposits that cannot be classified or the data that cannot be explained by a previous existing model are commonly those that originate an advance in the knowledge of ore-forming processes.

However, some pitfalls in the utilization of mineral deposit models have been frequently developed. Thus, Hodgson (1990) suggests up to a total of five different pitfalls in the making and using of models all related to corporate or institutional cults and affect industry, academic, and government institutions to an equal extent:

1. The cult of the fad or fashion: an obsession with being up to date and in possession of the newest model.
2. The cult of the panacea: the attitude that one model is the ultimate and will end all controversy.
3. The cult of the classicists: all new ideas are rejected as they have been generated in the hot house research environment.
4. The cult of the corporate iconoclasts: only models generated within an organization are valid; all outside models are wrong.
5. The cult of the specialist: in which only one aspect of the model is tested and usually not in the field.

3.3.1 Types of Models

A subdivision of mineral deposit models into various subtypes can be proposed (Cox and Singer 1986). These are dependent on the attributes used in their definition and on the specific fields of application the modeler has in mind (e.g., applications such as exploration/development, supply potential, land use, education, and research guidance). The following subtypes are proposed (Cox and Singer 1986): (1) descriptive models, (2) occurrence models, (3) grade and tonnage models, (4) occurrence probability models, (5) quantitative process models, and (6) genetic models. The first three are empirical or descriptive models and the last three are conceptual or genetic models. Previously, three basic model types, descriptive, grade and tonnage models, and genetic models, were considered. Basically, the model can be empirical (descriptive), in which several attributes are considered essential, or it can be theoretical (genetic). In the latter, the attributes are interrelated using some fundamental concepts.

Thus, the empirical or descriptive model is based on deposit descriptions, and the genetic model explains deposits in terms of causative geological processes.

Another model type that is very useful for initial economic analyses is the so-called grade and tonnage model. This type of model displays grade and tonnage data for known deposits, being possible from this information to assess the average size and grade of a mineral deposit and the cash-flow if one was met (Evans and Moon 2006). Ideally, mineral deposit types should reflect how the mineral deposit was actually formed. In many cases, there is considerable debate among geologists as to how a specific deposit was formed, and thus classifications based purely on a given genetic model will encounter problems.

Descriptive Models

The classification of mineral deposits based on empirical features will lead to the unique fingerprint of a particular deposit (Herrington 2011). Thus, descriptive models derive from the documentation of the geological, geochemical, and geophysical characteristics of individual mineral deposits. Of the various kinds of mineral deposit models, well-documented descriptive models are of the most direct use in mineral exploration or resource assessment. A descriptive model can be constructed from a single deposit but more commonly includes the essential common information of a group of related deposits. The attributes or properties of a mineral occurrence are, of course, those features exhibited by the occurrence.

Attributes can be considered on at least two scales: the first deals with local characteristics that can be obtained immediately in the field (mineralogy, local chemical halos, among many others), whereas the second incorporates features related to the regional geological setting and that must be interpreted from the local studies or can be inferred from global tectonic considerations. For instance, «the rock sequence under study represents a deep-water, back-arc rift environment, or the area is underlain by anomalously radioactive high-silica rhyolite and granite» (Cox and Singer 1986).

Grade and Tonnage Models

Grade and tonnage models had a profound influence on the creation of mineral deposit models. The idea of relating grade and tonnage data appears to have originated long time ago (e.g., Lasky 1950). Grade and tonnage models of ore deposits are very helpful for quantitative resource estimations as well as to schedule an exploration program. They are useful to classify the known deposits in a region and provide information about the potential value of undiscovered deposits in the exploration area. Thus, the frequency distributions of average grades and tonnages of deposits of various types are calculated and displayed graphically. In a limited area showing favorable geological features, grade or tonnage frequency distribution curves are used to estimate the amount of metal that possibly exists in the area (Box 3.1: Grade and Tonnage Models for Podiform Chromite Deposits).

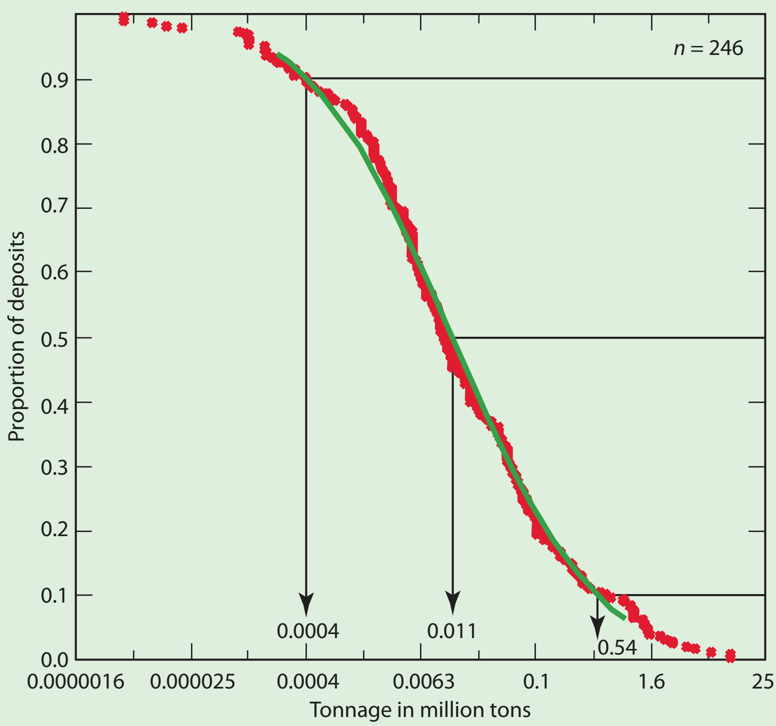
Box 3.1

Grade and Tonnage Models for Podiform Chromite Deposits

Construction of grade and tonnage models for podiform chromite deposits involves multiple steps. The first step is the identification of a group of well-explored deposits that are believed to belong to the mineral deposit type being modeled (Mosier et al. 2012). «Well explored» means completely drilled in three dimensions. After deposits are identified, data from each are compiled. These data consist of average grades of each metal or mineral commodity of possible economic interest and tonnages based on the total production, reserves, and resources at the lowest available cutoff grade. Thus, the grade and tonnage models are the frequency distributions of ore tonnage and grades of Cr_2O_3, ruthenium (Ru), iridium (Ir), rhodium (Rh), palladium (Pd), and platinum (Pt) for the podiform chromite types. The three subtypes of podiform chromite deposits modeled are major podiform chromite, minor podiform chromite, and banded podiform chromite. Percentiles of metal grades from incomplete data sets, such as Ru, Ir, Rh, Pd, and Pt, are based on the observed distributions and are represented by the smoothed curves on the grade plots. Chromic oxide grades for the major and minor podiform subtypes are each significantly different from the normal distribution at the 1% significance level. Only the chromic oxide grades for the banded podiform chromite are not significantly different from the normal distribution at the 1% significance level. In most cases, the departures of the grades from normality appear to be typical for grades greater than 10% in other deposit types.

The reporting of very low grades may be influenced by favorable economics or technology in processing low-grade ores and may indicate regional differences that allow lower cutoff grades. Because these are at the low-grade tail of the distributions and represent a small number of deposits, they may not be important for modeling purposes. For this analysis, grades lower than 30% chromic oxide are excluded. Reports of very high grades may be from deposits where hand sorting of ore was an important processing practice. For metallurgical ores, grades less than 45% chromic oxide are usually rejected at the mills and a Cr to Fe ratio of 3:1 is preferred. For refractory ores, coarser chromite is preferred, and chromic oxide grades can be low as long as the alumina content combines to form at least 60% of the ore. For chemical ores, the chromite must be fine grained, and the chromic oxide grades can be very low as long as there is enough to make chromium salts at a feasible rate. Such a range of chromic oxide grades can contribute to multiple peaks or skewness in the data set.

If there were no differences in grades or tonnages among deposit types, it could be used one model for all types. However, differences in tonnages or grades among the subtypes suggest they should be represented by different models. For example, the deposits associated with major podiform chromite are significantly larger than those associated with minor podiform chromite and banded podiform chromite, and banded podiform chromite deposits are significantly larger than minor podiform chromite deposits.
Frequency distributions of the tonnages and grades of chromic oxide, rhodium, iridium, ruthenium, palladium, and platinum in the three subtypes of podiform chromite deposits can be used as models of the grades and tonnages of undiscovered deposits. Some examples of these frequencies are plotted in ◘ Figs. 3.8 and 3.9. Grade and tonnage models are presented in a graphical format to make it easy to compare deposit types and to display the data. The grade and tonnage plots show the cumulative proportion of deposits versus the tonnage or grade of the deposits. Individual symbols represent the deposits, and intercepts for the 90th, 50th, and 10th percentiles are plotted. Percentiles of grades and tonnages are based on the observed distributions. Relations among grade and tonnage variables are important for simulations of grades, tonnages, and estimated number of undiscovered deposits. These relations also affect the understanding of how deposits form and the assumptions about resource availability. Correlation tests among the variables reveal the relations of grades and tonnage. In general, most of the variables show no relation to each other.



◘ **Fig. 3.8** Cumulative frequency of ore tonnages of major podiform chromite deposits. Each red dot represents an individual deposit (*n* is the total number of deposits). Intercepts for the 90th, 50th, and 10th percentiles of the lognormal distribution are provided. The smoothed green curve represents the percentiles of the data points (Mosier et al. 2012)

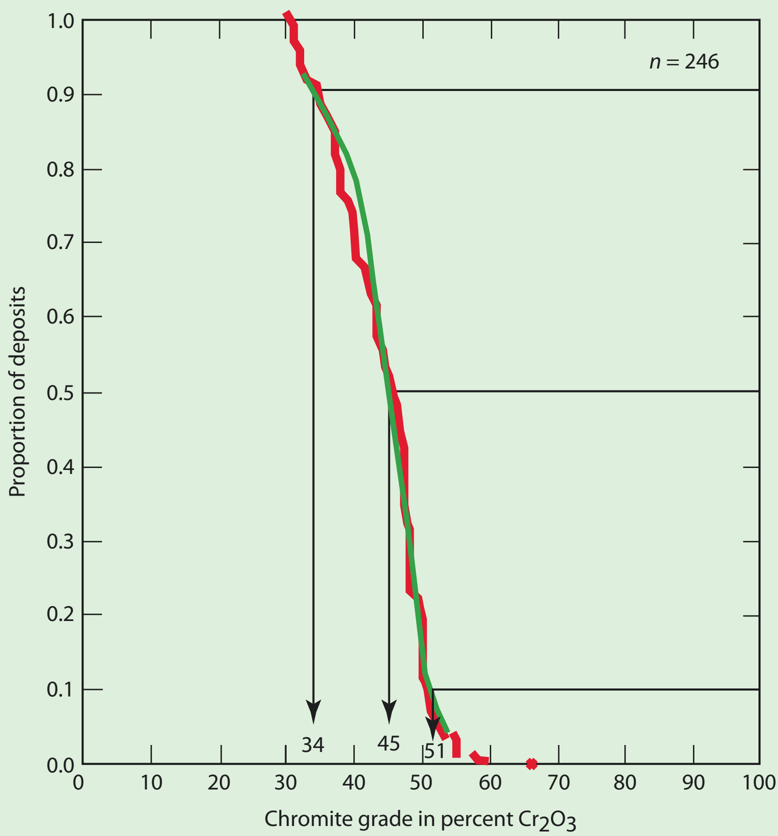


Fig. 3.9 Cumulative frequency of chromic oxide grades of major podiform chromite deposits. Each red dot represents an individual deposit (*n* is the total number of deposits). Intercepts for the 90th, 50th, and 10th percentiles of the normal distribution are provided. The smoothed green curve represents the percentiles of the data points (Mosier et al. 2012)

The mineral deposit density area is a variation of the grade and tonnage model. Deposit density modeling can be used to produce a quantitative mineral resource assessment by estimating the number of undiscovered deposits (Singer et al. 2001). In this type of model, the grade and tonnage model is carried out, and then the number of deposits per unit area is determined for a specific deposit type from a well-explored region. The process originates a frequency distribution that is utilized either directly for an estimate of the resources in a mineral deposit or indirectly as a guideline in some other method.

Genetic Models

Although correct documentation of descriptive models is of the most direct utilization to the exploration geologist, it is almost impossible to develop an adequate descriptive model in the absence of a good genetic one. Similarly, the generation of genetic models depends upon an understanding of the physics and chemistry of ore-forming processes. Therefore, the developing of a mineral deposit model is an iterative process (Duke 1990). Genetic models are more powerful than descriptive models because they provide a basis to distinguish essential from extraneous attributes. In general, the information of a descriptive model is a necessary precondition to create a genetic model.

Genetic models describe the origin of a deposit or deposit type and represent the combination of a descriptive model with one or more process models. Process models simulate physical and chemical ore-forming processes (Fig. 3.10), and they are generic as much as they can apply to a variety of deposit types. In this sense, Duke (1990) affirms that «the geologist engaged in mineral exploration and the government geologist carrying out a mineral-resource assessment combine descriptive deposit models with understanding of the regional geological framework to develop

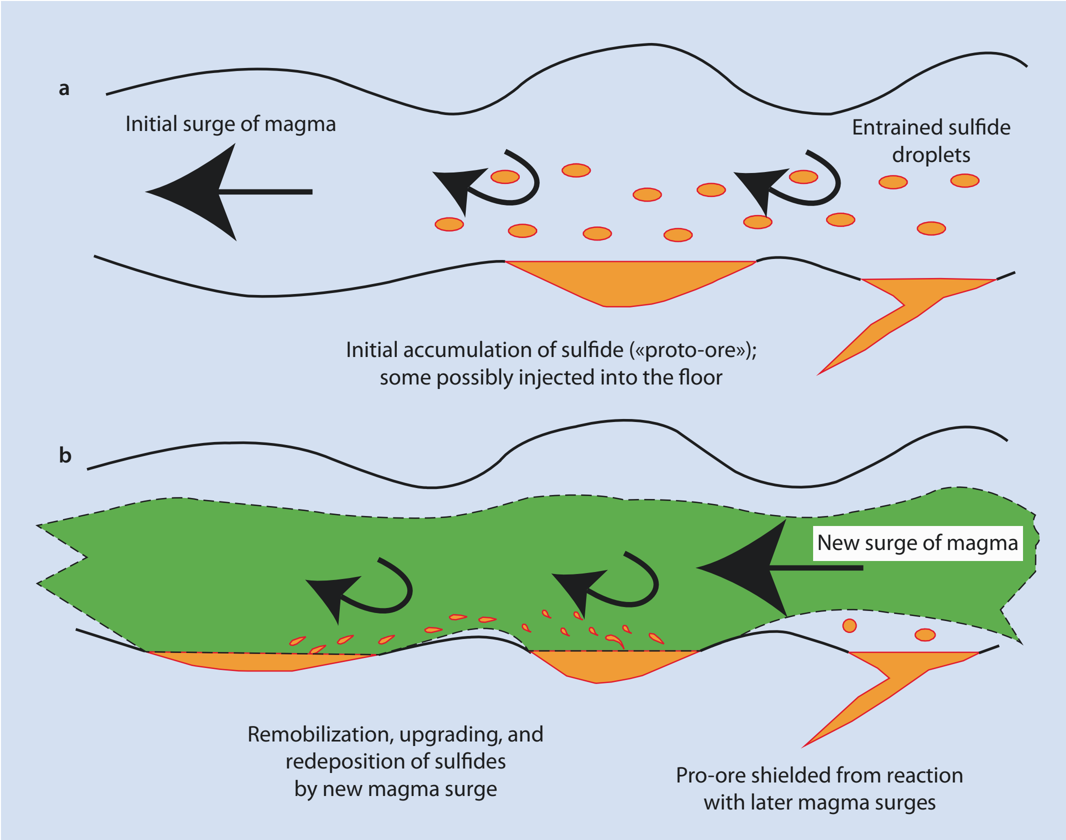


Fig. 3.10 Illustration of the continued flow of magma through an idealized magma conduit (process model) in magmatic sulfide-rich nickel-copper-(platinum-group element) deposits (Schulz et al. 2014)

exploration or resource-potential models.» Even though there are not two mineral deposits identical, empirical descriptions of deposits tend to show natural groupings into a small number of definable categories or types. In turn, these categories tend to coincide with genetically derived models. Therefore, even by using purely physically descriptive classifications, there is often a close coincidence between these and models defined using genetic criteria (Herrington 2011). Descriptive models evolve into genetic models, and as such they become far more flexible and powerful. In fact, there is an iterative relationship among descriptive, genetic, and grade/tonnage models. The consequence of examining these three is that they constituted a linear logical sequence leading toward the «final» model.

One factor favoring the genetic model over the simply descriptive is the great amount of descriptive information needed to represent the many features of complex deposits. If all such information were to be included, the number of models would reach the total number of individual deposits considered. As a consequence, the compilers must use the genetic concepts at their disposal to distinguish the critical from the incidental attributes (Cox and Singer 1986). From both the empirical and the genetic models, the exploration geologist assembles an exploration model, which is a set of recognition criteria for exploration. Some of these criteria are diagnostic for the presence or absence of an ore deposit while others are permissive. The criteria chosen should be as diagnostic as possible and should be both cost- and time-effective (Gocht et al. 1988).

Other Types of Models

In general, because of the previous models, two more model types can be originated: occurrence probability models and quantitative process models. The former are models that predict the probability of a deposit, size, and grade indicated by the appropriate grade and tonnage models, occurring within a given area. The latter are models that describe quantitatively some process related to ore deposit formation, being in fact only branches of the genetic model. All these models can be parts of the «final» model, and recycling of the model back to the early grouping phase assists in refining the selection procedure (Cox and Singer 1986).

Other types of models have also been described by different authors and applied to mineral deposits: cause-effect models (Knox-Robinson 2000; Sirotinskaya 2004), fractal and multifractal models (Mandelbrot 1983), fluid flow-stress mapping models (Heinrich et al. 1996), statistical/probabilistic models (Agterberg 1974), structural models (Kutina 1969), and spatial-temporal models (Ludington et al. 1993). As an example, probabilistic regression models have been especially attractive and useful to mineral resource exploration. In this model, an area of concern is splitted into a grid of square cells, and the presence or absence of the various predictive attributes (e.g., different lithologies, hydrothermal alteration, geophysical or geochemical anomalism) is expressed for each cell, in the form of magnitude, counts or occurrences, or percentage area occupied.

Geoenvironmental Models

Geoenvironmental models are specific because they are designed as natural extensions of mineral deposit models. A geoenvironmental model of a mineral deposit can be defined as «a compilation of geological, geochemical, geophysical, hydrologic, and engineering information pertaining to the environmental behavior of geologically similar mineral deposits prior to mining» (Plumlee and Nash 1995). Thus, the model offers information about natural geochemical variations associated with a particular deposit type and geochemical variations associated with its mining effluents, wastes, and mineral processing facilities, including smelters. Such information should prove beneficial to (1) environmental scientists interested in mitigating potential environmental problems associated with proposed mines; (2) environmental scientists interested in remediating existing problems at abandoned mine sites; (3) land-use planners that are involved in permitting proposed mines or reclaiming abandoned mine lands; and (4) industry interested in mine planning and mineral exploration (Seal et al. 2002).

3.3.2 Maturity of Descriptive-Genetic Models

The current level of genetic knowledge varies considerably from one deposit type to another. For example, placers and evaporites are genetically well-known types of deposits, and the problems in their exploration concern mainly local site-specific geological problems rather than mineral genesis. In contrast, deposits such as the Coeur d'Alene Ag-Pb-Zn veins remain genetic enigmas despite extensive research for a long time. Other deposits are geologically well understood regarding their origin but still very poorly understood in terms of the reasons for their existing at any particular site. Thus, the rate of acquisition of information is very irregular. The several scarps between plateaus in the knowledge curve for some mineral deposit models might mark, successively, the recognition of very important aspects related to the genesis of the deposit, while plateaus denote periods of absence of new knowledge. For instance, «a scarp in the Mississippi Valley-type ores might involve recognition, from fluid-inclusion evidence, that the ores were deposited from warm, about 100 °C, highly saline solutions that could represent neither simple surface nor marine waters» (Cox and Singer 1986).

Moreover, some aspects of any model always remain to be determined and the model never reaches a definitive format. Indeed, «the approach to complete understanding is asymptotic, and a lot of additional effort to clear up the last uncertainty in a nearly perfect model is probably unwarranted» (Cox and Singer 1986). However, new ideas and new technologies can provide the impetus for new improvements in knowledge for until now incomplete models. Obviously, different deposit types can require different amounts of effort to achieve a similar level of genetic understanding.

3.4 Exploration Methods

The geological, geophysical, and geochemical methods applied at different stages of mineral resource exploration are described in the next sections. The methods are organized in order of scale and stage, from remote sensing to drilling, through photogeology, geophysical, and geochemical surveys.

3.4.1 Remote Sensing

Remote sensing is the characterization of the surface of the Earth based on measurements of its reflected or emitted electromagnetic radiation in wavelengths from 0.3 to 3 m, being satellites the main observation platforms. These wavelengths cover the range from the ultraviolet to the microwave radar spectrum although a great number of measurements are made in the visible range by passive methods, in which the reflected natural radiation is estimated. Remote sensing lead to the recognition of major regional topographic features and geologic relationships and helping in the discovering of regions with mineral potential. Since remote sensing was forthcoming since the late 1970s, the data from land observation satellites have supplied a powerful tool for the exploration of mineral resources. Moreover, satellite imagery (Fig. 3.11) investigates the geological characteristics of remote areas of the surface of the Earth without the requirement to access the region on the ground. Thus, remote sensing is providing information on mineral deposit exploration targets without being in contact with the objects.

Remote sensing can highlight ore bodies and their respective mineralization or alteration signatures as well as associated other features such as lineaments and faults. For instance, this method originates strong signals where gossans associated with hydrothermal alteration and oxidation of porphyry deposits are present. Another example would be the discovery of fractures and faults in volcanic regions with veins of precious metals. On the other hand, the interpretation of satellite imagery can originate very useful models before the start of geophysical investigations. In turn, geological and geophysical data can gage models obtained from this technique.

The resolution of remote sensing is restricted by the resolution of the imagery. According to this

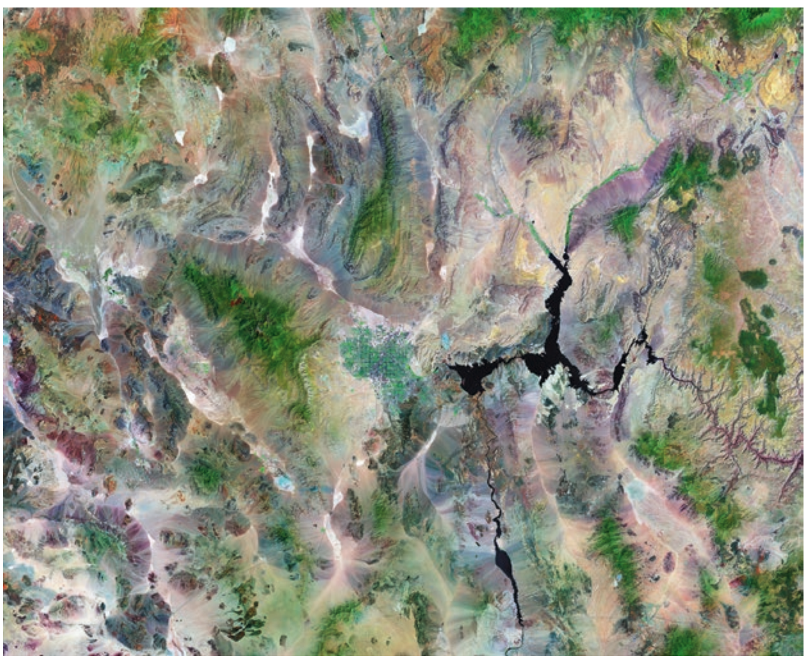


Fig. 3.11 Satellite image (Landsat) used in mineral exploration (pixel = 14.5 m × 14.5 m)

factor, satellites can be classified into three main categories: (1) VHR (very high resolution), sub-meter pixels; (2) HR (high resolution), 2.5–10 m pixels; and (3) MR (mid resolution), greater than 10 m pixels. An image with 50 m resolution would start to pixelate at scales larger (more detailed) than 1:100,000. By contrast, a very high-resolution (VHR) satellite scene with a 50 cm resolution could be viewed at scales to 1:2500 before pixelation became apparent. Mid resolution data can be used for the initial, broad-scale study, to derive, locate, and designate smaller areas of interest, while higher-resolution data are utilized for subsequent analyses.

In contrast to electrical, magnetic, and gravity methods that compute force fields, remote-sensing technique is usually referred to methods that use the electromagnetic energy as radio waves, light, and heat as the means of finding and measuring target features. In the context of geological mapping, electromagnetic methods can be classified as (1) passive optical methods (utilize the sunlight as the source and estimate the reflectance of the surface of the Earth in the visible and infrared spectral bands) (e.g., Landsat 7 ETM+ and the ASTER instrument from the Terra satellite) and (2) active microwave radar methods (use a microwave source onboard of the satellite and calculate the backscatter from the Earth) (e.g., Radarsat-1 and the radar sensor from the Shuttle Radar Tomographic Mission [SRTM]). For its part, infrared imagery is divided into three classes: (1) very near infrared, which detects particularly vegetation; (2) short wave infrared, the best possibility to discriminate sedimentary rocks; and (3) thermal infrared, utilized to discriminate dark materials such as non-sedimentary rocks (Laake 2011). The most famous satellite used in geological studies is Landsat (Box 3.2: Landsat Program).

Box 3.2

Landsat Program

The Landsat program is a series of Earth-observing satellite missions jointly managed by NASA and the US Geological Survey. In the mid-1960s, stimulated by the USA's successes in planetary exploration using unmanned remote-sensing satellites, the Department of the Interior, NASA, and the Department of Agriculture embarked on an ambitious effort to develop and launch the first civilian Earth observation satellite. Their goal was achieved on July 23, 1972, with the launch of Landsat 1, originally named «ERTS» for Earth Resources Technology Satellite. Thus, the Landsat program, a joint effort of the US Geological Survey (USGS) and the National Aeronautics and Space Administration (NASA), was established to routinely gather land imagery from space. NASA develops the remote-sensing instruments and spacecraft, then launches and validates the performance of the instruments and satellites. The USGS then assumes ownership and operation of the satellites, in addition to managing all ground reception, data archiving, product generation, and distribution. Since 1972, Landsat satellites have continuously acquired space-based images of the Earth's land surface, coastal shallows, and coral reefs. Landsat satellites image the Earth's surface along the satellite's ground track in a 185 km-wide swath as the satellite moves in a descending orbit (moving from north to south) over the sunlit side of the Earth. Landsat 7 and Landsat 8 orbit the Earth at 705 km altitude. They each make a complete orbit every 99 min, complete about 14 full orbits each day, and cross every point on Earth once every 16 days.

For most geologists and other Earth scientists, multispectral imagery is synonymous with NASA's Landsat series. The primary sensor onboard Landsats 1, 2, and 3 was the Multispectral Scanner (MSS), with an image resolution of approximately 80 m in four spectral bands ranging from the visible green to the near-infrared (IR) wavelengths. In July 1982, the launch of Landsat 4 saw the inclusion of the Thematic Mapper (TM) sensor with a 30 m resolution and 7 spectral bands. Although the Landsat series was designed initially to provide multispectral imagery for the study of renewable and nonrenewable resources, geologists immediately recognized the geological potential of the Landsat images, and the bands 5 and 7 in the TM were chosen specifically for their geological applicability. The approximate scene size of TM images is 170 km north-south by 183 km east-west, and the radiance measured by the Landsat sensor is a measure of the integration of soil, rock, and vegetation characteristics. Landsat 7 carries the Enhanced Thematic Mapper Plus (ETM+), with 30 m visible, near-IR, and SWIR bands, a 60 m thermal band, and a 15 m panchromatic band. Landsat 8 is the latest satellite (2013) in this series (Fig. 3.12) and operates in a near-circular, near-polar, sun-synchronous orbit with a 705 km altitude at the equator. It carries two push-broom sensors: the Operational Land Imager (OLI) and Thermal Infrared Sensor (TIRS), both of which provide improved signal to noise ratio and 12-bit radiometric quantization of the data.

The use of satellite imagery is now a standard technique in mineral exploration, and Landsat imagery has been used to provide basic geological maps, to detect hydrothermal alteration associated with mineral deposits, and to produce maps of regional and local fracture patterns, which can have controlled mineralization or hydrocarbon accumulations. For instance, TM band 7 (reflected IR) of Landsat satellite, with wavelengths between 2.08 and 2.35 micrometers and resolution of 30 m, is very useful for mapping hydrothermally altered rocks associated with mineral deposits.

Fig. 3.12 Artist concept of Landsat 8 (Image courtesy of NASA's Goddard Space Flight Center)

Because different rock types reflect radiation to different degrees and in different spectral ranges, remote sensing allows preliminary geological interpretations of an area. Thus, some of the geological features intimately associated with ore deposits provide strong signals that can be detected by this technique. These features are often clearly recognizable, even through soil cover or vegetation. Different surface materials such as water, vegetation, or clay alteration generate different signals of radiation in varying wavelength bands. This pattern of reflectance is characteristic for each type of land surface and is known as its reflectance signature. In mineral exploration, this can be especially meaningful in looking for surface alteration systems where argillic alteration can be present (Sabbins and Oliver 2004). Finally, the full potential of remote-sensing data can only be obtained by combining all forthcoming spectral bands in digital processing. This is because the combination enables improving the interpretation of linear structures, gossans, hydrothermal alterations, and so on.

3.4.2 Photogeology

World War I was the onset of the development of aerial photography and photointerpretation. Photointerpretation is the study of the character of the ground surface using the aerial photographs. Aerial photographs are pictures of the ground surface taken from the air with a camera pointing downward, and they are mainly used for the production of topographic maps. Some important additional uses are regional geological mapping (1:36,000 to 1:70,000), detailed geological mapping (1:5000 to 1:20,000), open-pit management, land use, agricultural and forestry applications, water resource applications, urban and regional planning, and environmental impact assessment, among many others. While satellite imaging covers very large areas of the Earth's surface, aerial photography and photogeological interpretation provide the topographic and geological basis for exploration work of smaller areas of 10 km^2 or less. For this reason, most exploration studies involve multi-image interpretation.

Interpretation of standard aerial photographic images remains as an important tool, being highly effective especially when used for logistic and planning. The study of the aerial photographs cannot substitute the field investigations, but rather it helps and contributes to them. The advantages of the study of the aerial photographs are as follows: (a) they save time and provide to observe a larger area; (b) they have more detailed ground surface than maps; (c) they can be studied anytime and anywhere; and (d) the studies carried out on the photographs are cheaper and easier than studies in the field.

In turn, photogeology is the interpretation of the geological and geomorphological features as well as various lithofacies on the aerial photographs, a source of geological information that can be unobtainable elsewhere. The use of aerial photographs in geology includes (a) outlining the structure and structural relationship in an area; (b) outlining the stratigraphic succession; (c) preparation of a geological map; (d) measurements of stratigraphic sections; (e) measurements of dip and strike and thickness of formations; and (f) inferences about rock types present in the area (Dirik 2005).

Based on scale, there are different types of aerial photographs: large-scale (1:5000 to 1:10,000), medium-scale (1:10,000 to 1:20,000), small-scale (1:20,000 to 1:60,000), and very small-scale (>1:60,000) aerial photographs. The photographs used mostly are at the scale of 1/35000, with a size of 18 × 18 cm. The size of the photograph cannot be greater than 25 × 25 cm because stereographic viewing is only possible for this size. In turn, based on film used, aerial photographs can be panchromatic black and white photographs, infrared black and white photographs, and infrared colored photographs. Aerial photographs can also be classified as oblique or vertical. Oblique photographs can be either high angle oblique photographs or low angle oblique photographs. Vertical photographs are those taken by a camera pointing vertically downward.

The factor that produces the strongest three-dimensional effect in photointerpretation is stereoscopic vision. Two photographs of the same terrain, but taken from different camera stations, generally permit three-dimensional viewing and are said to comprise a stereoscopic pair, also commonly referred to as a stereo pair. Overlapping adjacent photographs (overlap of 60–90%) along the flight path enable subsequent stereoscopic (three-dimensional) viewing. The two main pieces of photointerpretation equipment are field stereoscopes and mirror stereoscopes. The latter are mainly utilized in the office and can view full 23 cm × 23 cm photographs without overlapping.

Where available at a suitable scale and resolution, aerial photographs are the best medium upon which to construct a geological map. Thus, the initial interpretation made from the images will provide: «(a) definition of areas of outcrop and areas of superficial cover; (b) preliminary geological interpretation based on topographic features, drainage patterns, colors and textures of rocks, soils and vegetation, trend lines of linear features, etc.; (c) geological hypotheses for field checking; (d) selection of the best areas to test these hypotheses; and (e) familiarity with the topography and access routes to assist in logistic planning of the field programme: access roads and tracks, fording points for streams, potential helicopter landing sites, etc.» (Marjoribanks 2010) In this sense, topographic studies using drones are common in mineral exploration (Fig. 3.13).

Tone in aerial photographs refers to the brightness at any point on a panchromatic photograph and is affected by many factors (e.g., nature of the rock – sandstone is light, but shale is dark). Basic extrusive and intrusive igneous rocks display usually darker tone while bedded sandstone, limestone, quartzite, and acid igneous rocks are commonly lighter; mudstone, shale, and slate show intermediate tones (Whateley 2006). With regard to the texture, there is a large variation in apparent texture of the ground surface as seen on aerial photographs. Moreover, texture is often relative and subjective. However, drainage pattern indicates the bedrock type that affects soil characteristics and site drainage conditions. For instance, dendritic drainage occurs on relatively homogenous material such as flat-lying sedimentary rocks and granite, and radial drainage radiates outward from a central area, typical of domes and volcanoes. Moreover, the distribution of vegetation commonly offers information about the rock types. For example, sandstone and shale can be cultivated, while dolerite is left as rough pasture. On the other hand, lines of vegetation (e.g., trees) are the best indicator of fractures, faults, veins, and joints.

Fig. 3.13 Drone for topographic study at Nigeria (Image courtesy of Eduardo Revuelta)

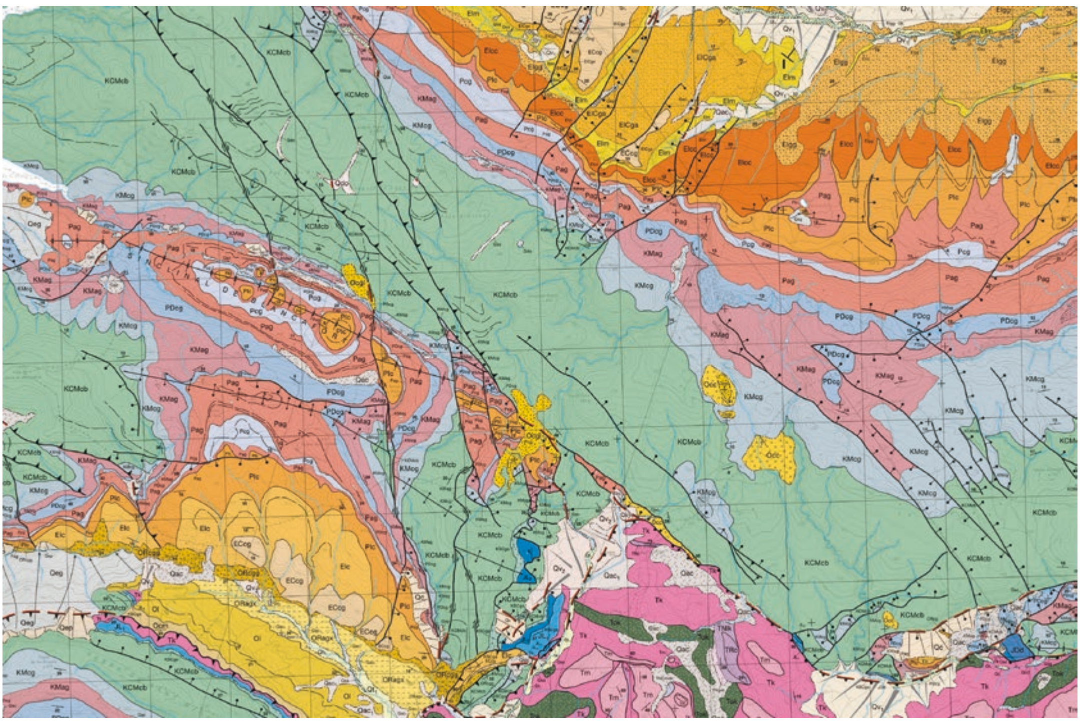


Fig. 3.14 Part of a 1:25,000 geological map (IGME, Spain)

3.4.3 Geological Mapping

Publication in 1815 of the first colored, hand-painted geological map of England and Wales by William Smith heralded the birth of modern geology (Winchester 2001). Today, two centuries after this early mapping was done to locate bedrocks suitable for construction of canal systems, geological maps (Fig. 3.14) are used as a means of presenting the observations as well as constructing geological hypotheses. Geological mapping plays an important role throughout the mine life cycle, from regional- to district-scale exploration targeting, through drilling and ore discovery, to deposit assessment, ore-reserve estimation, pre-production mine planning to production, and, ultimately, mine closure.

3

Geological mapping has been used extensively for mineral exploration for more than 100 years. Beyond the use of traditional paper-based mapping tools, recent technological advances incorporate global positioning systems, pen tablet computers, and laser ranging devices that all support direct (paperless) field-based digital geological mapping. In this sense, geographic information systems (GIS) revolutionized exploration practices (Box 3.3: Geographic Information Systems).

Box 3.3

Geographic Information Systems

A geographic information system (GIS) is a computer system (hardware, software, and netware) and associated database designed to efficiently capture, store, update, manipulate, analyze, retrieve, and display all forms of geographically referenced information. The first known use of the term geographic information system was in 1968. In 1986, Mapping Display and Analysis System (MIDAS), the first desktop GIS product emerged for the DOS operating system. Then, this was renamed in 1990 to MapInfo for Windows when it was ported to the Windows platform. Recently, a growing number of free, open-source GIS packages run on a range of operating systems and can be customized to perform specific tasks.

Modern GIS technologies use digital information, for which various digitized data creation methods are used. The most common method of data creation is digitization, where a hard copy map or survey plan is transferred into a digital medium through the use of a CAD program and geo-referencing capabilities. Geographic data can be stored in a vector or a raster format. Using a vector format, two-dimensional data is stored in terms of X and Y coordinates. For instance, a road or a river can be described as a series of X and Y coordinate points. Thus, the vector system is good for describing well-delineated features. A raster data format expresses data as a continuously changing set of grid cells. The raster model is better for portraying subtle changes such as soil-type patterns over an area. Most geographic information systems make use of both kinds of data. Once all of the desired data have been entered into a GIS system, they can be combined to produce a wide variety of individual maps, depending on which data layers are included.

In mineral exploration, the data are usually organized in layers of different types such as topography, remote sensing, geophysical and geochemical results, etc. Some GIS applications, for instance using ArcGIS, are specifically developed to represent and process particular types of geological, geochemical, and geophysical information. Raster images, such as satellite or geophysical imagery, can be integrated and overlain with vector data such as geology, faults, and sample information. Thus, GIS is essential in customizing and integrating a broad range of mineral exploration data consisting of information on drillholes with summary stratigraphic logs, rock sample and drillhole sample geochemistry, mineral occurrences, magnetic and gravity images, digital geology, current and historic exploration details, and much more (Fig. 3.15).

The ultimate objective of using a GIS during Mineral exploration is to predict the approximate positions of new mineral deposits. For doing this, the data to be integrated should be indicative of the mineral deposits searched according to an exploration model customized for the area under analysis. In this sense, remote-sensing data often constitutes an important part of the database introduced in a GIS because of its intrinsic digital nature and because it can be used as the base over which to overlap other data. By combining GIS technology with the enormous progress in recent years in remote sensing, it has been possible to extend the mineral exploration all over the world. Moreover, recent integration of exploration data with GIS, supported by intelligent systems, has greatly enhanced the acquisition, analysis, and interpretation of complex problems of probabilities and decisions involved in mineral projects. Mapping of mineral potential using GIS is conducted to delineate areas with different probabilities of hosting certain types of mineralization. The main steps in generating mineral potential maps are (a) establishing the exploration conceptual model; (b) building a spatial database; (c) spatial data analysis (extraction of evidence maps and assigning of weights); and (d) combination of evidence maps to predict mineral potential.

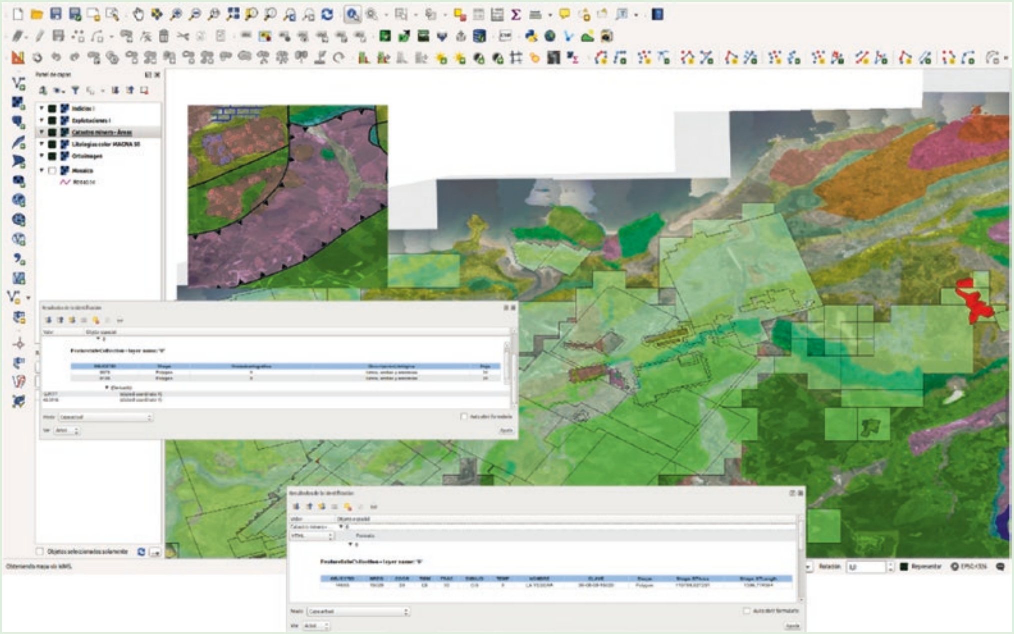


Fig. 3.15 GIS image released with QGIS (a free and open-source geographic information system) including different types of information such as geology or mining data (Image courtesy of Miguel Ángel Sanz)

There are two main reasons that mapping remains an essential part of mineral exploration. First, mapping creates the geometric patterns that represent the geological attributes of an exploration target. Second, there are scientific, engineering, and financial implications of mapping because subsequent geophysical modeling, ore-reserve estimation, financial forecasting, and economic evaluation are based on the interpretation of such work (Brimhall et al. 2006). The quality and scale of the geological map will vary with the importance of the program and the finance available. Scales of geological maps range from reconnaissance (1:24,000 or smaller) to detailed project scale (1:100 to 1:12,000).

Geological mapping is widely used in planning exploration strategies such as the selection of regions to explore for certain types of ore deposits. Prior to mapping campaigns, existing geological maps are examined and can be compiled to emphasize key geological features to assess exploration potential. Exploration geologists commonly use existing maps as the basis for preliminary examinations to assess mineral potential, frequently in conjunction with geochemical, geophysical, or remote-sensing surveys or compilation of mine and prospect data.

However, geological maps available today, either published by government surveys or in many scientific journals, are generally not well suited for special needs of mineral exploration and development and require exploration geologists to undertake specialized mapping. Whereas published maps of general geology do outline information essential to exploration, including rock units, stratigraphy, ages of rocks, and general structure, they are in most cases not sufficiently detailed to help delineate mineral deposits that are typically 1–2 km^2 in outcrop area even for world-class deposits. Consequently, the geological mapping at this stage generally is done at a more detailed and larger scale than published mapping, and key lithologic units and features of mineralization or hydrothermal alteration are mapped using the reconnaissance techniques.

Since geological information is commonly recorded on maps and cross sections at a scale appropriate to the aims, property geology must be defined at a scale of 1:5000, while mineral deposit geology must be mapped to a scale of 1:1000 or even more detailed. Information displayed in this type of map includes faulting, folding, rock types, fracture/vein density and orientation, evidence of primary porosity/permeability, and phases of mineralization, among many others.

Regarding geological mapping in underground mines, it can play an essential role in mineral exploration. Abandoned mine workings are the most direct guides of the mineralization in a region and provide the immediate information on ore occurrences. If the workings are active, they provide a series of fresh geological exposures with each meter of advance, and they supply well-located sites for underground drilling and sampling.

3.4.4 Geophysical Exploration

Introduction

Mineral exploration is increasingly being addressed to searching for buried and deep targets since there are few large ore bodies to be found at the surface. Unlike geochemistry and other remote-sensing techniques, geophysics helps to look at into the subsurface and to provide information about the concealed geology. Thus, geophysics is an integral part of most mineral exploration programs. Geophysical techniques have been used in mineral prospecting for the past 300 years, beginning in Sweden around 1640 with the use of magnetic compasses in exploring for iron ore. These techniques are essential in areas where outcrop is poor or has been subject to intense mineral search over a long period. In some cases, geophysical techniques also enable for quick regional appraisal of areas where ground access is almost impossible, for instance, rain forest terrain or developing countries with insufficient infrastructure (Marjoribanks 2010).

For a geophysical technique to be useful in mineral exploration, there must be a clear contrast in the physical characteristics of the minerals, rocks, and ores related to the existence of valuable minerals. Geophysical anomalies, defined as differences from a constant or slowly varying background, can be recorded. Ideally, the actual economic minerals will produce them, but even the presence of a clear physical contrast between mineralization and surrounding rocks does not imply a significant anomaly (Milson 2006).

Geophysical measurements in the natural environment will be contaminated with unwanted information. This is called noise, which is a source of error, while the information being sought in the measurement is known as signal. Signal amplitude should be as high as possible whereas noise signal should be as low as possible, in order to obtain an accurate measurement of the parameter of interest. In any case, suppression of noise is of outmost importance and must be considered at every stage of the geophysical program, from data acquisition to presentation of the data for interpretation (Dentith and Mudge 2014).

Geophysical methods can be classified as passive (magnetism, specific gravity, and radioactivity) and active methods (electric conductivity, electromagnetic properties, and seismicity). Passive methods use natural sources of energy, of which the Earth's gravity and magnetic fields are two examples, to investigate the ground. The geophysical measurement is made with a detector, sensor, or receiver, which measures the response of the local geology to the natural energy. In turn, active geophysical methods involve the deliberate introduction of some form of energy into the ground, for example, seismic waves or electric currents. Again, the response of the ground to the introduced energy is measured with some form of detector (◘ Fig. 3.16). These methods are more complicated and expensive to work with.

◘ **Fig. 3.16** Geophones for receiving seismic signal (Image courtesy of International Geophysical Technology)

The geophysical signal can be directly related to mineral deposits, for example, a magnetic anomaly caused by magnetite ore in an iron deposit. More commonly, geophysical methods provide indirect evidence that leads to interpretations of the subsurface geological distribution of rocks, but it does not directly or necessarily reflect the presence of a mineral deposit. These types of methods are applied to both mineral discovery and geological mapping. They are useful because geophysical responses of materials can be measured through vegetation, soil cover, and extraneous overburden. In many cases, geophysical measurements provide the only means of interpreting the geological characteristics of the subsurface short of drilling, which is much more expensive (Gocht et al. 1988).

Over the area of interest, geophysical instruments are deployed in the field to measure variations in a physical parameter associated with variations in a physical property of the subsurface, and the measurements are used to infer the geology of the survey area. Of particular significance is the ability of geophysical methods to make these inferences from a distance and, for some methods, without contact with the ground. A considerable number of geophysical exploration methods are available for mineral exploration, and each method exists in several variants. The specific choice is a function of the geological and exploration model of the targeted deposits; of general conditions such as remoteness, climate, and human land use; and of the costs (Shen et al. 2008). Through either ground, airborne, or in-ground (downhole) methods, geophysical studies employ the types of surveys cited above to detect anomalous signals related to the presence of minerals.

The chief advantages of airborne surveying relative to ground surveying are the greater speed of data acquisition and the completeness of the survey coverage. After their introduction in the 1950s, airborne geophysical surveys became commonly used as a first step in geophysical exploration. They provide the quickest, and often the most cost-effective, ways of obtaining geological information about large areas. Two or more methods are commonly combined in one survey to obtain data that are more accurate. In surface geophysics, geophysical work on the ground is normally rather slow. Results from airborne and surface surveys are matched with surface geological data to decide if it is worth proceeding with further exploration.

Geophysical techniques are routinely used in exploration programs to help the project geologist delineate areas favorable for the type of target being pursued. They can be used to directly detect some minerals, indirectly detect others, and map geological and structural features in exploration programs. Direct detection includes using induced polarization (IP) to find disseminated sulfides, magnetics to delineate magnetite-hosting rocks, and gravity and electrical techniques for massive sulfides. For instance, indirect detection of targets includes «using IP to detect pyrite in association with sphalerite and gold (both non-responders to IP geophysical techniques), and copper and molybdenum in porphyry systems; magnetics are routinely used to search for hydrothermal alteration in association with porphyry systems, and can be used to map buried stream channels (e.g. magnetite sands) that might host placer gold» (Mukherjee 2011). Seismic surveys are highly effective for investigating layered stratigraphy, so they are the mainstay of the petroleum industry but are comparatively rarely used in the minerals industry. Regarding costs of geophysical surveys, the seismic method is the most expensive, while airborne magnetic and radiometrics are the less expensive.

It is very important to note that most important advances in geophysical exploration for ore deposits in the last 25 years dealt with advances in theory or practice of the different methods but also with the development of more sophisticated instrumentation and especially more powerful data processing. These advances together with the use of GPS for survey positioning control have greatly reduced the cost and time involved in all geophysical surveys and have increased their resolution in the detection of anomalous signals in the data.

Traditionally, most geophysical data has been presented for interpretation in the form of contoured or raster plans and sections that can be interpreted in terms of the geology and ore mineralization that they represent. However, new methods of analyzing and presenting geophysical data have been introduced in the last two decades to revolutionize the interpretation process. These methods are generally referred to as «data inversion» (McGauchy 2007; Oldenburg and Pratt 2007) (▫ Box 3.4: Data Inversion in Geophysical Exploration).

Box 3.4

Data Inversion in Geophysical Exploration

Geophysics is traditionally used to predict the position of a mineralized body by seeking out geophysical anomalies. The new inversion techniques establish the geophysical properties of rocks and then measure their geophysical signatures in the field. Thus, it is possible to generate three-dimensional models of their potential mineralization and the surrounding geological environment. Inversion models are generally much easier to interpret than the original data and provide a superior understanding of the subsurface.

The target deposit in mineral exploration is commonly buried within a complex geological structure, and the contribution of the other units masks the sought response. In such cases, direct visual interpretation of the target location is difficult or impossible. Thus, the geophysical data need to be «inverted» to recover a distribution of the relevant physical property that can explain the observations. Geophysical data inversion enables resource explorers to extract more insight from geophysical data by converting geophysical measurements into 3-D images of the subsurface that can be integrated with other surface and subsurface geologic observations. Insights generated from geophysical inversion have helped to improve prospecting and focus drill targeting, particularly in deeper and more complex subsurface environments. The 3-D geophysical inversion is now possible for almost all geophysical methods that are commonly used in mineral exploration. Over the past decade, geophysical inversion has proved its effectiveness in exploring for ore deposits and major oil reserves around the world.

Inversion techniques make use of complex computer algorithms and information of the geophysical properties of the rocks and potential mineral deposits of the prospect, to construct mathematically a geological model that agrees, or is at least compatible, with the geophysical observations. The results are presented as a 2-D or 3-D geological model of the body of rocks that were surveyed (Fig. 3.17). Instead of finding the single possible response to a given earth model (forward modeling), inverse modeling will help determine what 3-D distribution of physical properties yields a measured field response. The known information (e.g., overburden thickness, lithology from drill data, and borehole assay results) can help constrain the inverse problem to a limited number of plausible models. The most useful models are the result of exploring the inversion model space by running many scenarios with different constraints and sensitivity to other geological information. Therefore, new algorithms and faster computers have a huge impact on the success of geophysical inversion for exploration. Similarly, the ability to easily integrate and use supplementary information to better constrain the inversion is critical to producing reliable models. In summary, geophysical inversion produces physical property models from geophysical data whereas forward modeling produces data from a physical property model of the Earth.

However, it is important to realize that, as with all computer

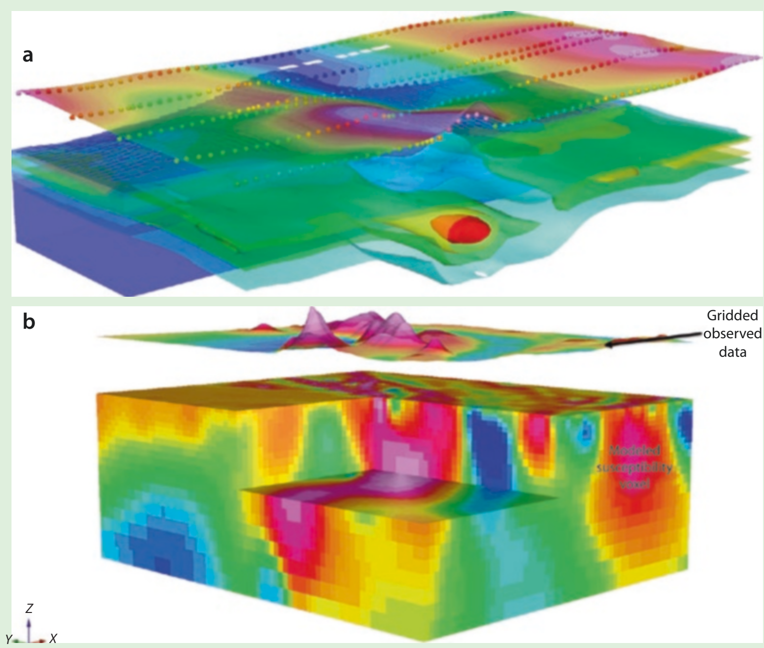


Fig. 3.17 3-D geophysical inversion images. **a** 3-D conductivity model from frequency-domain electromagnetic field inversion (Geosoft VOXI Earth Modelling); **b** 3-D susceptibility model from magnetic field inversion (Geosoft VOXI Earth Modelling) (Images courtesy of Geosoft)

models, the product of inversion modeling is only as good as the geological choices made in setting up the model parameters and the accuracy of the geophysical properties that are used in its construction. It is a feature of geophysical inversion models that shows they are not unique: many different models can be constructed that will reproduce the geophysical pattern that was measured in the field. Choosing between different possible models requires geological knowledge about the area, and the better that knowledge, the more useful and realistic the inversion model.

Gravity Methods

In this geophysical method, subsurface geology is investigated based on variations in the Earth's gravitational field developing from differences of density between subsurface rocks. Gravity surveys have been widely used to understand general subsurface structure as measurement of gravity by gravimeters is relatively easy. The mean value of gravity at the surface of the Earth is about 9.8 m/s^2, and variations in gravity caused by density variations are of the order 100 μm/s^2. This unit of the micrometer per second squared is referred to as the gravity unit (gu). An accuracy of ±0.1 gu is quickly attainable in gravity surveys on land and corresponds to approximately one hundred millionth of the normal gravitational field (Kearey et al. 2002). The instrument used in gravity surveying is called a gravimeter (◘ Fig. 3.18), an extremely sensitive weighing machine. At each survey station, location, time, elevation, and gravimeter reading are recorded. The measurement of relative values of gravity, which is the differences of gravity between locations, is the standard procedure in gravity surveying. Before the results of a gravity survey can be interpreted, it is necessary to correct for all variations in the Earth's gravitational field that do not result from differences of density in the underlying rocks. This process is known as «gravity reduction» (LaFehr 1991) and basically includes instrument drift, latitude, elevation, and tidal corrections.

◘ **Fig. 3.18** Gravimeter (Image courtesy of International Geophysical Technology)

Gravity differences over the surface of the Earth are due to density differences between adjacent rocks. Density contrasts of different materials are controlled by a number of factors such as type of rock, grain density of the particles forming the rock, and the porosity and interstitial fluids within the material. Rock densities are among the least variable of all geophysical parameters and range from less than 2.0 g/cm^3 for soft sediments to more than 3.0 g/cm^3 for mafic and ultramafic rocks. Obviously, many ore minerals (e.g., metal sulfides) are clearly denser than their host rock. For this reason, the ore bodies are commonly denser than their surroundings. However, it is important to note that actual effects are tiny, usually amounting to less than 1 ppm of the total field of the Earth, even considering large massive sulfide deposits. Gravimeters must then be very sensitive, a specification which is commonly in conflict with the request to be also rugged and field worthy.

The variations in the density of the crust and cover are presented on a gravity anomaly map (◘ Fig. 3.19). A gravity anomaly map looks at the difference between the value of gravity measured at a particular place and the predicted value for that place. Gravity anomalies form a pattern, which can be mapped as an image or by contours. The wavelength and amplitude of the gravity anomalies give geoscientists an idea of the size and depth of the geological structures causing these anomalies. Deposits of very dense and

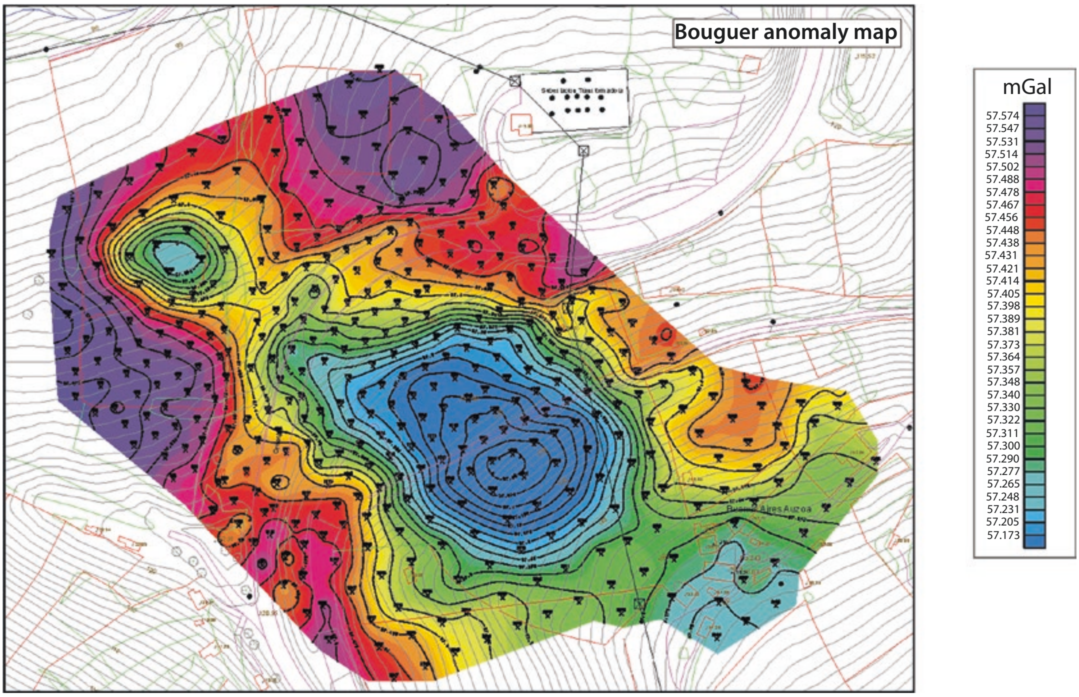


Fig. 3.19 Bouguer anomaly map (Illustration courtesy of International Geophysical Technology)

heavy minerals will also affect gravity at a given point and will produce an anomaly above normal background levels. Anomalies of exploration interest are often about 0.2 mgal, and data have to be corrected for variations due to elevation, latitude, and other factors.

Much less can be deduced about the shape or the depth of the investigated mineral deposit. A deeper body will, other things being equal, give rise to a broader and flatter anomaly. Likewise, the peaks of gravity anomalies are generally situated just above the causative bodies (a causative body is a rock unit of different density from its surroundings), which is not the case for many other geophysical methods. Regarding the interpretation of the measurements, «the reliability of any interpretation, no matter how sophisticated the technique, depends, of course, on the validity of the input assumptions» (Milson 2006).

Gravity surveys can be carried out either from airborne or ground surveys, but the most sensitive measurements are currently achieved from the ground. They are used to evaluate the amount of high-density mineral present in an ore body, and as a general rule, gravity prospecting is only used for mineral exploration if substantial density contrasts are expected. Thus, chromite and sulfide bodies possess very high densities, and buried channels, which can contain gold or uranium, can be detected because they have relatively low density. In fact, gravity surveying is rarely used in reconnaissance exploration because it is relatively slow to execute and therefore expensive. However, gravity methods are very useful as a follow-up method utilized on a target defined by another, more cost-effective method. In this sense, gravity surveys, along with regional aeromagnetic data, played a significant role in the discovery of the giant deeply buried Olympic Dam mineral deposit in Australia (Rutter and Esdale 1985), and the discovery of the Neves Corvo sulfide deposits in Portugal was carried out utilizing regional gravity surveys of the Portuguese pyrite belt on 100 and 200 m grids (Leca 1990).

Magnetic Methods

Magnetic methods, which are probably the oldest of geophysical exploration methods, thrived after the World War II. Magnetic surveys (Fig. 3.20) measure variations of the Earth's magnetic field

Fig. 3.20 Magnetic survey (Image courtesy of International Geophysical Technology)

caused by the presence of magnetic minerals. Magnetic outcomes result primarily from the magnetization induced in susceptible rocks by the magnetic field of the Earth: everywhere in the Earth there is a natural magnetic field. These methods are widely used, both as an essential assistance in regional mapping and for direct location of those mineral deposits that show distinct magnetic signature. Magnetic and gravity methods have much in common. The magnetic map, however, is generally more complex, and the variations in field are more erratic and localized than the gravity map. Thus, the precise interpretation of magnetic field data is usually much more difficult than for gravity.

Magnetic surveys are often utilized in metallic mineral exploration, particularly locating iron ores. However, ordinary hematite, the most abundant ore of iron, rarely produces anomalies large enough to be detectable in conventional aeromagnetic surveys. The combination effect of several geophysical techniques such as aeromagnetic interpretation with study of regional gravity and radiometric data can produce further gains in the interpretation of the underlying rocks.

Minerals can be diamagnetic, paramagnetic, or ferromagnetic. In diamagnetic minerals, all the electron shells are full; there are no unpaired electrons. Diamagnetic minerals show negative susceptibilities and examples of these materials are quartzite and salt although many of the elements and compounds exhibit diamagnetism. Salt domes thus give diamagnetic anomalies (weak negative anomalies). Paramagnetic minerals are ones where the electron shells are incomplete; as a result, they generate weak magnetic fields. By definition, all materials that are not diamagnetic are paramagnetic. Examples of materials that are paramagnetic are the 20Ca to 28Ni element series. Finally, ferromagnetic minerals are minerals that are paramagnetic, but where groups of atoms align to make domains. There are only three ferromagnetic elements: iron, cobalt, and nickel. Almost all natural magnetic minerals are of this kind. Magnetite, which is the most abundant, ilmenite, hematites, titanomagnetite, and the oxides of iron or iron and titanium are common ferromagnetic minerals. Magnetite (Fe_3O_4) is found disseminated through most rocks in differing concentrations. The magnetization can be either temporary (induced) in the same direction as the field of the Earth or permanent (remanent) and fixed in direction with respect to the rock, regardless of folding or rotation. All geologically significant magnetic minerals lose their magnetic properties at about 600 °C, a temperature reached near the base of the continental crust. Consequently, local features on magnetic maps are virtually all of crustal origin (Milson 2006).

As a rule, the magnetite content and, therefore, the susceptibility of rocks are very variable, being present a considerable overlap between different mineralogies and lithologies (Table 3.2). Basic igneous rocks are commonly highly magnetic because this type of rock has a relatively high magnetite content. In this sense, the proportion of magnetite in igneous rocks usually decreases with increasing acidity; for this reason, acid igneous rocks are generally less magnetic than basic rocks. Metamorphic rocks are also very different in their magnetic character, depending of the metamorphism grade. Regarding sedimentary rocks, they are effectively nonmagnetic unless they contain a significant amount of magnetite in the heavy mineral fraction. Thus, if magnetic anomalies are detected in areas covered with sediments, these anomalies are mainly originated by an underlying igneous or metamorphic basement or by intrusions into the sediments.

Table 3.2 Magnetic susceptibility of some common rocks

Rock types	Maximum volume susceptibility (SI units)
Igneous rocks	
Andesite	0.17
Basalt	0.18
Dolerite	0.062
Diabase	0.16
Diorite	0.13
Gabbro	0.09
Norite	0.09
Dacite	0.05
Granite	0.05
Granodiorite/tonalite	0.062
Peridotite	0.2
Quartz porphyries/quartz-feldspar porphyries	0.00063
Pyroxenite/hornblendite (Alaskan type)	0.25
Rhyolite	0.038
Dunite	0.125
Trathyte/syenite	0.051
Monzonite	0.1
Phonolite	0.0005
Spilites	0.0013
Avg. igneous rock	0.27
Avg. acidic igneous rock (pegmatites	0.082
Avg. basic igneous rock (komatiites, tholeiite)	0.12
Sedimentary rocks	
Clay	0.00025
Coal	0.000025
Silt/carbonates	0.0012
Dolomite	0.00094
Limestone	0.025
Red sediments	0.0001
Sandstone	0.0209
Shale	0.0186
Tuffs	0.0012
Conglomerate/akose/pelites	0.0012
Arenites/breccia	0.0012
Avg. sedimentary rock	0.05
Metamorphic rocks	
Amphibolite	0.00075
Gneiss	0.025
Granulite	0.03
Acid granulite	0.03
Basic granulite	0.1
Phyllite	0.0016
Quartzite	0.0044
Schist	0.003
Serpentine	0.018
Slate	0.038
Marble	0.025
Metasediments	0.024
Migmatites	0.025
Magnetite skarn	1.2
Avg. metamorphic rock	0.073
Magnetite ~0.1%	0.0034
~ 0.5%	0.018
~ 1%	0.034
~ 5%	0.175
~ 10%	0.34
~ 20%	0.72

The practical unit of magnetic field for survey work is the nanotesla (nT), sometimes also known as the gamma. At the magnetic poles, the field is about 60,000 nT and vertical, while at the equator it is about 30,000 nT and horizontal. The magnitude of the Earth's magnetic field averages to about 5×10^{-5} T (50,000 nT). Magnetic anomalies as small as 0.1 nT can be measured in continental magnetic surveys and can be of geological significance. Today, «with improvements in instrumentation, navigation and platform compensation, it is possible to map the entire crustal section at a variety of scales, from strongly magnetic basement at a very large scale to weakly magnetic sedimentary contacts at small scale» (Likkason 2014). Methods of magnetic data treatment, filtering, display, and interpretation have also improved significantly, especially with the advent of high performance computers and color raster graphics as well as GPS technology.

The instrument used for magnetic surveys is called a magnetometer. Magnetometers record disturbances in the Earth's magnetic field caused by magnetically susceptible rocks. Since the early 1900s, a variety of surveying instruments have been designed. The first device to be developed was the fluxgate magnetometer, which found early application during the Second World War in the detection of submarines from the air. Actually, three types of magnetic sensor are commonly used in geophysical surveying, namely, the proton-precession, the Overhauser, and the alkali-vapor sensors. The operation of all three is based on quantum-mechanical properties of atoms. Importantly, they are sensitive to the strength of the Earth's magnetic field, but they do not however measure its direction but the total magnetic intensity (TMI) (Dentith and Mudge 2014). With the magnetometer data, a map of magnetic variation at the surface, called a TMI map, can provide an image of lithology distribution.

Magnetic methods are used to detect different types of ore bodies in mine prospecting. Magnetic surveys are fast, provide a great amount of information for the cost, and can offer information about the distribution of rocks under thin layers of sedimentary rocks, useful when trying to locate ore bodies. Therefore, magnetic observations are obtained relatively easily and cheaply, and a few corrections are applied to them, explaining why the magnetic methods are one of the most frequently utilized geophysical tools. Three types of correction are carried out in magnetic methods to remove all causes of magnetic variation: diurnal variation, geomagnetic, and elevation and terrain corrections.

Despite these obvious advantages, interpretations of magnetic observations suffer from a lack of uniqueness due to dipolar nature of the field and other various polarization effects. The greatest limitation of the magnetic method is that it only responds to variations in the magnetic properties of the materials of the Earth, which means that many other characteristics of the subsurface are not solvable. «The inherent ambiguity in magnetic interpretation for quantitative techniques is problematic where several geologically plausible models can be attained from the data».

Most magnetic work for mineral exploration is carried out from the air since aeromagnetic surveying is quick and cost-effective, with a cost some 40% less per line kilometer than a ground survey. In this type of survey, the flight lines are spaced 0.5–1.0 km apart at an elevation of roughly 200 m above the ground. Line separations have decreased over the last years and can now be as little as 100 m. Data are recorded digitally and presented commonly as a contour map. Obviously, flying at lower altitudes and decreasing the spacing of the flight line increase the final sensitivity of the survey. In this sense, it is noticeable that extremely detailed surveys, comparable in their resolution to ground magnetic surveys, can be developed using low-flying helicopter.

Ground surveys are conducted to follow up magnetic anomalies identified through aerial surveys. Such surveys can involve stations spaced only 50 m apart. The magnetic survey is generally suspended if periods of large magnetic fluctuation (e.g., magnetic storms) are present. Solar activity, such as spots and flares, cause short-term irregular disturbances with amplitudes that can surpass 1000 nT. Although data are usually displayed in the form of a contour map of the magnetic field, interpretation is often made on profiles. According to Kearey et al. (2002), magnetic anomalies range in amplitude from a few tens of nT over deep metamorphic basement to several 100 nT over basic intrusions and can reach an amplitude of several 1000 nT over magnetite ores.

Direct search for magnetic targets related to mineralization is an important exploration method, especially in those provinces with banded iron formations, IOCG mineralization types, strongly oxidized porphyry copper intrusives, magnetite skarns, or pyrrhotite-bearing massive sulfides. In such cases, favorable anomalies are commonly obtained from high-quality low-level aeromagnetics, followed up then by ground magnetometer traverses and magnetic modeling to define a drill target. «Magnetics have been also used to define subtle exploration targets such as heavy mineral concentrations in palaeo-strand lines and potential iron ore and gold orebodies in palaeochannels» (Marjoribanks 2010). Examples of ore deposits found largely as a result of their magnetic response are the Olympic Dam mineral deposit (Reeve et al. 1990) and the Broken Hill-type deposit of Cannington in Australia. This deposit was discovered as a consequence of drill testing and an air magnetic anomaly, generated by associated pyrrhotite, in a zone of thick younger cover (Walters et al. 2002).

Radiometric Methods

Radiometric surveys carried out the estimation of the gamma rays emitted from the Earth by natural decomposition of frequent radiogenic minerals, being a useful technique to map fault zones or boundaries between geological units. Natural radioactive decay produces alpha particles, beta particles, and gamma rays. These are very high-frequency electromagnetic waves. Thus, radiometric survey is a passive geophysical method because it measures a natural source of energy, similar to gravity and magnetic methods. Most gamma rays are produced in the top 30 cm of soil and rocks that can be sensed by airborne investigations and on surface rocks utilizing a portable spectrometer. Radiometric surveys for mineral exploration are made from the air, on the ground, and within drillholes. Airborne radiometrics is particularly common in mineral exploration where the radiometric data are acquired simultaneously with magnetics during airborne surveying, measurements usually calculated from a low-flying aircraft simultaneously as air magnetic studies. As aforementioned, aerial and ground use are restricted to areas with little soil cover, because most radiation on the surface comes from the uppermost 10–50 cm.

Although the Geiger-Muller radiation detector was used in the early era of radiometric surveying, the instruments used nowadays are scintillometers, the simplest form of instrument, and spectrometers, a more complex type, that detect gamma rays by their interaction with matter. Small handheld and larger portable spectrometers for ground surveying have internal memories to store the large quantity of data acquired, which is generally restricted to measurements in the K, U, and Th energy windows and the total count (Dentith and Mudge 2014). The presentation of the obtained data in radiometric methods is similar to that of magnetic data. In this sense, the high geochemical mobility of elements such as K and U in surficial environments is the motive for the common use of ratios (U/Th, K/Th) in these maps due to the almost immobile Th. With respect to the presentation of the obtained data in radiometric methods, it is similar to that of magnetic data. As in gravity and magnetic methods, corrections must be also made in radiometric surveys for the effects of scattered thorium radiation in the uranium window and for the effect of both thorium and uranium in the potassium window.

There are over 50 occurring naturally radioactive elements, but the elements of main concern in radiometric studies are uranium (^{238}U), thorium (^{232}Th), and potassium (^{40}K). The latter is common in potassium-rich rocks that cannot be related to concentrations of U and Th. The most abundant radioactive element in the crust is the potassium isotope ^{40}K, which is widely included into the crystal structure of the rock-forming

■ **Fig. 3.21** Instrumentation for ground radiometric (Image courtesy of International Geophysical Technology)

mineral orthoclase. Therefore, potassium can interfere with the existence of valuable mineral deposits, constituting thus a form of noise in this type of surveying. Nevertheless, it facilitates the recognition of potassium salts in evaporites, beach placer horizons in sand, and other economically important deposits.

Ground radiometry (■ Fig. 3.21) was proving very useful in the discovery of major uranium districts in the last decades because this element is essential for nuclear fuels. Nevertheless, the present unpopularity of nuclear power and the availability of uranium from dismantled nuclear bombs made exploration for uranium much less attractive, and the importance of radiometric methods has declined accordingly (Milson 2006). Anyway, this geophysical method is attractive in geology since many rocks can be differentiated from their distinct radioactive signal. The advent of new multichannel detectors, which are capable of separating radiation from different radioactive elements, the better sensitivity and resolution of airborne surveying methods, and the development of new data reduction algorithms have approached airborne radiometric surveys toward new applications. These include detecting and mapping areas of hydrothermal alteration as well as weakly radioactive mineral deposits such as heavy mineral sands (Dentith and Mudge 2014).

Electrical Methods

Electrical methods use direct currents or alternating currents of low frequency to study the electrical properties of the subsurface, being all of these methods ground based. This is in contrast to the electromagnetic methods, described in the following section, which use alternating electromagnetic fields of higher frequency for this purpose. The most commonly measured property is electrical conductivity (Siemens per meter, S/m) or its reciprocal, resistivity (Ohm). In general, these surveys are applied: (a) to locate mineral deposits at shallow depth, (b) to map geological structures, and (c) to trace groundwater table in hydrogeological investigations. There are different methods of electrical surveying: resistivity, induced polarization (IP), and self-potential (SP). Some utilize fields within the Earth (SP), while others need the incorporation of artificially produced currents into the ground (resistivity and IP). In general, resistivity surveys are often accompanied by induced polarization measurements.

Rocks and minerals show widely varying resistivity, with lowest values displayed by clay, saline pore water, acid rock drainage, sulfide ore, native metals, and graphite, whereas common rocks and minerals have low conductivity, being this contrast used in exploration. Thus, the induced polarization method utilizes the capacitive action of the subsurface to identify areas where conductive

Fig. 3.22 Electrodes arrangement (Image courtesy of International Geophysical Technology)

minerals are scattered in their host rocks. The self-potential method uses natural currents present in the ground and originated by electrochemical processes with the aim of finding shallow bodies that display anomalous conductivity. Although the origin of the potentials is not well understood, conductive mineralization can be associated with a negative self-potential anomaly.

Electrical methods are used at regional and prospect scale for direct detection of electrically anomalous targets and, in particular, to detect metal sulfide and metal oxide mineralization. Establishing the depth of the source of the response is problematic in electrical methods. Varying the position of the electrode array and the separation of the electrodes (Fig. 3.22), lateral and vertical variations in electrical properties can be mapped and used to produce data pseudosections, volumes, and maps. However, moving cables, electrodes, and equipment from one point to the next makes these methods laborious and slow. Electrical methods, as with electromagnetic methods, operate much better in the upper few 100 m of the surface with unweathered rocks relatively close to the surface.

As a rule, the resistivity method is scarcely used in mineral resource exploration. Conversely, IP is important in base metal exploration because it depends on the surface area of the conductive mineral grains rather than their connectivity, being successfully employed to a maximum depth of around 600 m. Induced polarization (IP) surveys cause an electric field in the ground and calculate the chargeability and resistivity of the subsurface. Thus, this method is capable of identify changes in the electric currents produced by the existence of different rocks and minerals. IP surveys are conducted along grid lines with readings taken at receiving electrodes planted in the ground and moved from station to station.

IP is especially sensitive to disseminated mineralization that can produce no resistivity anomaly. After magnetic methods, IP technique is probably one of the oldest geophysical methods utilized in

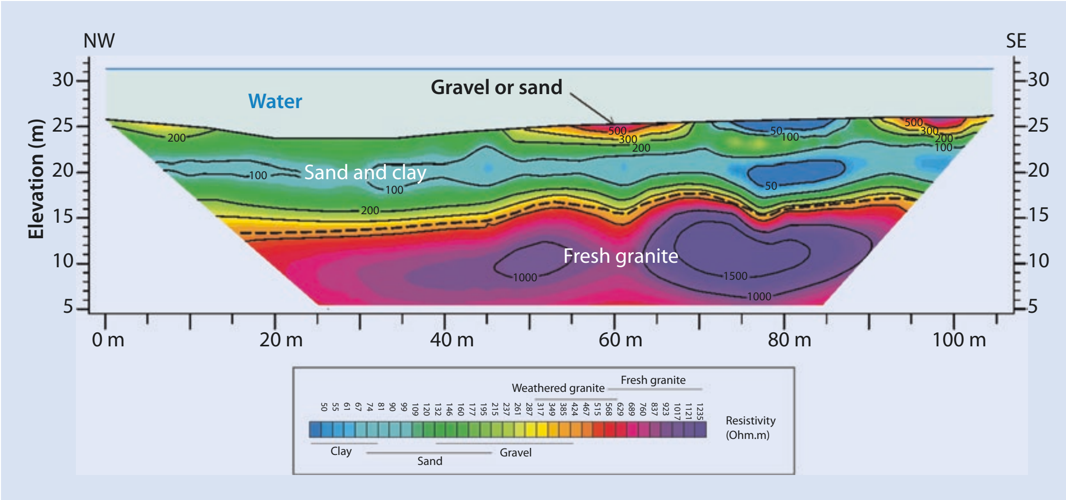


Fig. 3.23 Graphic display of an IP survey (Illustration courtesy of International Geophysical Technology)

mineral deposit exploration. Figure 3.23 shows a graphic display of the interpretation of an IP survey. This method commonly detects sulfide ore minerals (e.g., of Cu and Mo in porphyries) or other minerals that are disseminated in a matrix with high resistivity. Since both massive and disseminated deposits can be identified, IP is very widely used although the method is slow and commonly relatively expensive. In fact, IP is virtually the only geophysical method to detect direct disseminated sulfides in the ground. Examples of the successful use of an IP survey in mineral resource exploration are the detection of the blind, sediment-hosted, lead/zinc sulfide Gortdrum deposit of Ireland (Hitzman and Large 1986) and the discovery of San Nicolas VMS deposit in Mexico (Johnson et al. 2000).

Electromagnetic Methods

Electromagnetic induction (EM) utilizes the induction principle to estimate the electrical conductivity of the subsurface. Thus, EM surveys are based on variations of electric conductivity in the rock mass commonly using an external electromagnetic field, the primary field, and inducing a current to flow in conductive rocks below. These are classified as natural field methods and controlled source methods, respectively. In the latter, a transmitter is used to create a primary alternating electromagnetic field. The passage of current in the general frequency range of 500–5000 hertz (Hz) induces in the Earth electromagnetic waves of long wavelength, which have considerable penetration into the Earth's interior. Induced currents (eddy currents) produce a secondary field in the rock mass. The resultant field can be traced and measured, thus revealing the conductivity of the underground masses.

Electromagnetic methods are often employed as the reconnaissance tools used to identify anomalies for greater detailing because EM instruments provide rapid and easy data collection. As a rule, higher resolution is achieved by using higher frequencies and greater depth penetration by lower frequencies. The problems to analyze the results of EM investigations commonly increase with depth of penetration, and electromagnetic methods thus operate best for ore bodies located as much as 200 m below the surface. Most of the sensor devices of the electromagnetic methods are useful without contact from the ground, having a high operational efficiency in the field.

There are two fundamental categories of electromagnetic measurements: frequency-domain and time-domain measurements. In the frequency domain, a continuous sinusoidal current is used. It is a very sensitive tool' detecting variations as little as 3%. In the time domain, the change in the primary magnetic field is produced by either abruptly turning off or turning on a steady current. It is a powerful transmitter and receiver, and the method can approach the depth, thickness, and

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Fig. 3.24 DHEM survey (Illustration courtesy of International Geophysical Technology)

conductivity of layers down to 300 m below surface. Unlike conventional resistivity techniques, EM works without a physical contact to the ground, without electrodes, which is in advantage for use above ice, water, swamps, frozen, or arid ground.

EM surveys are conducted from the air (AEM), on the ground surface, and in drillholes (downhole electromagnetics – DHEM) (Fig. 3.24). Ground-based EM methods are relatively expensive, being used mainly to define targets for drilling in specific mineralization styles. For its part, airborne investigations incorporating this geophysical method have been commonly utilized for direct ore location and sometimes in regional geological mapping. They were originally developed in the frequency domain to detect conductive massive sulfide bodies within the resistive rocks of the Precambrian shield of Canada. The subsequent need to explore other kinds of geological environments, combined with developments in EM systems, has led to higher-sensitivity time-domain systems now being used almost exclusively for mineral exploration. Airborne electromagnetic surveys are used in mineral exploration to discover mineral deposits such as sulfides containing copper or lead, magnetite, pyrite, unconformity-style uranium mineralizations, kimberlite pipes, certain manganese minerals, and paleochannels as potential hosts for placer deposits and sandstone- and calcrete-hosted uranium deposits (Dentith and Mudge 2014). On the other hand, the discovery of massive sulfide deposits that form major base metal producers in eastern Canada is immediately related to the development of airborne electromagnetic surveys (Lulin 1990).

EM surveys can be also applied in drillholes (DHEM) measuring the effects of currents flowing between the drillhole and the surface or between contiguous holes. This method can reduce the amount of delineation drilling required. In general, DHEM is one of the most important geophysical tools in the exploration for conductive massive sulfide mineralization, especially deep nickel sulfide bodies. For many reasons (e.g., many host rocks and mineralization can give a similar geophysical signal), electromagnetic methods are useful in locating ores in some regions of the world where fresh and not oxidized rocks are present near the surface. An example of this type of regions is the recently glaciated areas of North America, northern Europe, and Russia.

Besides the described techniques, the magnetotelluric method (MT) is a passive electromagnetic technique used for exploring the conductivity structure of the Earth from tens of meters to a depth of more than 10,000 m. It is a survey method that utilizes the Earth's telluric current produced in the ground by variations of the Earth's magnetic field. The main applications of this technique are in hydrocarbon exploration. Finally, ground-penetrating radar (GPR) is a geophysical method that uses radar pulses to image the subsurface, being utilized in rock, soil, ice, fresh water, structures, etc. It can detect changes in material as well as voids and cracks.

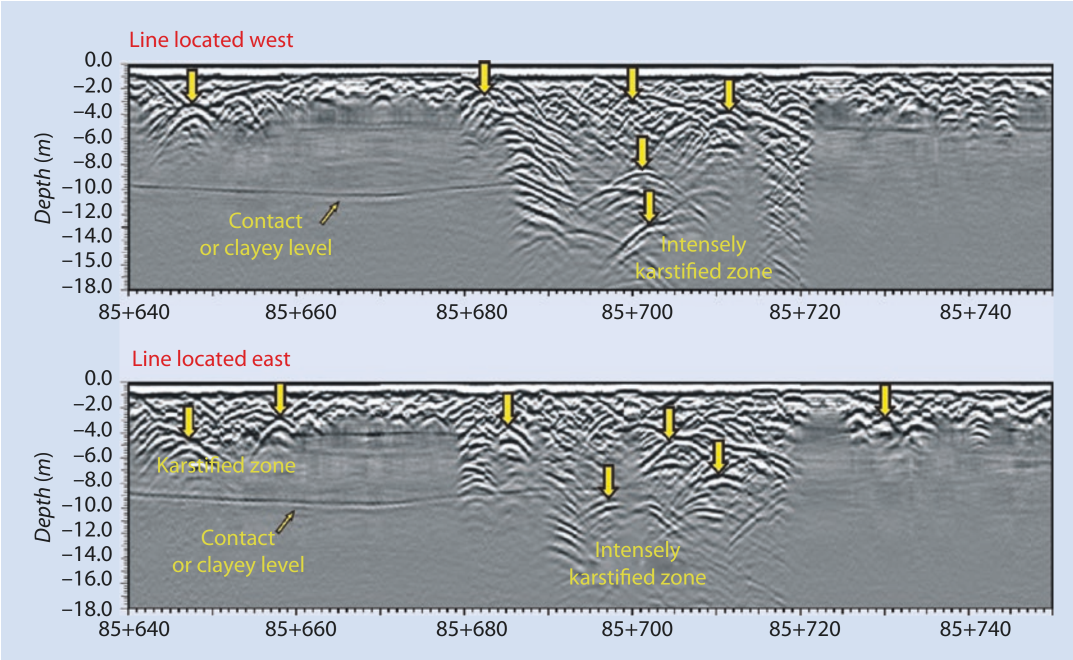

Fig. 3.25 Results of a GPR survey (Illustration courtesy of International Geophysical Technology)

This method has a great similarity with seismic method and may be considered as a mini reflection seismic survey. Figure 3.25 is an example of a GPR survey.

Seismic Methods

Seismic methods are based on measurements of the time interval between initiation of a seismic (elastic) wave and its arrival at detectors in order to obtain an image of the subsurface. The seismic wave can be generated by an explosion, a dropped weight, a mechanical vibrator, a bubble of high-pressure air injected into water, and other sources. The seismic wave is detected by a geophone on land or by a hydrophone in water. Since seismic waves (e.g., P-waves and S-waves), which propagate with different velocities in different rock types, are reflected and refracted at bedding or fault contacts, reflection and refraction are the most commonly used seismic techniques. Refraction methods use simpler equipment (Fig. 3.26) and need less processing than reflection methods. Compared with other geophysical methods, the seismic method, in any of its forms, is rarely used in mineral exploration.

Most seismic work uses reflection methods because they produce better resolution than other techniques, with the exception of measurements made in close proximity (e.g., as with borehole logs). Seismic methods dominate oil industry since reflection seismic is the most important geophysical method to prospect for oil and gas at greater depths. As aforementioned, these techniques are comparatively little used in mineral exploration, mainly due to their high cost and because most mineralizations in igneous and metamorphic rocks display incoherent layering. Applications of these techniques include searching offshore placers or subsea resources of bulk minerals such as sands and gravels. Where ores occur in sedimentary rocks that are just gently folded or faulted, seismic surveys can be useful. However, reflection work onshore is slow and expensive because geophones have to be positioned individually by hand and sources can need to be buried. The use of reflection in onshore exploration for solid minerals other than coal is consequently rare, although Witwatersrand gold reefs, flat-lying kimberlite sills, and some deep nickel sulfide bodies have all been investigated in this way (Eaton et al. 2003).

Fig. 3.26 Seismic refraction survey (Image courtesy of International Geophysical Technology)

Fig. 3.27 Electromagnetic airborne survey (Image courtesy of Geotech)

Likewise, the mining industry uses detector and/or seismic sources located in the subsurface, with access provided by drillholes or underground workings. Thus, seismic surveys can map mineralization between drillhole intersections and are used for exploration at a prospect scale and during mining. Seismic survey also utilized seismic waves that are deliberately guided through coal seams to gather information of its characteristics prior to mining (Dentith and Mudge 2014).

Airborne Geophysics

Magnetic, electromagnetic (Fig. 3.27), gamma-ray, and more recently gravity measurements do not need physical contact with the ground; therefore, they can be carried out from aircraft. Obviously, there is a loss of sensitivity because detectors are far away from sources. The main merit of airborne work is that it enables coverage of large areas quickly and inexpensively per unit area. Moreover, airborne surveys measure physical properties of rocks and ores through dense vegetation, swamps, lakes, and soils, among many others. They are usually part of the reconnaissance phase of mineral resource exploration, although some modern airborne systems offer higher resolution by surveying very close to the ground and can find application in the later stages of exploration. Airborne geophysical surveys are typically undertaken using low-flying helicopters or light aircraft that fly in a grid pattern, being the instruments mounted on the aircraft or positioned underneath. According to the survey type, the aircraft can fly ranging from 20 to 70 m above the ground and the flight lines can be delimited from 20 to 200 m apart. Airborne investigations can be flown either at a constant altitude or at a nominally constant height above the ground, which is more common in mineral exploration.

Since airborne methods need a very good navigational control, airborne surveys have been completely transformed by the use of global positioning satellites (GPS). With this instrument, velocities can be estimated with great accuracy, making airborne gravimetry, which requires velocity corrections, usable for the first time in mineral exploration. In the near future, pilotless drones can fly some airborne surveys, especially aeromagnetic surveys. In fact, drones are already used in topographic applications (◘ Fig. 3.13).

The most frequently used first stage in geophysical exploration includes the aeromagnetic survey. Thus, a magnetometer or a series of magnetometers attached to an aircraft estimate the intensity of the Earth's magnetic field, producing the detection of magnetic anomalies originated by the minerals present in the ground. Among other factors, the resolution of the data is dependent upon (a) the distance between the traverse line spacing, (b) the distance between the aircraft and the ground, (c) the magnetic signature of the aircraft itself, and (d) variations in the diurnal activity.

Airborne electromagnetic surveys generate the strongest EM responses from massive sulfides and can use man-made primary electromagnetic fields to measure the electromagnetic properties of rocks. Very low-frequency EM system can be useful as a mapping tool, particularly when combined with magnetics. Finally, airborne gravimetry measures the changes in the gravity field with an airborne gravimeter on a helicopter or an aircraft. It involves using ultra-sensitive equipment, called a gravimeter, to look at the structure density of rock in the subsurface of the Earth. New generation gravimeters back out the movement of the aircraft from the data, providing a more accurate measurement. Once corrections are made to the data, critical information can be derived for mapping purposes.

◘ **Fig. 3.28** Caliper geophysical borehole logging (Image courtesy of Robertson Geologging)

Borehole Geophysical Logging

Because exploration drillholes are usually cored completely, the industry has been slow to identify the importance of geophysical borehole logging (◘ Fig. 3.28). However, since drilling is expensive, geophysical borehole logging is essential to obtain the maximum possible information from each drillhole; in this sense, the geophysical characteristics of the rocks surrounding a borehole are often the best guides to discover the existence of ore (◘ Box 3.5: Borehole Geophysical Logging). Borehole geophysical surveys result in the higher resolution of data, especially in conjunction with geological, physical, and chemical core logging results (Ellis and Singer 2007). Downhole geophysical surveys increase the radius and depth of investigation and provide greater resolution of buried targets. For instance, in the uranium industry, borehole logging is actually a basic tool in the exploration and delineation of uranium deposits (Mwenifumbo and Mwenifumbo 2013).

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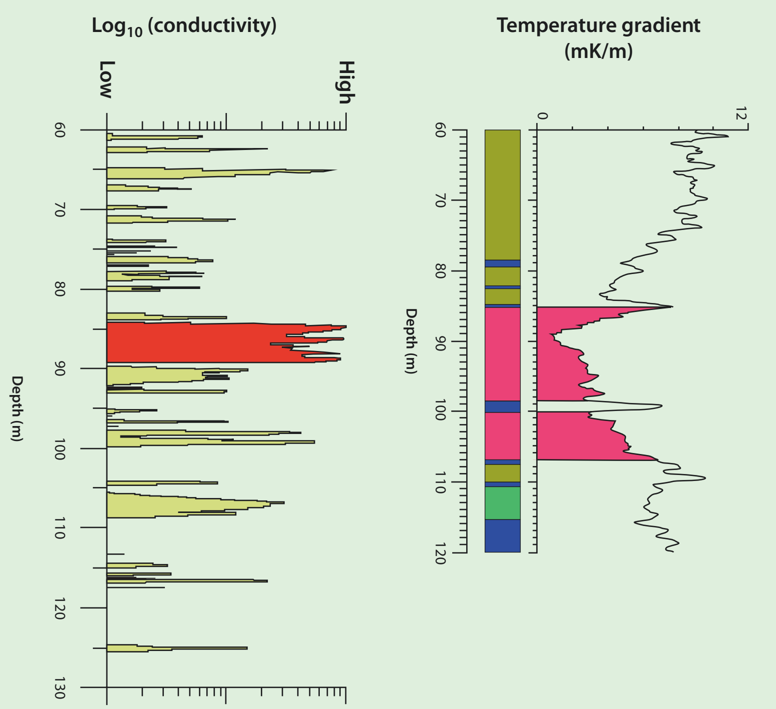
Box 3.5

Borehole Geophysical Logging

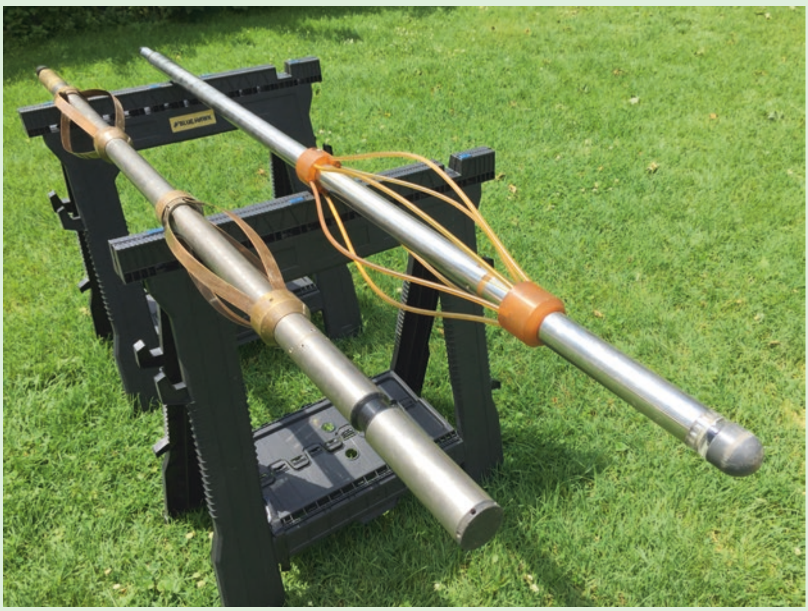
Most geophysical techniques can be modified for use in boreholes. Thus, borehole geophysics is the science of recording and analyzing measurements of physical properties made in boreholes. Compared to geophysical measurements made on the ground surface, they have better resolution in the depth dimension. Probes that measure different properties are lowered into the borehole to collect continuous or point data that is graphically displayed as a geophysical log (◘ Fig. 3.29). Multiple logs typically are collected to take advantage of their synergistic nature because much more can be learned by the analysis of a suite of logs as a group than by the analysis of the same logs individually. The primary components of a geophysical logging system include the probe, cable, winch, wellhead pulley assembly at the top of the hole, a depth counter, and the surface recording instrumentation that displays the data and usually supplies the power to the probe. In most cases, the probe sends information up to the surface in real time, either wirelessly or via the cable. A string of different probes can be connected to collect more than one type of geophysical information. Borehole geophysical measurements are made by sensors (receivers/detectors) that are housed inside a probe. The probe is lowered downholes in which the measurements are to be made. A series of continuous measurements are made with the data transmitted to the surface. The logging speed is commonly about 6 m/min. Data sampling rates range from one sample to five samples and provide measurements every 2–10 cm along the hole.

There are two quite distinct modes of making downhole measurements: downhole logging and downhole surveying. The first is used where the in situ physical properties of the rocks penetrated by a drillhole are measured to produce a continuous record of the measured parameter. Measurements of several physical parameters, producing a suit of logs, allow the physical characterization of the local geology. Despite the valuable information obtainable, multiparameter logging is not ubiquitous in mineral exploration, but its use is increasing along with integrated interpretation of multiple geophysical data sets. On the other hand, downhole surveying is designed to investigate the larger region surrounding the drillhole, with physical property variations obtained indirectly, and to

◘ Fig. 3.29 Borehole geophysical logs



indicate the direction and even the shape of targets. For instance, downhole electromagnetic survey detects conductivity variations, probably owing to mineralization, in the volume surrounding the drillhole. Downhole geophysical surveys increase the radius and depth of investigation and provide greater resolution of buried targets. For instance, exploration of an iron ore body could be improved by a density log. The located mineralization can be split into layers of magnetite and hematite using a magnetic susceptibility log. Common geophysical logs and/or probes include caliper, gamma, single-point resistance, spontaneous potential, normal resistivity, induced polarization, fluid resistivity, temperature, flowmeter, television, and acoustic and optical televiewer (◘ Fig. 3.30). For instance, the caliper probe (◘ Fig. 3.28) measures the diameter of the borehole as a continuous record against depth and is used as a check of borehole condition before casing operations or before running more expensive logging probes. Gamma logs record the amount of natural gamma radiation emitted by the rocks surrounding the borehole; clay- and shale-bearing rocks commonly emit relatively high gamma radiation because they include weathering products of potassium feldspar and mica and tend to concentrate uranium and thorium by ion absorption and exchange. The optical televiewer probe gets optical views of the wall and is useful in locating structures such as faults and also bed boundaries where there is a significant change in rock formation colors. Acoustic televiewer tools have a transmitter that scans the borehole wall with an acoustic beam, and the acoustic energy reflected at the borehole fluid and rock interface is recorded by a receiver.



◘ Fig. 3.30 Acoustic and optical televiewer probes (Image courtesy of Enviroscan)

3.4.5 Geochemical Exploration

Introduction

In geochemical exploration, anomalous surface enrichments of elements that point to potential mineral deposits in the subsurface are sought. For this reason, geochemical surveys play an essential role in mineral exploration because it is an essential component in most modern integrated mineral exploration programs. Geological mapping and geophysical surveys are usually carried out simultaneously with geochemical exploration. Several elements caused the quick development of geochemical exploration during the twentieth century. First, most metallic mineral deposits are surrounded by zones of uncommon trace element concentrations in the nearby and enclosing rocks. Thus, chemical deviations can be expressed by enrichment or depletion of certain minerals, elements, isotopes, etc. On the other hand, geochemical exploration has gained widespread acceptance with the development in the last decades of rapid, sensitive, and accurate analytical methods. This type of mineral resource exploration is conducted at several scales, from regional reconnaissance to very detailed local sampling at high sampling

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Fig. 3.31 Soil for geochemical sampling (Image courtesy of Mari Luz García)

densities. Most exploration programs begin with regional stream sediment sampling followed by soil and then rock sampling. Geochemical surveys in mineral exploration are based on two features of an ore deposit: (1) association with abnormal concentrations of elements in the nearby rocks, and/or (2) association with secondary scattered patterns of elements in the surficial materials of their weathering and erosion; it substantially increases the area in which evidence for the presence of a mineral target can be detected.

The most commonly measured chemical property in a mineral or rock is the trace content of an element or cluster of elements. The analyzed material can be rock, soil (Fig. 3.31), gossan, glacial debris, vegetation, stream sediment, or water. Thus, «the purpose of the measurements is the discovery of a geochemical anomaly or area where the chemical pattern indicates the presence of ore in the vicinity» (Hawkes 1957). Obviously, the absence of such anomalies helps to eliminate areas for further consideration. However, it is essential to bear in mind that the basic geochemical question required 60 years of discussion: what constitutes a geochemical anomaly and how can this be enhanced (sample processing and analytical methods) and detected (a number of univariate and multivariate mathematical techniques)? (Cohen et al. 2007).

The modern techniques of geochemical prospecting originated in the Soviet Union and Scandinavia where extensive research was conducted during the 1930s (Garret et al. 2008). Modern surveys are conceptually similar to earlier surveys but are considerably more complex in their details. This complexity in modern geochemical surveys arises from several sources (Adcock et al. 2013): (1) the number of samples collected in a single survey can sometimes reach several thousand, and national- or continental-scale projects can involve numerous surveys carried out over several years. (2) different sample types (e.g., glacial sediment, multiple soil horizons, water, vegetation) can be collected at each sampling site. (3) a sample can be processed in various ways (crushing, sieving, heavy and/or magnetic mineral separation, washing, etc.) before being subjected to chemical analysis. (4) the sample can be analyzed by a variety of different methods, by different laboratories, over a time period of years or even decades. A clear example of this complexity is the China Geochemical Baselines project, which was carried out in 2008, being sampling completed in 2012. The main goal was to establish the abundance and spatial distribution of chemical elements throughout the whole China. Running the project, 6617 samples from 3382 sites were collected across the country (Wang 2015).

In general, a geochemical survey is divided into the following phases: (1) planning, (2) sampling, (3) chemical analysis, and (4) interpretation of data. As a general rule, samples are collected in

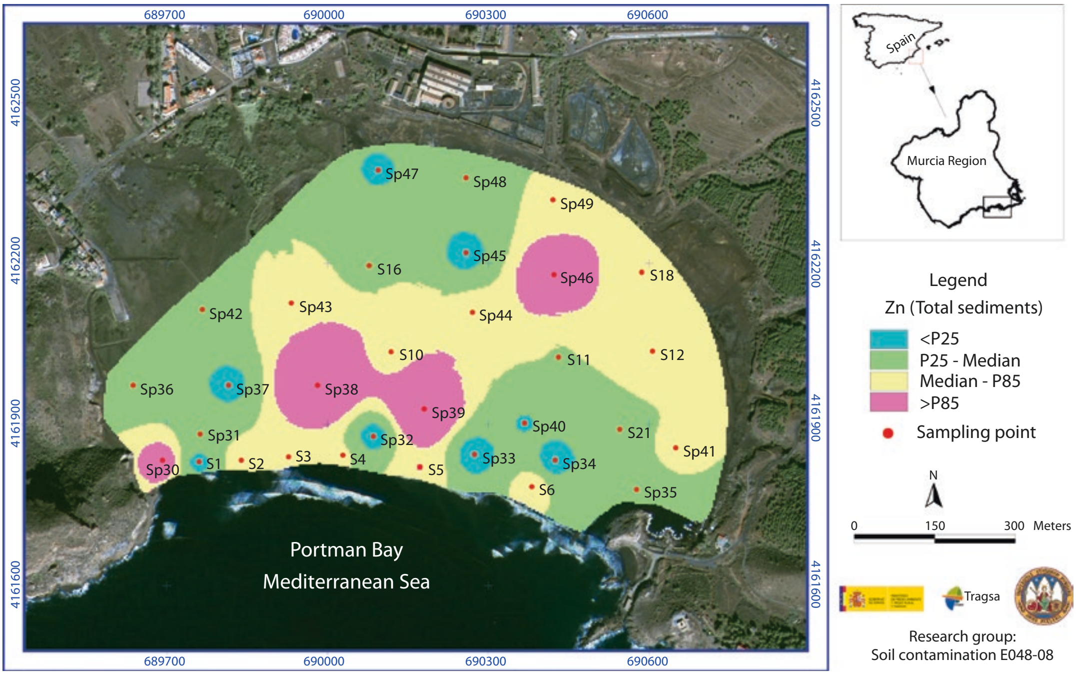


Fig. 3.32 Color contour map of a geochemical survey (Spain) (Illustration courtesy of Mari Luz García)

the field. They are brought to a laboratory facility where they are subjected to preparation prior to analysis, including crushing, sieving, drying, and filtering. The prepared samples are then sent to different laboratories for chemical analysis. Where the data are returned, they are verified and reported typically in a spreadsheet format, with a set of rows and columns. Finally, these data are processed using different statistical methods (e.g., multivariate analysis) and displayed commonly as color contour map (Fig. 3.32). The objective is to establish a geochemical anomaly that separates the mineral deposit from enhancements in background and nonsignificant deposits.

Primary and Secondary Geochemical Anomalies

Geochemical anomalies are commonly divided into primary or secondary. The primary geochemical anomalies are formed as by-product of the processes that concentrate ore; they are larger than the ore target itself. As defined originally by Safronov (1936), «the primary halo of a mineral deposit is an area including rock, surrounding mineral deposit (ore bodies) and enriched elements that make up that deposit.» In general, primary dispersion halos are produced in the host rocks at the time of ore formation. For instance, some of the fluid permeates into wall rocks in hydrothermal deposits causing different alterations which include chemical changes. Halos of this type are very useful in exploration since they can commonly be hundreds of times larger than the mineralization they surround. Moreover, they extended both laterally and vertically, hence being easier to locate. Primary dispersion halos have a great variety of sizes and shapes due to the numerous variables that influence fluid movement in rock. Thus, some halos can even be identified at distances of hundreds of meters from their related mineralization. The factors that control the development of primary halos are manifold: fractures in the host rock, porosity and permeability of the host rock, inclination of mineralizing fluids to react chemically with the host rock, and so on. Obviously, the composition and distribution of these primary halos depend on the type of deposit. For instance, porphyry copper deposits usually display chemical halos that measure hundreds of meters horizontally and vertically.

Since trace elements of mineralization and their linked primary halos are commonly discharged by weathering processes to soils, overburden, and vegetation, they generate a subsequent generation of enrichment called secondary halos. Thus, secondary geochemical anomalies or halos are formed by processes that acted on the deposit after its formation. These types of halos are generated by mechanical breakdown and chemical dissolution of rocks and ores. Chemical weathering involves breakdown of rocks and minerals by chemical means with further discharge of their contained trace elements to the environment. It requires abundant water, oxygen, and carbon dioxide. In general, chemical weathering is more abundant in tropical regions although it can also be substantial in temperate areas. In turn, physical weathering includes all processes of rock disintegration not involving chemical changes, being more frequent in very cold or hot arid regions.

Mechanical breakdown and further transport in surface water runoff concentrate resistant minerals such as cassiterite, rutile, monazite, diamonds, gold, etc. Therefore, anomalies are detected by heavy mineral panning of stream sediments or soils. Other minerals can be dissolved and the metals can be either redeposited locally or carried away into solution by ground- and surface water. Groundwater frequently dissolves some of the constituents of mineralized bodies that can be transported along considerable distances before eventually emerging in springs or streams. During dispersal, the elements can be reconcentrated in vegetation, on clay minerals, or in organic matter, all of which are attractive sampling media in geochemical exploration. Regarding the vegetation, some metals in solution can be collected by plants and trees and then concentrated in the living tissue. In some cases, the element that originates the most important primary halo is not necessarily the one of greatest economic significance in the mineralization.

Mobility is an indicator of how far an element can go dissolved in water, broadening the signal originated from the mineral deposit. For this reason, the usefulness of the mobility of an element is essential in geochemical prospection. This type of element is commonly referred to as a pathfinder. The pathfinders are very useful in geochemical exploration since their halo is generally greater than that of the element with the most economic interest or because it can be identified more easily by classical analytical procedures (Table 3.3). For instance, arsenic is commonly utilized as a pathfinder in exploration for gold. The choice of pathfinder elements/metals depends on many factors such as consistency of association with the ore deposits sought, characteristics of primary dispersion, and ease with which geochemical analysis can be performed (Levinson 1974).

The variable mobility of elements is of great significance in the process that causes secondary dispersion. Elements with high mobility under surficial conditions enlarge the anomalous zone. For instance, a project targeting polymetallic deposits of Pb, Zn, and Cu would use mobile Zn for regional sampling with a low density, whereas dense sampling of Zn anomalies for Cu and Pb should reveal the drilling targets (Pohl 2011). In this context, there are many important properties in the elements such as electronic configuration, ionic potential, pH and Eh, trend to originate complexes with organic matter, and trend to coprecipitate or to be absorbed with iron or manganese hydroxides.

The mobility of elements in secondary dispersion is strongly influenced by factors including the nature of rocks, climate, vegetation, relief, and groundwater flow. Thus, in cold climate, large and well-defined anomalies do not develop because chemical dissolution is inefficient and drainages are poorly developed; in dry, arid climate, chemical dissolution is ineffective and dispersal by occasional flash floods does not lead to the formation of well-defined anomalies. By contrast, in tropical climate decomposition and leaching of the ore-forming elements can be so complete that no traces of the metals remain in weathered rocks or soils. Therefore, the best environment for geochemical exploration is located in a temperate climate in regions of gentle topography, in which abundance of water and warm temperatures leads to effective dissolution of ore minerals and the gentle topography fosters both chemical dissolution and the development of good secondary dispersion halos (Gocht et al. 1988).

Stream Sediment Sampling

Stream sediment geochemical surveys are the cornerstone of all types of reconnaissance exploration, mainly in regions undergoing active weathering.

Table 3.3 Major components and pathfinders for different types of mineral deposits (Moon 2006)

Type of deposit	Major components	Pathfinders
Magmatic deposits		
Chromite ores (Bushveld)	Cr	Ni, Fe, Mg
Layered magnetite (Bushveld)	Fe	V, Ti, P
Immiscible Cu–Ni–sulfide (Sudbury)	Cu, Ni, S	Pt, Co, As, Au
Pt–Ni–Cu in layered intrusion (Bushveld)	Pt, Ni, Cu	Sr, Co, S
Immiscible Fe–Ti–oxide (Allard Lake)	Fe, Ti	P
Nb–Ta carbonatite (Oka)	Nb, Ta	Na, Zr, P
Rare-metal pegmatite	Be, Li, Cs, Rb	B, U, Th, rare earths
Hydrothermal deposits		
Porphyry copper (Bingham)	Cu, S	Mo, Au, Ag, Re, As, Pb, Zn, K
Porphyry molybdenum (Climax)	Mo, S	W, Sn, F, Cu
Skarn-magnetite (Iron Springs)	Fe	Cu, Co, S
Skarn–Cu (Yerington)	Cu, Fe, S	Au, Ag
Skarn–Pb–Zn (Hanover)	Pb, Zn, S	Cu, Co
Skarn–W–Mo–Sn (Bishop)	W, Mo, Sn	F, S, Cu, Be, Bi
Base metal veins	Pb, Zn, Cu, S	Ag, Au, As, Sb, Mn
Sn–W greisens	Sn, W	Cu, Mo, Bi, Li, Rb, Si, Cs, Re, F, B
Sn–sulfide veins	Sn, S	Cu, Pb, Zn, Ag, Sb
Co–Ni–Ag veins (Cobalt)	Co, Ni, Ag, S	As, Sb, Bi, U
Epithermal precious metal	Au, Ag	Sb, As, Hg, Te, Se, S, Cu
Sediment hosted precious metal (Carlin)	Au, Ag	As, Sb, Hg, W
Vein gold (Archaean)	Au	As, Sb, W
Mercury	Hg, S	Sb, As
Uranium vein in granite	U	Mo, Pb, F
Unconformity associated uranium	u	Ni, Se, Au, Pd, As
Copper in basalt (L. Superior type)	Cu	Ag, As, S
Volcanic-associated massive sulfide Cu	Cu, S	Zn, Au
Volcanic-associated massive sulfide Zn–Cu–Pb	Zn, Pb, Cu, S	Ag, Ba, Au, As
Au–As rich Fe formation	Au, As, S	Sb
Mississippi Valley Pb–Zn	Zn, Pb, S	Ba, F, Cd, Cu, Ni, Co, Hg
Mississippi Valley fluorite	F	Ba, Pb, Zn
Sandstone-type U	U	Se, Mo, V, Cu, Pb
Red bed Cu	Cu, S	Ag, Pb
Sedimentary types		
Copper shale (Kupferschiefer)	Cu, S	Ag, Zn, Pb, Co, Ni, Cd, Hg
Copper sandstone	Cu, S	Ag, Co, Ni
Calcrete U	U	V

Stream sediment sampling can be applied only if a well-developed drainage system is present. In this technique, stream sediments are taken from active stream channels and studied to find anomalous element concentrations since the sediment sample from an active riverbed is considered to represent an average of its upstream watershed. Thus, the objective is to obtain sample(s) representative of the catchment area. The relatively easy use of the method leads to a quick evaluation of regions at fairly low cost.

Small streams give maximum resolution and sharpest contract, as opposed to large streams in which any anomaly from a mineralized zone will be diluted by large amounts of stream sediment from barren areas. Sampling densities are about one sample per two square kilometers in regional reconnaissance programs. In more detailed investigations, higher density of sampling density is usually carried out depending on the local conditions and the characteristics of the target. For instance, sampling densities can range from one sample over 100 km^2 in reconnaissance studies to a few samples per km^2 in more specific follow-up. In general, the values of the background and anomalous element concentrations are computed statistically, and metal distributions are illustrated in geological maps. Previously, samples are sieved to 80 mesh (0.157 mm) and the fine fraction is analyzed since it reflects better metal anomalies. It is important to remember that the coarser fractions commonly include pebbles, which are usually depleted in trace elements.

Panning heavy minerals to obtain concentrates is also a classical method of stream sediment studies, being a very useful geochemical prospecting technique (e.g., panning for gold). Panning refers to the process in which a sample is agitated in water to separate minerals by their specific gravity. Thus, heavy mineral panning is widely employed for searching native elements such as gold grains, platinum, diamonds, and heavy resistant mineral grains such as magnetite, zirconium, ilmenite, rutile, monazite, and cassiterite. If positive results are obtained, immediately follow-up campaign is carried out to look for the source of the anomaly.

Soil Sampling

Soil sample geochemistry is a powerful tool in the exploration of anomalies found by stream sediment investigations. The method works very well since weathering and leaching of buried deposits can discharge anomalous concentrations of elements to soil and groundwater. The released heavy metals spread outward and originate a dispersion halo in the soil that is much larger than the mineral deposit itself. As a rule, the dispersion halos in soils are smaller than those in stream sediments but larger than those in primary halos in rocks. Similar to stream sediment sampling, samples collected (▣ Fig. 3.33) are commonly the fine silty or

▣ **Fig. 3.33** Soil sampling (Image courtesy of Mari Luz García)

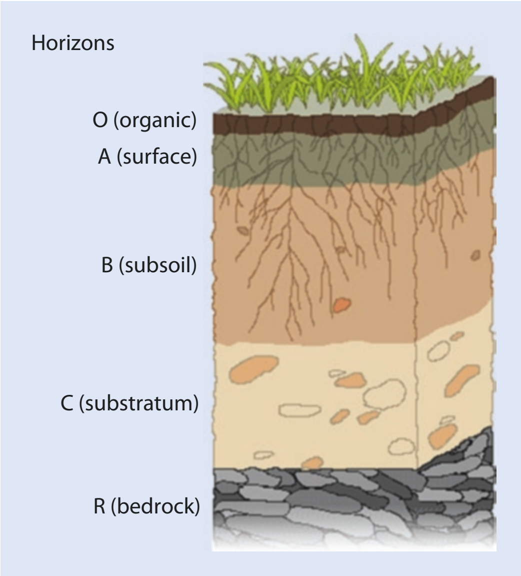


Fig. 3.34 Soil profiles

clayey material that results from weathering of the underlying bedrock, being usually obtained just below the organic-rich surface grassroots layer.

It is a comparatively expensive technique and is typically carried out in detailed exploration where it is used to identify specific targets for drilling. Soil geochemistry surveys can be performed on a regional basis in areas without well-developed drainages to allow stream sediment surveys. The dispersion halos of elements in soils are much smaller than those in stream sediments but still considerably larger than those in primary halos in rocks. Soil sampling is especially recommended in areas of residual soil over any bedrock and in areas with soil developed on in situ regolith.

Elements can accumulate in different forms within a soil profile (Fig. 3.34). Traditionally, B-horizon has represented a position where elements have concentrated as minerals such as silicates, iron oxyhydroxides, and carbonate crusts. This preconcentration can represent an ideal sample material for collection. It is also the most homogeneous horizon and provides the best sampling medium. The C-horizon, which is closest to the rock, generally shows little dispersion of the target elements. The A-horizon, the uppermost soil horizon, can show the largest dispersion, but a variable content of organic matter leads to irregular element distribution. AS analytical technology has advanced, becoming increasingly sensitive, new possibilities have been developed related to the positions where the target and pathfinder elements can be analyzed.

Regarding the sample spacing in soil sampling, sampling density and patterns are determined by the style of target, stage of exploration, topography of the exploration area, prospective geology, and orientation of the anomaly. Soil samples are typically collected on a rectangular pattern, generally with closer spacing of sample sites along more widely space sample lines. The optimum spacing between sampling lines and sample sites will depend on the purpose of the survey and the expected size of the dispersion halo to be detected. For instance, usual sample spacing for reconnaissance studies ranges from 200/400 to 400 m. For more detailed anomaly investigation, samples are taken at 100 m intervals on 200 m spaced lines. In this case, an infill sampling down to 50 m on 100 m spaced lines is commonly carried out. The main goal is to acquire at least two samples from the detected anomaly on a sampling line.

Water Sampling

In general, water samples collected from springs, wells, boreholes, and streams are rarely useful for mineral deposit exploration. According to Pohl (2011), dissolved metal content in water is usually very low, in the ppb range, and varies strongly with pH and Eh, which makes interpretation difficult. Thus, the concentration of elements of geochemical interest is very low compared to that in stream sediments. For these uncertainties, hydrogeochemistry is not a widely method used in exploration. However, high concentrations of chemical elements can be found in groundwater. For this reason, groundwater surveys are preferred to surface water studies. Groundwater surveys are commonly conducted in conjunction with soil studies for detailed surveys. Thus, groundwater has the potential to be a powerful mineral exploration tool for different considerations: «(1) recent advances in analytical methods have resulted in lower detection limits; (2) groundwater is chemically reactive with mineralization and host rocks, in particular where water is O_2-bearing; (3) groundwater flows away from the site of reaction with mineralization, providing a potentially broader exploration target than lithogeochemistry; and (4) for many species of interest, background concentrations are low, enhancing anomaly contrast» (Leybourne and Cameron 2007). Interpretation of groundwater geochemistry in mineral exploration is easier if data related to the local and regional hydrology is available.

Rock Sampling

Rock geochemical surveys seek the primary dispersion halo around mineral deposits. Because this type of halos is restricted to a small area immediately surrounding any prospective mineral deposit, rock surveys are mainly applied to evaluate specific targets outlined by regional surveys. Although this technique has been also applied with relatively good results in regional reconnaissance, it becomes most effective in detailed campaigns (Moon 2006), being rock sampling included in the techniques devoted to for follow-up mineral exploration. It provides direct evidence about the geochemical characteristics of the rocks that cause the anomaly, helping in the geological interpretation of stream sediment and soil surveys (Govett 1983).

Geochemical exploration with rock samples or selected minerals is based on specific geological-petrological models. Examples include regional sampling of granites in order to locate fertile intrusions, discrimination of prospective and barren porphyries by analyzing copper in biotite, and identification of rare metal pegmatites by muscovite analysis (Pohl 2011). On a regional basis, the most successful applications deal with delineation of mineralized felsic plutons and exhalative horizons because these plutons with mineralization of copper and tungsten are commonly enhanced in these elements, although invariably display a high variability inside the pluton. For instance, tin mineralization associated with highly evolved and altered intrusive bodies is delineated examining the geochemistry of minerals such as micas.

Biogeochemical Sampling

Biogeochemistry is a viable first-pass exploration method, and it can show multi-element halos at small scale, being more refined if more detailed exploration methods are carried out in the target. Biogeochemical sampling is a relatively cheap, efficient, and environmentally passive method in the initial stages of mineral exploration programs (Reid and Hill 2010). Biogeochemical techniques utilized in mineral deposit prospecting are based on soil and plant relationships. In this sense, plants incorporate elements from soil and groundwater into their branches and leaves, and this absorption of trace elements depends on the plant species, plant organs, grow stage, and soil type. Biogeochemical exploration with sampling and chemical analysis of plant tissues has been utilized extensively in Canada and Russia and more recently in Australia (Närhi et al. 2014).

Plant samples (▣ Fig. 3.35) have benefits compared to other sample media in terms of providing data that represent a broad area, due to their deep spreading root systems. Biogeochemical

▣ Fig. 3.35 Biogeochemical sampling of plants (Image courtesy of Andrea Castaño) exploration «relies on the fact that plant roots penetrate soil horizons, have access to weathered/fractured bedrock and associated groundwater, and accumulate elements in their organs» (Dunn 2007). Accordingly, if some plant organs include excessive amounts of particular metals, they can be used as indicators of ore zones in bedrock for geochemical exploration (Brooks et al. 1995). Plant growing on soil is dramatically affected by the host soil composition, which leads to the selection of specific flora. Thus, plants answer to elemental composition of soil in three ways: exclusion, indication, and accumulation (Rajabzadeh et al. 2015). Biogeochemists use soil indicator plants for prospecting ore deposits. For instance, because serpentine plants have been studied and ultramafic rocks are profuse on the crust of the Earth, plants growing on serpentinized materials are satisfactorily utilized in biogeochemical exploration (Freitas et al. 2004).

Gas Sampling

At present, mineral deposits susceptible to be prospected are commonly buried deep below the surface of the Earth. However, the alteration and oxidation of a deposit release gaseous components that can be detected at the surface using gas samples from soil or down drillholes. This method can identify a few different gases if they are present in sufficient amount (e.g., mercury, oxygen, CO_2, and radon) (Fig. 3.36). The characteristics of these gases and their concentration can provide hints on minerals occurring at depth and, consequently, where a mineral deposit can be present.

Gases are potentially an attractive medium to sample because they can diffuse through thick overburden. Thus, a number of gases have been used in mineral exploration: sulfur vapors indicate the presence of sulfide deposits, radon gas guides to uranium deposits, and gaseous hydrocarbons reflect the presence of petroleum and natural gas. However, mercury has been the most successful gas studied in mineral prospecting due to mercury is the only metallic element that constitutes a vapor at room temperature. Thus, it is widely present in sulfide deposits, particularly volcanic-associated base metal deposits. Enrichment of carbon dioxide and depletion of oxygen produced by weathering of sulfide mineral deposits have been tested recently. The results are commonly inconsistent due to the large changes in gas concentration (partial pressure) caused by variations in environmental conditions, specifically changes in pressure and rainfall.

Indicator Minerals

Indicator minerals are mineral species transported as grains in clastic sediments and indicating the presence in bedrock of a specific type of mineralization, hydrothermal alteration, or lithology (Fig. 3.37). The preservation and identification of these minerals is provided by their physical and chemical characteristics, including relatively high density. They are quickly recuperated at the parts per billion level from stream, alluvial, glacial, or aeolian sediments or soils producing large exploration targets. Indicator mineral methods differ from

Fig. 3.36 Measurement of soil radon using a soil gas probe (Image courtesy of DURRIDGE Company Inc.)

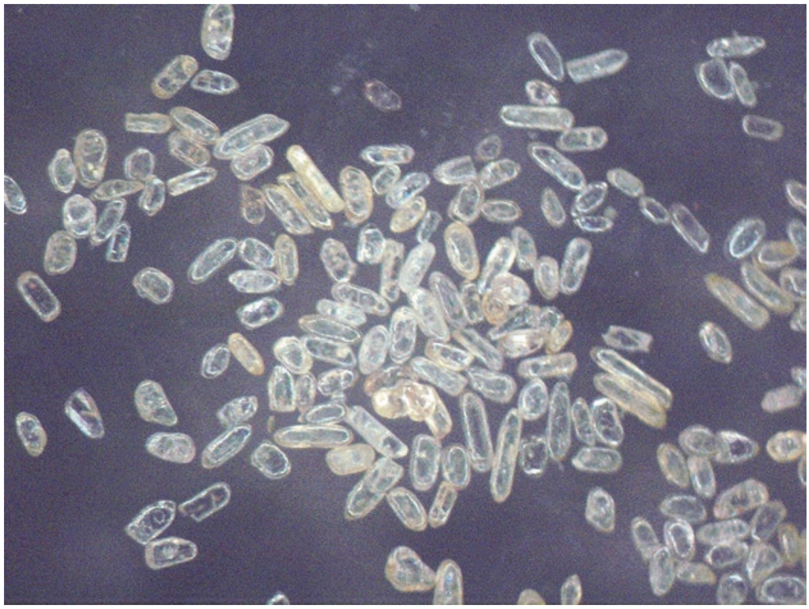


Fig. 3.37 Zircons used as indicator minerals (Image courtesy of Javier Fernández)

traditional geochemical methods for soil, stream sediment, or till sampling in that the indicator grains reflect mechanical dispersion and the individual grains are visually examined and counted. The greatest advantage of indicator mineral methods over traditional geochemical analysis of the heavy mineral or some other fraction is that the mineral grains are visible and can be studied (McClenaghan 2005). The choice of sample media will depend on the climate, topography, and size of area to be sampled. For example, in glaciated terrain, till is most often used for indicator mineral surveys due to its simple transport history. Stream and alluvial sediments are sampled in glaciated, temperate, tropical, and arid terrains. In turn, aeolian sediments can be sampled in arid terrain where other media are not available.

Nowadays indicator minerals are used to detect a great number of mineral deposits such as diamond, gold, Ni–Cu, PGE, porphyry Cu, massive sulfide, and tungsten deposits. The resulting benefits of using indicator minerals are numerous: (1) the ability to detect halos or plumes much larger than the mineralized target including associated alteration; (2) physical evidence of the presence of mineralization or alteration; (3) the ability to provide information about the source that traditional geochemical methods cannot, including nature of the ore, alteration, and proximity to source; (4) sensitivity to detect only a few grains, equivalent of ppb-level indicator mineral abundances (Averill 2001). One of the most important and typical occurrences in the application of indicator mineral techniques was the bloom in diamond exploration activity in the glaciated terrain of Canada, which originates drastic changes in the concepts of sampling and processing methods because indicator minerals improved the knowledge of kimberlite host rock. Since most of Canada has been glaciated, the glaciers advanced, eroded, homogenized, and redistributed the components of the bedrock that they pass over. For this reason, diamonds in glacial drift are the best indicators of a bedrock source of diamond. However, they are very scarce even in the highest-grade diamond-bearing rocks. For example, one carat – 0.2 g – of diamond per ton of mineralization is regarded a very high-grade diamond deposit. As a result, indicator minerals are an indirect but very useful tool to locate bedrock sources of diamond.

Analytical Methods

The analytical methods applied in geochemical exploration depend on the requirements of exploration stages. Techniques can be grouped according to the attribute being measured. Some techniques utilize X-rays in different forms for analytical objectives, while other techniques use the optical effects of samples. Obviously, each method has a minimum detection limit and below the concentration cannot be calculated. Therefore, geochemical analysis has a degree of uncertainty, being uncertainty expressed in terms of precision. The theoretical lower detection limit is an intrinsic function of the technique, although the quality of the calibration and the cleanliness of the equipment used in sample and standard preparation also limit detection. The goal of most analysis is the determination of the trace metal concentrations in a sample, but currently it is still impossible to analyze all elements simultaneously at the needed levels.

The differences between methods are the costs involved, analysis detection limits, velocity of analysis, and the requirement to take material into solution. The method selected will depend upon the element being analyzed and the amount expected. In developed countries, most common analysis is actually performed by inductively coupled plasma optical emission spectrometry (ICP-OES), often in combination with inductively coupled plasma mass spectrometry (ICP-MS) and X-ray fluorescence (XRF) (Box 3.6: X-Ray Fluorescence Analysis). The three methods require numerous constraints such as highly sophisticated laboratories, very pure chemicals, continuous and nonfluctuating power supplies, and readily available service personnel, among others. In less sophisticated situations, relatively high quality analysis can be carried out using atomic absorption spectrophotometry (AAS), which was the most commonly utilized technique in developed countries until 1980. Regarding individual minerals, detailed identification is commonly provided

Box 3.6

X-Ray Fluorescence Analysis

X-ray fluorescence (XRF) analysis (Fig. 3.38) is one of the most common relatively nondestructive methods for qualitative as well as quantitative (more interesting in mineral exploration) determination of elemental composition of materials. This technique is extremely versatile and is suitable for solids, liquids, as well as powders and can be used to measure many elements simultaneously. The relative ease and low cost of sample preparation and the stability and ease of the use of X-ray spectrometers make this one of the most widely used methods for analysis of major and trace elements in rocks and minerals. In the field, portable X-ray fluorescence analyzers are increasingly used for on-site data acquisition. The lightweight portable nature of the this instrument allows it to be used in the field to survey locations of potential mines directly as well as measuring drill cores to determine the depth profiles of the mineral deposit.

X-rays cover the part of the electromagnetic spectrum between ultraviolet and gamma radiations and are produced by a radioactive source, an X-ray tube, and a synchrotron radiation. XRF technique consists in the study of the produced characteristic spectrum because each element has its unique characteristic energy spectrum (fluorescence spectrum) composed by the allowed transitions of the specific atom in the result of X-ray excitation. In general, a quantitative XRF analysis can be conducted using two basic methods: (a) creating a standard curve: this method involves measuring several samples with a known element concentration (standard reference materials) and finding the relationship between the intensity of the measured element's fluorescent X-ray and the concentration; by referring this relationship, element concentration of unknown sample is obtained only with information on its fluorescent X-ray intensity; or (b) considering the type and properties of all elements that compose a sample, the intensity of each fluorescent X-ray can be derived theoretically: with this method, the composition of unknown sample can be extrapolated by the fluorescent X-ray intensity of each element.

XRF is useful for the geochemical analysis of a wide range of metals and refractory compounds, such as SiO_2 and Al_2O_3, and even some nonmetals (chloride and bromide). The quality of XRF data is a function of the selection of appropriate standards. It is considered best practice to use standards that are similar to the samples in question to minimize matrix effects. XRF can measure down to parts per million concentrations and lower, depending on the element and the material. Regarding the detection limit of each element, it depends upon the specific element and the sample matrix, but in general heavier elements have higher detection limit.

Because X-ray spectrometry is essentially a comparative method of analysis, it is vital that all standards and unknowns be presented to the spectrometer in a reproducible and identical manner. Any method of specimen preparation must give specimens which are reproducible and which, for a certain calibration range, have similar physical properties such as surface roughness, particle shape, particle size, homogeneity, and particle distribution. In addition, the specimen preparation method must be rapid and cheap and must not introduce extra significant systematic errors, for example, the introduction of trace elements from contaminants in a diluent. Thus, specimen preparation is essential in the ultimate accuracy of any X-ray determination.

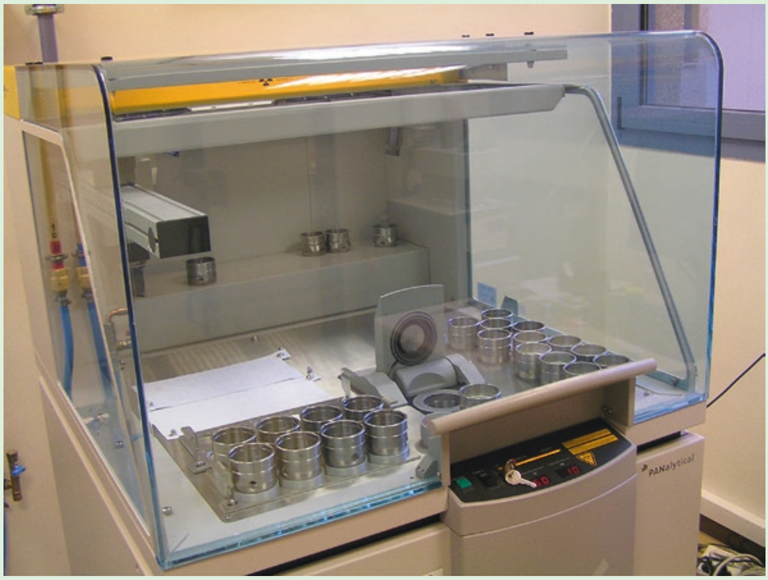


Fig. 3.38 XRF analyzer (Image courtesy of AGQ Labs)

utilizing a scanning electron microscope (SEM) or an electron microprobe.

The advent of sensitive, rapid throughput instrumentation such as ICP-OES and ICP-MS used to complement one another has revolutionized exploration geochemistry in the last decades. ICP-OES and ICP-MS are widely used because of their convenient, virtually simultaneous multi-element capabilities. Plasma used in these techniques permits the simultaneous analysis of up to 40 elements, which means that ICP-MS and ICP-OES (Fig. 3.39) are multi-element techniques. In some cases, detection limits for certain elements can be as low as parts per trillion level in aqueous solutions. AAS uses the absorption of light to estimate the concentration of gas-phase atoms. Concentrations are commonly established using a working curve after calibrating the instrument with standards of known concentration.

In exploration geochemistry, it is very important to note that absolute element content in a sample is not always necessary or, in other words, accuracy cannot be essential. Deviations of ±30% from the absolute value, for example, using international standards, are

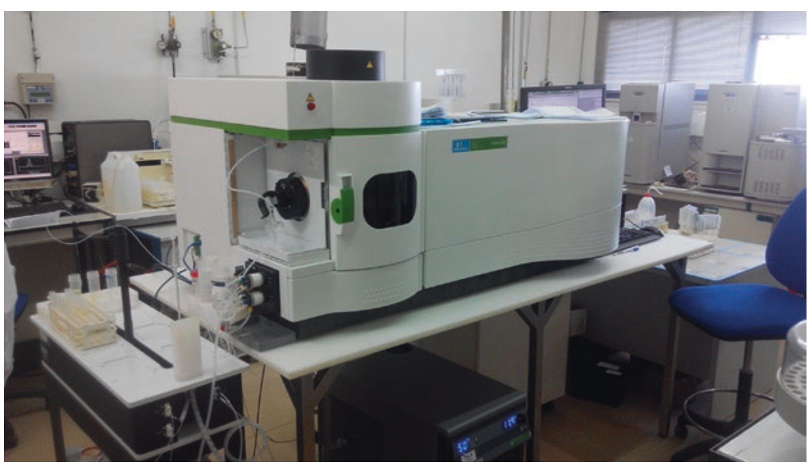


Fig. 3.39 ICP-OES instrument (Image courtesy of AGQ Labs)

endured, if the relative error remains within narrow limits. In contrast, excellent reproducibility of results (high precision) is needed. In fact, this is the most important characteristic of any data evaluation, particularly if the contrast between background and anomalies is small. In all geochemical programs, error control is a fundamental aspect, and for this reason, it is good practice to repeat at least 10% of sampling and/or control the data by another laboratory (Pohl 2011). The process of analysis is generally done at some distance from the exploration project, which means that analytical data is usually accepted and utilized without making criticisms. However, while most laboratories generate good quality results, they are usually looking for a business to make a profit. For this reason, a good quality control minimizes biases, confirms that laboratory assays are correct within a defined degree of accuracy and precision, and detects the presence of contamination between samples.

Interpretation of Data

Introduction

Once the analytical data have been obtained from the laboratory and the results are checked for precision and accuracy, the next question is how to treat and interpret the data. A geochemical exploration data set consists mainly of sample location and values of element concentration in many samples. Since the data are usually multi-element and the number of samples is large, the use of statistical analysis using computer software is essential. This is because the development of low-cost, rapid multi-element analytical techniques has originated large geochemical databases in many exploration programs, including usually thousands of observations with as many as fifty or more elements. Thus, the resulting data matrix is enormous, and effective interpretation utilizing all of the elements becomes cumbersome.

To study these large matrices, the use of multivariate statistical techniques can extract geochemical patterns related to the underlying geology, weathering, alteration, and mineralization. Modern methods of evaluating data, structures, and patterns are clustered under the term «data mining.» This term involves the use of multivariate data analysis and statistical methods in combination with geographic information systems and significantly assists the objective of data interpretation and further model building (Grunsky 2010). It involves the use of automatic and knowledge-based procedures for the recognition of patterns that can be attributed to known processes (e.g., crystal fractionation, hydrothermal alteration, or weathering).

According to Grunsky (2010), issues dealing with geochemical data are numerous: «(a) many elements have a censored distribution, meaning that values at less than the detection limit can only be reported as being less than that limit; (b) the distribution of the data is not normal; (c) the data have missing values: not every specimen has been analyzed for the same number of elements; often, missing values are reported as zero, which is not the same as a specimen having a zero amount of an element and this can create complications in statistical applications; (d) combining groups of data that show distinctive differences between elements where none is expected; this can be the result of different limits of detection, instrumentation or poor quality control procedures; and (e) the constant sum problem for compositional data.» These problems generate difficulties to apply typical statistical procedures to the data. For instance, in the case of varying detection limits, the data need separation into the original groups so that appropriate adjustments can be applied to the groups of data. To avoid the problems of censored distributions, different processes have been designed to estimate replacement values for the objectives of statistical calculations. On the other hand, if missing values are present, several methods can be provided to impute replacement values that have complete analyses.

The normal concentration of an element in non-mineralized Earth materials is referred to as background, which fluctuates around a mean value. It is more realistically viewed as a range of values rather than an absolute value because the distribution of any element in any particular Earth material is rarely uniform and varies considerably from one type of Earth material to another and from one location to another. The upper limit of the range of background values is called the threshold, and uni-element concentrations greater than the threshold are collectively called anomaly. Regarding the concept of threshold, it is possible that in the same exploration project, a lower threshold can be applied in regional exploration, whereas a higher threshold is selected to locate the best targets for further drilling campaigns. Anomalous uni-element concentrations that indicate presence of mineral deposits are called significant anomalies. Thus, the identification of a geochemical anomaly needs an implementation of a geochemical background, which in itself can be difficult to establish. As a rule, geochemical values that deviate too far from the background (values that are atypical) can be considered as anomalous.

In an exploration area, anomalies can be delineated once threshold values in individual uni-element data sets are determined. Analysis of frequency distributions of uni-element concentrations is commonly the easiest way to define the modeling of geochemical thresholds. To do that, there are many classical methods such as comparison of data from the bibliography, data comparison with results of an orientation geochemical survey, graphical discrimination from a histogram of the data, or estimation of thresholds as the sum of the mean and some multiples of the standard deviation of data.

A method of selecting threshold values that is still much used involves calculating the mean (m) and standard deviation (s) of the data set and «applying the classification of anomalous to those values that exceed the value of m + 2 s» (e.g., Hawkes and Webb 1962). This ancient definition was based on the assumption of normality of the data, and its application is no longer legitimated in many cases. In this sense, the introduction of computer-based methods for evaluating geochemical data has provided powerful tools to identify outliers and specimens that can be related to mineralization targets. As a result, the previous commented method of selecting thresholds with the calculation of the mean plus two standard deviations can be erroneous, and a better method is the use of percentiles, specifically 97.5 percentile.

Exploratory data analysis (EDA) is concerned with studying geochemical data to detect patterns or structures in the data. The methods of exploratory data analysis can be grouped in univariate, bivariate, and multivariate methods. Davis (2002) and previous editions of this classical book (the first edition was at 1973) offer an invaluable support to understand the application of these statistical techniques to geological sciences, especially in multivariate techniques. SPSS and Statgraphics are common statistical software packages used in this type of data interpretation.

Univariate Methods

Univariate methods relate to each element separately and with data for which only one variable is considered at a time. These types of methods are crucial in statistically oriented geochemical studies, in particular in the interpretation of results of multivariate methods because achievements derived from multivariate studies can be often predicted by a detailed univariate approach. Many of the exploratory and descriptive methods introduced in this section are recommended as routine ways of investigating properties of new data, even if the final analysis required is bivariate and/or multivariate.

Summary Statistical Tables (Descriptive Statistics)

These types of tables provide useful descriptions of data where quantitative measures are desired. Usually, they show listings of the minimum, maximum, arithmetic mean, median, mode, kurtosis, skewness, etc. Measures of dispersion, a measurement of the spread of data values, include variance, standard deviation, and coefficient of variation (CV). The latter parameter is useful because the mean divided by the standard deviation is expressed as a percentage and represents a relative measure for comparison of different elements.

Skewness means lack of symmetry. A distribution is symmetrical where the frequencies are symmetrically distributed about the mean. The mean, mode, and median coincide in such a type of distribution. Positively skewed distributions occur where the mean is greater than the median and the tail end is more to the right (high values). This is in contrast to negative skewed distributions, where the tail end is toward the left (low values) (Fig. 3.40). Skewness is important as it indicates whether a distribution is described as normal or lognormal. The coefficient of variation is commonly used for this purpose: values of CV less than 0.5 indicate normal distribution whereas values greater than 0.5 indicate skewness and usually represent a lognormal distribution or a combination of distributions.

The analysis of percentiles allows handling of univariate geochemical data. In a data set, the first percentile corresponds to the value of the variable below which 1% of the entries lie. The 50th percentile (median) divides the data set into two

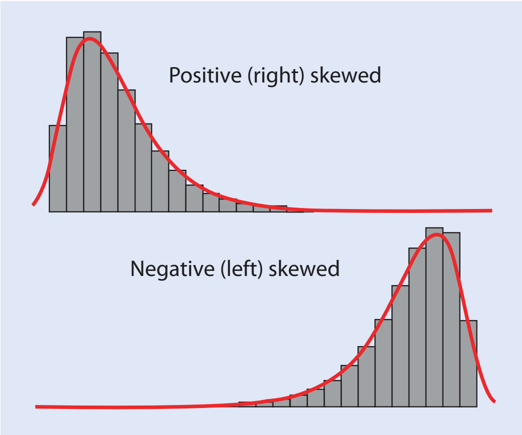


Fig. 3.40 Positive and negative skewness

equal parts. The 25th and 75th percentile are also typically used; they are known as quartiles and are used to calculate the interquartile range (IQR).

Summary tables are useful for the purpose of publishing actual values. However, as described below, graphical methods contribute to visualize the type of distributions and the relationships between observations. The values of a summary table are more easily interpreted where they are combined with graphical summaries.

Box (and Whisker) Plot

The box plot displays order statistics in a graphical form. Unlike the histogram, the shape of the box plot does not depend on a choice of interval. The box plot provides fast visual estimate of the frequency distribution and allows comparison of sets of data. Box plots are made of a rectangular box covering the central 50% of the data set. «Hinges» of the box are the 25th and 75th percentile values, respectively, and the median is marked by a line at the appropriate value (Fig. 3.41). The symmetry and skewness of the data are well reflected; if the data are symmetrically distributed, they are more central and closer to each other. Lines that extend beyond the box are called «whiskers,» whose lengths on each side of the box are indicative of the symmetry of the distribution.

Histograms

Histograms are formed of contiguous upright rectangles (Fig. 3.42). The width of the rectangles indicates the range of values for a particular variable whereas the histogram height expresses the frequency of observations within that range. The scale of the height can be expressed either in number of observations or as a percentage of the total number of observations.

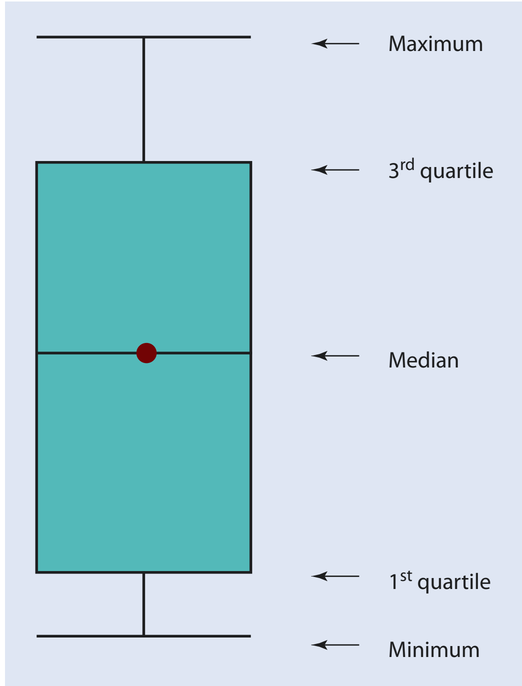


Fig. 3.41 Box and whisker plot

The histogram is a very popular graphical means for showing a distribution. At the initial stage of an investigation, histograms should be plotted for all variables, this helping in geochemical interpretations. Moreover, the histogram will suggest the procedure that can be applied in a further stage. Histograms can be directly drawn in spreadsheets such as Excel. The number of intervals must be clearly defined since too few intervals will avoid the representation of finer details of the distribution while too many intervals will result in a discontinuous distribution. Sturge's rule, which sets the number of intervals equal to $\log_2 n + 1$ (n is the number of observations), can be applied if the distribution is normal or close to normal.

Cumulative Frequency Plots, Probability Plots, and Q-Q Plots

Cumulative frequency diagrams show the percentage values that fall below a value plotted against that value. The shape of a cumulative frequency

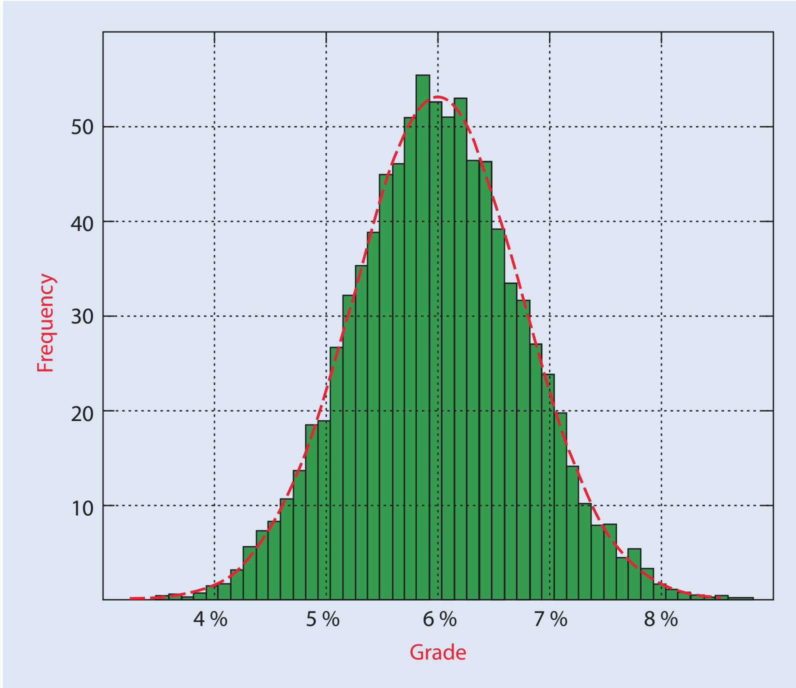


Fig. 3.42 Histogram

curve representative of a normal distribution looks like «S.» A probability plot is a special adaption of that curve when the *Y* axis is scaled in such a way that a normal distribution plots as a straight line. In probability plots any deviations from normality can be quickly identified (Fig. 3.43). These plots have been applied in the splitting of univariate, polymodal geochemical populations into unimodal subpopulations as they help in the identification of anomalies. Cumulative frequency diagrams and probability plots are better than histograms in displaying data.

Equivalent to normal probability plots are quantile-quantile (Q-Q) plots. They also allow graphical comparison of a frequency distribution with respect to an expected frequency distribution (usually the normal distribution). In the Q-Q plots, the quantile values are calculated for the normal frequency distribution, and then they are plotted against the ordered observed data. The plot will be a straight line where the frequency distribution is normally distributed, but it will be curved or discontinuous for skewed frequency distributions or for polymodal populations.

Geostatistical Techniques

Although geostatistics will be described in detail in the next chapter because this technique is mainly devoted to mineral resource/reserve evaluation, the use of geostatistical procedures provides quantification of the spatial variability of an element, for instance, by constructing a semivariogram. Semivariograms measure the average variance between sample points at specific distances (lags). Usually, the variance increases as distance increases between any pair of points. Thus, evaluation of the semivariogram allows assessing the spatial continuity of an element. The effectiveness of applying geostatistical methods relies on adequate sampling density to represent the variation of the data.

The effective use of geostatistical techniques requires knowledge and experience in order to model and extract information from spatial data. They permit better estimates of geochemical trends though geostatistical techniques must be used with the awareness of the problems with techniques of interpolation and the spatial behavior of the data (Grunsky 2010).

Contoured Plans and Profiles

Contour plots of both plans and sections can provide relevant information where variables are gradational in nature, and this gradational character exists between control points. Contours indicate trends, directions of preferred elongation and indications if more than one domain is needed. Since contouring

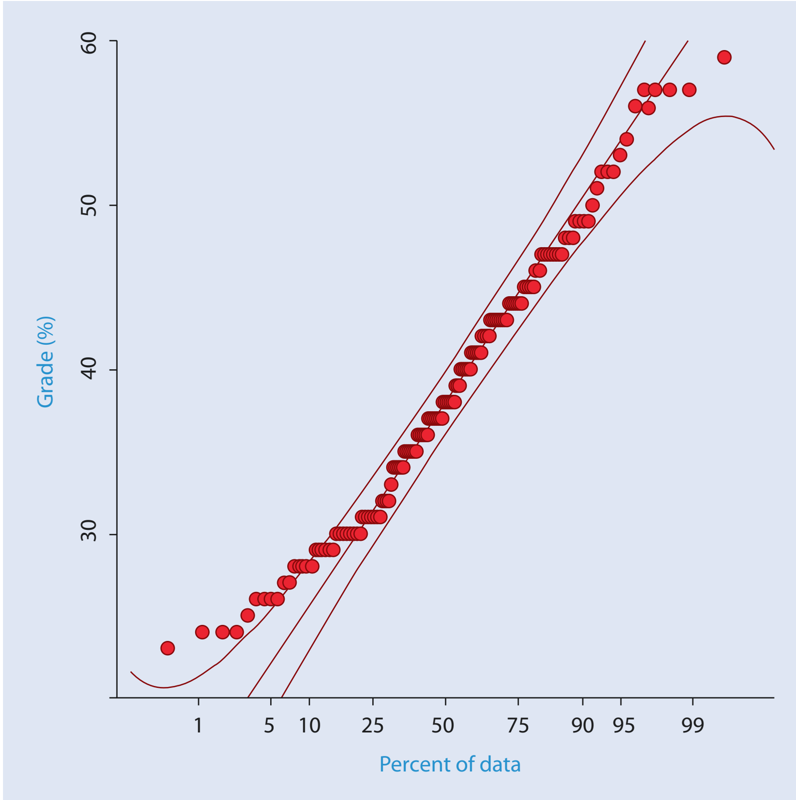


Fig. 3.43 Normal probability plot

is made with computer software, it is important to get a clear understanding of the contouring criteria contained within a given software package (Sinclair and Blackwell 2002). Contouring routines use some kind of interpolation criterion to construct a regular grid of values that can be contoured easily. Interpolation algorithms include inverse distance weighting, nearest point, and triangulation or kriging, among others. This graphical expression of the data is commonly used not only for geochemical data but also in geophysical surveys.

Bivariate Methods

This section considers the analytical methods used if it is necessary to take in account simultaneously the variation of two variables where both are measured on each element in a sample. In addition to providing extra information about the frequency distribution of a sample, these methods generate information on the relationship between variables (Swan and Sandilands 1995). All the techniques of bivariate statistics can be regarded as ways of describing and analyzing the shape of the bivariate scatter.

X-Y Plots

In this method, the values of one variable are plotted against those of another variable determined in the same group of samples (Fig. 3.44). Normal distributions of the data are not assumed in *X-Y* plots, but log transformation can be used in scaling the data. The resulting plots supply better visual estimate of the relationship between two variables and can highlight clusters within the data. This can be improved by display in a scatterplot matrix that allows to represent *X-Y* plots for every variable against every other variable simultaneously.

Correlation Coefficients

Correlation is an exploratory technique used to examine if the values of two variables are significantly related. It means that the values of both variables change or are not together in a consistent way. There is no expectation that values of one variable can be predicted from the other or that there is any causal relationship between them (McKillup and Dyar 2010). Quantitative correlation and calculation of simple linear correlation coefficients

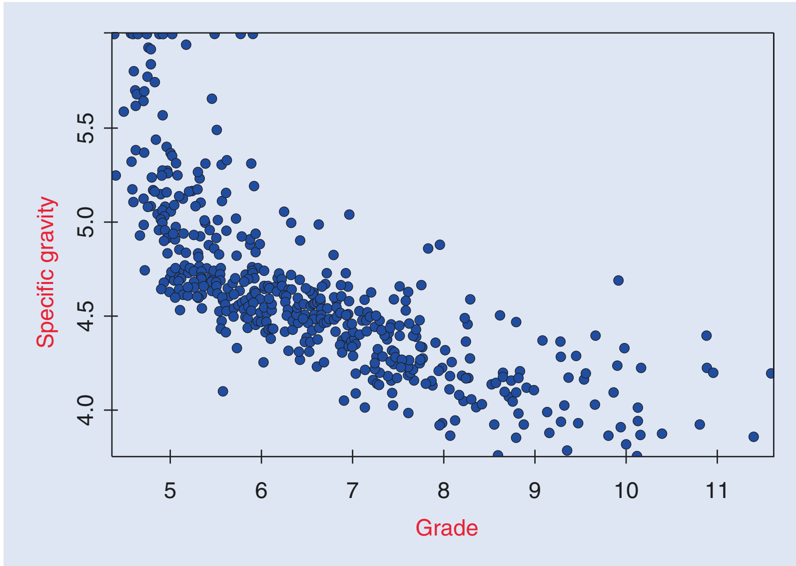


Fig. 3.44 *X-Y* plot

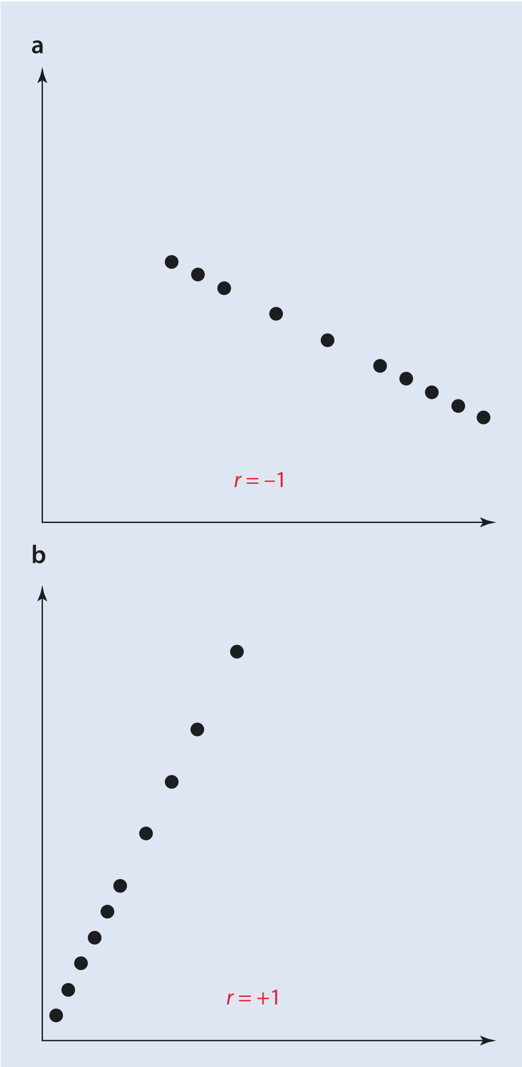
are useful tools for estimating the degree of interdependence between two variables. This can be of great importance provided that it can indicate that the variables are linked, directly or indirectly, in the underlying causative geochemical process.

The most common correlation coefficient used is the Pearson correlation coefficient (abbreviated to r). This coefficient is defined as the covariance of the two variables divided by the product of their standard deviations. As explained in a next section, the concept of R-mode factor analysis is based on the correlation coefficients among a large number of variables. Correlation coefficients are dimensionless. They range between +1.0 (perfect positive linear relationship) and −1.0, the latter value representing a perfect negative relationship (Fig. 3.45). Real data rarely lead to perfect, whether positive or negative, correlation. Like other summary statistics, the correlation coefficient can display abnormal values in the nature of the distribution. These must be always rectified before any important conclusions are drawn from the data or if the correlation coefficient is used as input to other statistical methods like factor analysis. A classic example is a low value of the correlation coefficient in a group of essentially random bivariate data, which increases excessively where a single outlier is introduced in the data set. It is important to remember that linear correlation will only detect linear relationship between variables. Sometimes two variables are clearly related, but their correlation coefficient is near zero, since this correlation is not linear.

Regression Analysis

The correlation coefficient measures the strength of the relationship between two variables. In contrast, regression analysis leads to express the nature of the relationship in quantitative terms. Thus, regression analysis is used to describe the functional relationship between two variables so that the value of one can be predicted from the other. Regression analysis is often preferred to measure the linear relationship between two variables because the nature of the bivariate relationship can be more precisely defined in the form of equation. Regression analysis is essential in geochemistry and geology since the derived equation can be used to describe and aid understanding of the geological process and permits predictions to be made (Swan and Sandilands 1995).

In the case of simple linear regression, a set of bivariate data, expressed graphically as *X-Y* plots, is fitted with a straight line, which can or cannot pass through the origin. This line represents a close relationship between the dependent variable (normally plotted on the *Y* axis) and the independent variable (*X* axis). Total deviation of the predicted values from the observed values is estimated. Moreover, the deviations are squared to remove the plus or minus effects so that the method is known as «least squares.» Sometimes, the values of dependent and independent variables are fitted with a curve, rather than a straight line, and it is called polynomial regression.



■ **Fig. 3.45** Correlation coefficient of: **a** maximum negative = −1; **b** maximum positive = +1

Multivariate Methods

Multivariate statistics relate several elements to each other and facilitate the geochemical interpretation of multi-element data. Multivariate methods are important because virtually all geochemical data are inherently multivariate. Leaving aside some methods such as triangular diagrams, multiple linear regression, or multi-element indices, multivariate data analysis techniques simplify the variation and data relationships in a reduced number of dimensions or groups, which can commonly be tied to specific geochemical/geological processes. Many specific texts, (e.g., Davis 2002), include basics of multivariate data analysis techniques. The multivariate methods most commonly employed in studying and quantifying multi-element associations in exploration geochemical data include principal components analysis (PCA), factor analysis (FA), cluster analysis (CA), and discriminant analysis (DA). PCA and FA are useful in studying inter-element relationships hidden in multiple uni-element data sets, CA is utilized for studying inter-sample relationships, whereas RA and DA are useful for studying inter-element as well as inter-sample associations (Carranza 2009). It is important to note that multivariate analysis requires large samples: in the same way that two observations on a pair of variables are sure to give a correlation coefficient of 1; multivariate data with few observations on many variables will give misleading results.

Triangular Diagrams

Triangular or ternary graphs are used routinely to display relative compositions of samples in terms of three variables. In cases where metal abundance differs by several orders of magnitude, multiplication of one or two of the elements by an appropriate factor is common practice, this resulting in spreading of the plotted points over much of the triangular field. This procedure leads to strong distortion of the ratio scales in the diagrams. In the triangular diagram, each apex represents 100% of one of the elements and the coordinates are numbered for one element on each side in a clockwise direction. It is only necessary to know the percentages of two of the three variables to plot the point.

Multiple Linear Regression

Multiple linear regression is a straightforward extension of simple linear regression. Where there is no single variable sufficiently closely related to the variable being estimated, several variables can be taken together and the estimate of the derived variable will be satisfactory. For example, the sediment yield of a river can be dependent on its drainage area plus other factors such as topographic relief, precipitation, and flow rate (McKillup and Dyar 2010).

Multi-element Indices

Methods exist for dealing with multi-element data that strictly do not involve multivariate statistics. The calculation of multi-element indices is an example of how element associations can be

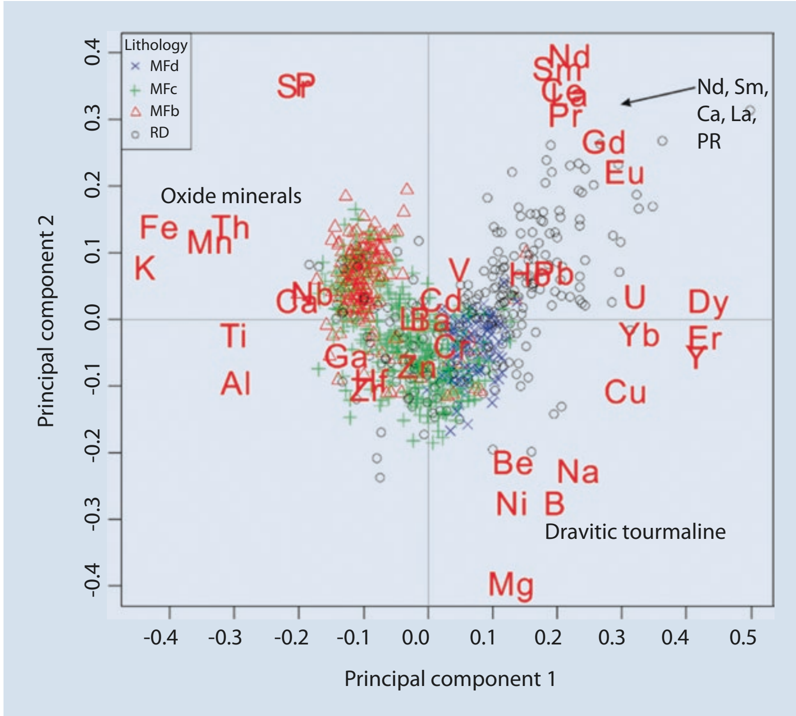


Fig. 3.46 Example of a graphic representation of principal component samples and variable scores (Chen et al. 2015)

applied to optimize features such as types of mineralization or lithologies. Under certain circumstances, some elements are deserving of greater weighting in such an index because of their greater importance as pathfinders for the deposit type sought. If detailed multivariate analysis cannot be achieved because of time limitation, the calculation of multi-element indices provides a way of combining the tendency of certain elements to be enriched in mineralization.

Principal Components Analysis

Principal components analysis, one of the oldest multivariate techniques, is a multivariate procedure to reduce the dimensionality of a data set with a large number of variables while retaining the variation in the variables. This is achieved by forming linear combinations of the variables (principal components) that describe the distribution of the data. In fact, PCA uses the redundancy within the data set to reduce the number of variables, although it does not exclude variables. Instead, PCA identifies variables that are highly correlated with each other and combines these to construct a reduced set of new variables that still describes the differences among samples.

The linear combinations are derived from some measure of association such as correlation or covariance matrix. Principal components are chosen in such a way that the first principal component accounts for most of the variation in the data set and subsequent components for decreasing amount of variation. The interpretation of PCA results points to geological/geochemical interpretation on the element loadings comprising the components. Ideally, each principal component might be interpreted as describing a geological process (e.g., crystal fractionation, mineralization processes, or weathering). Figure 3.46 shows an example of a graphic representation using principal component samples and variables.

Factor Analysis

The term R-mode factor analysis is given to several related techniques that try to identify a limited number of controls on a much greater number of observational variables. It is called R-mode because it is based on r, the correlation coefficient, and deals with relationships between variables. On the opposite, Q-mode factor analysis deals with relationships between samples instead of variables. They are designed as linear

Table 3.4 Factors are linear combinations of variables

Factor	Factor 1	Factor 2	Factor 3	Factor 4	Communality
Ba	*0.92*	0.06	−0.08	−0.09	0.863
Ce	*0.59*	0.01	0.34	*0.62*	0.855
Cr	*0.81*	−0.24	0.26	0.41	0.947
Fe	−0.07	*0.87*	−0.08	−0.00	0.766
K	0.31	0.04	*0.75*	−0.28	0.746
Mn	−0.06	*0.94*	0.04	−0.08	0.902
Ni	*0.79*	−0.38	−0.42	0.20	0.977
Ra	*0.67*	0.49	−0.03	0.17	0.730
Rb	0.21	*0.69*	−0.13	0.46	0.746
Sr	*0.82*	−0.31	0.21	−0.15	0.833
Th	0.05	−0.02	*0.79*	0.07	0.629
Ti	−0.39	*0.75*	0.49	0.09	0.962
U	0.10	−0.41	−0.13	*0.62*	0.579
Zn	−0.18	−0.08	0.00	*0.83*	0.734
Zr	−0.49	0.40	*0.63*	0.29	0.886
Eigenvalue	5.38	2.80	2.30	1.68	
% Var. expl.	36	18	15	11	
Cum. % var.	−36	54	69	80	

combinations, or «factors,» of those variables (Table 3.4). Where geochemistry is considered, such factors will be related to the processes acting on the environment, and furthermore they can correspond to geochemical relationships. R-mode factor analysis allows to condensate a large number of geochemical variables into a smaller number of linear combinations of those variables that account for most of the total data variance. The number of factors is likely to be much smaller than the number of variables. The factors can be plotted and interpreted more easily than the full data set because more geochemical information can be summarized at each sampling point (Gocht et al. 1988). Since the method is based on the correlation coefficients between the variables, FA is quite sensitive to their variations. Therefore, it is crucial to calculate the correlation coefficient so that distortion by outliers can be avoided.

Cluster (Dendrogram) Analysis

Cluster analysis leads to grouping of points that represent individual geochemical samples in multi-element space. The procedure is performed without prior knowledge of the groupings or their compositional characteristics. Cluster analysis methods offer an excellent exploratory tool for analyzing groups of multi-element data, not clearly observable in simple scatter plots or by means of PCA. Thus, the main objective of clustering algorithm is to distinguish natural groupings within multi-dimensional data: it links the most similar pairs of observations or clusters in successive stages until all points are grouped.

Discriminant Analysis

Discriminant analysis is utilized to deal with problems of classification (Fig. 3.47). It is one of the most widely used multivariate procedures in Earth sciences. Discriminant analysis uses

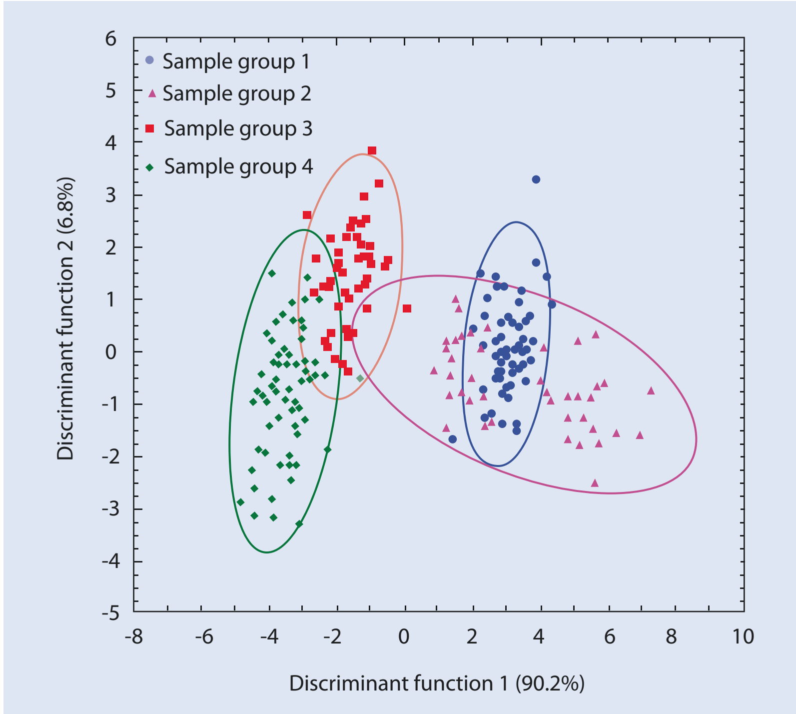


Fig. 3.47 Graphical representation of groups using discriminant functions 1 and 2

all of the analyses in a data set, being the objective to maximize the distinction between two or more previously defined groups. It enables the further allocation of samples of unknown origin based on analyses of the same elements. The objective is to find discriminant functions: these are vectors in the directions of optimal separation between the groups, and they transform the original set of measurements on a sample into a single discriminant score. The discriminant function provides not only the possibility of assigning samples of unknown association to one of these two groups but also of measuring the degree to which each of the variables contributes to the classification.

3.4.6 Drilling

Introduction

Where an anomaly is found, by using geophysical and/or geochemical prospection, the mining company will initiate a drilling program in order to test the extent of the mineralization. The density of drilling will be set up by the wanted level of geological confidence and project economics. The drill program searches to confirm the presence of the mineralization and must determine its shape and continuity by studying the samples collected from every drill target of the drill program. Mining requires drilling mainly for two different goals: (1) production drilling, making holes to place explosives for blasting (the holes drilled for this purpose are defined as blasthole and this topic will be covered in the exploitation chapter), and (2) exploration drilling, to estimate the amount and grade of a mineralization using the sample collections (Fig. 3.48). Likewise, drilling is a continuous process throughout the entire life of the mine to supplement reserve for the mined ore. This will increase the mine life and continue mining operation. Moreover, it also upgrades the categories of the reserves by using underground drilling. A strategically placed underground drilling program can even probe for new ore bodies in the neighborhood.

Drilling is the most frequently used technology in mineral exploration, and it is usually the most expensive because its expenditure can reach up to half of the costs of total exploration. In most cases, drilling locates and defines economic mineralization. The first objective of drilling is to safely obtain representative samples of the target mineralization in a cost-effective manner.

Fig. 3.48 Preparing samples after drilling (Image courtesy of Anglo American plc.)

The rock types are defined using the study of the samples, and portions of them are commonly chemically analyzed with the aim of further characterization of rock types and to search the existence of valuable minerals. Thus, the different methods of drilling are for diverse objectives at various phases of an exploration program. Studying drill core also allows for geotechnical/rock mechanics data, being logs gathered during surface drilling.

There are a large number of drilling techniques. This heading is centered on the three main types used in mineral exploration: reverse circulation (RC) drilling, rotary drilling using tricone roller bits, and diamond core (DC) drilling. Each drilling method has its own characteristics, which affect the quality of the collected samples. DC drilling generates a cylinder-shaped sample of the ground at an accurate depth. On the opposite, RC drilling and rotary drilling using tricone roller bits yield a crushed sample that includes cuttings from a precise depth in the drillhole.

Rock Drillability

Rock drillability is defined as the penetration rate of a drill bit into the rock. It is a feature that cannot be exactly defined by a single mechanical property of the rock. For this reason, drillability is a function of numerous rock properties such as mineral composition, grain size, texture, and weathering degree. Quartz is one of the commonest minerals in rocks. Since quartz is a very hard material, high quartz content in rock can make it very hard to drill and will certainly cause heavy wear, particularly on drill bits. On the other hand, a coarse-grained structure is easier to drill and causes less wear of the drill string than a fine-grained structure.

Drillability is not only decisive for the wear of tools and equipment but is, along with the drilling velocity, a standard factor for the progress of drilling works. Hoseinie et al. (2008) suggest that the most important rock mass parameters that affect the drilling are the following: the origin of the rock's formation, the Mohs hardness, the texture of the rock (shape and size of grains), porosity, density, abrasiveness, rigidity, P-wave velocity, elasticity and plasticity, UCS (point load index and Schmidt hammer), tensile strength, structural parameters of the rock mass (joints, cracks, and bedding), and RQD.

The factors that concern the drillability of rocks are numerous and can be classified into two main groups: controllable and uncontrollable parameters. Regarding the controllable parameters, these are bit type and diameter, rotational speed, thrust, blow frequency, and flushing. Rock properties and geological conditions are uncontrollable parameters (Yarali and Kahraman 2011). The drillability of rocks depends on not only their physical properties but also on the type of drill being used and drilling parameters such as rotation speed, feed rate, etc. The physical properties of rocks which have some effect on drillability are:

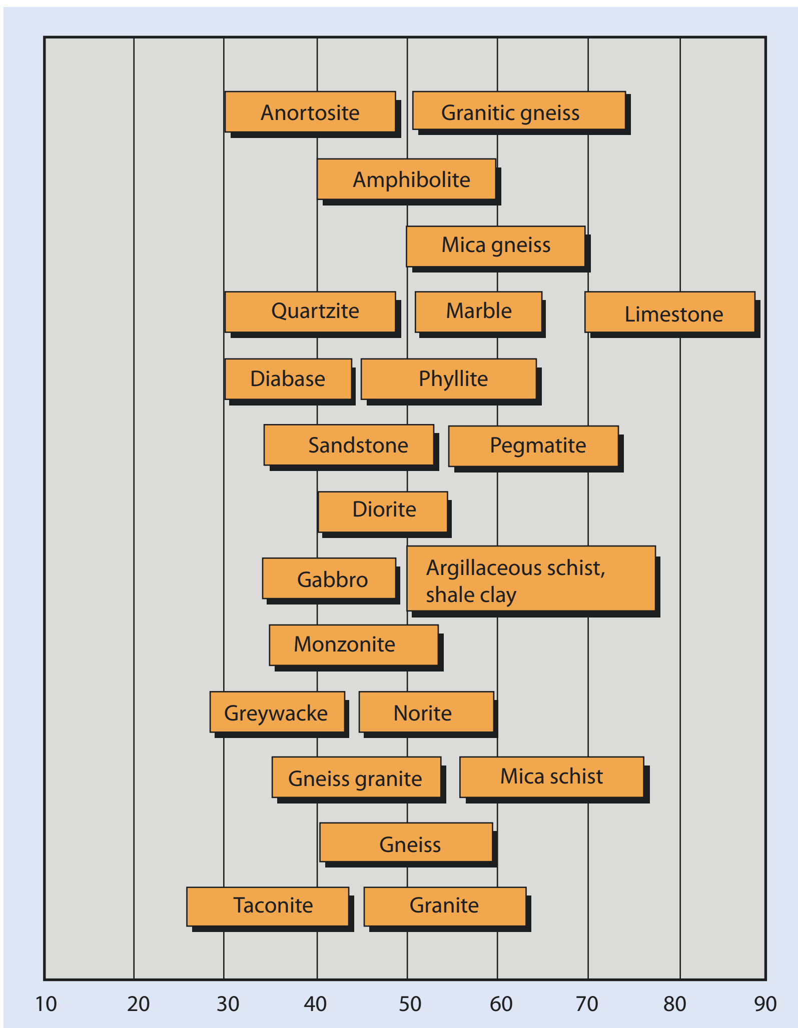
1. Crushing strength, defined as the pressure a rock sustains before breaking and related to grain hardness and strength, grain bond strength, porosity, and weakness planes.
2. Toughness, a measure of how difficult it is to pull a rock apart and related to grain shape and bond, fissibility, and tenacity.
3. Chip separation, this is how readily the cuttings are cleared from the face, and it is related to pore pressure and permeability.

4. Abrasiveness, the ability to wear downhole tools and related to grain hardness and shape (Hartley 1994).

The Norwegian Technical University has defined two methods to evaluate the rock drillability: the drilling rate index (DRI) and the bit wear index (BWI). The DRI describes how fast a particular drill steel can penetrate. It includes measurements of brittleness and drilling with a small, standard rotating bit into a sample of the rock. The higher the DRI, the higher the penetration rate, and this can vary greatly from one rock type to another (◘ Fig. 3.49). It should be noted that modern drill bits greatly improve the penetration rates in the same rock types. The BWI gives an indication of how fast the bit wears down, as determined by an abrasion test. The higher the BWI, the faster will be the wear. Thus, in most cases the DRI and BWI are inversely proportional to one another. However, the presence of hard minerals can produce heavy wear on the bit despite relatively good drillability. This is particularly the case of quartz, which has been shown to increase wear rates greatly. Certain sulfides in ore bodies are comparatively hard, impairing drillability (Samuelsson 2007). Other means of commonly used rock classification include the Q-system; rock mass rating (RMR) of Bieniawski, incorporating the earlier rock quality designation (RQD); and the geological strength index (GSI).

Selection of Drilling Method

Selecting the right technique or combination of techniques depends on many factors: speed, cost, actual conditions (surface or underground),



◘ **Fig. 3.49** Relationship between drilling rate index and various rock types (Samuelsson 2007)

Fig. 3.50 Rock chips and core samples (Image courtesy of Atlas Copco)

depths of the drillholes, type of rocks, required sample volume and quality, logistics, environmental considerations, and finally the preference of the geologist. Moreover, each of these factors depends in turn on many parameters. For example, drilling velocity is dependent on a lot of geological parameters such as jointing of rock mass, rock anisotropy (e.g., orientation of schistosity), degree of interlocking of microstructures, porosity and quality of cementation in clastic rock, degree of hydrothermal decomposition, and weathering of a rock mass, among others (Thuro 1997).

Modern core drilling rigs carry out fast and efficient core sampling of different diameters to very large length. There are many items to select the appropriate method of drilling: target, host rock, water presence, sample required, access, and politics (Hartley 1994). From a sampling viewpoint, there are two types of drilling methods in mineral exploration: drilling methods that originate rock chips and those that generate core samples (Fig. 3.50). A three-key-factor selection process can be established: the time needed, the cost of getting the job done, and confidence in the quality of the samples brought to the surface (Gustaffson 2010).

Time Factor

For any exploration drilling, the sample is the most important goal result. RC drilling generates continuous drilling with high penetration rate and can offer three times the productivity of core drilling. Thus, significant timesaving can be obtained using RC. When the ore body is located, driller can decide to continue with RC drilling or switch to diamond core drilling to extract cores. In so doing, RC drilling and classical core drilling are perfectly combinable. The logistics of the drilling program have clear influence on the number of meters drilled per shift and thereof it is a time factor.

Cost Factor

Costs are mainly related to the time factor, except that investment in RC rigs and equipment is higher compared to core drilling. For shallow exploration applications, time and costs are in favor of RC drilling. For deeper exploration applications, shallow subsoil water and rocky terrain, core drilling is still the only practical alternative. Technical developments in drilling tools and rig technology have resulted in lower drilling costs.

Confidence Factor

The third variable in the equation is the confidence factor. In an evaluation with positive results, a program of core drilling is the common way to drill for the purpose of bringing the project to a resource/reserve status because geologists need dry and representative samples to carry out optimum evaluations. Therefore, core drilling remains the only viable method in these situations. The core helps the geologist to calculate the cost of extracting the

Fig. 3.51 Reverse circulation drilling machine (Image courtesy of Atlas Copco)

mineral from the ore. Moreover, cores also yield geotechnical data; for instance, data about slope stability can be of the highest significance. Finally, the geologist plays an extremely important role in finding an intelligent and balanced choice between the two methods.

Reverse Circulation Drilling

RC drilling technique (Fig. 3.51) starts its utilization in searching mineral deposits since the early 1970s in Australia. It can be used in unconsolidated sediments or for drilling rock. Since this method is clearly less expensive than core drilling, it is the selected method for most preliminary mineral exploration work. The advantages of using this method to collect rock chippings are that all the sample is collected, the method is very fast, up to 200–300 m per day is common at drilling rates exceeding 10 m per hour, and there is very little contamination. RC sample content ranges from dust to 25 mm chips. Often, reverse circulation drillholes are of larger diameter than common diamond drillholes, but it can be hard to acquire sound geological descriptions because the material is obtained/recovered in the form of broken rock chips.

The RC method uses dual wall drill rods that include an outer drill rod with an inner tube situated inside the drill rod (Fig. 3.52). The inner tube affords a continuous pathway with sealed characteristic for the drill cuttings to be translated to the surface. High-pressure air is the common way to define the drill flushing medium. Water can also be injected to reduce dust and to assist in transporting cuttings to the surface. At the surface, the cuttings are derivated to a cyclone for collection and bagging.

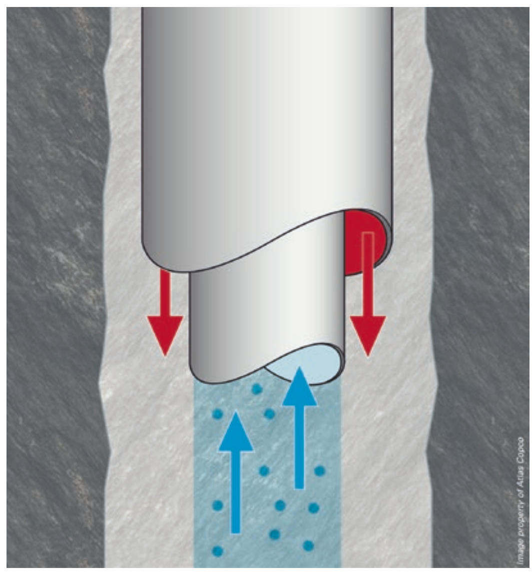


Fig. 3.52 RC system (Illustration courtesy of Atlas Copco)

There are many different drill bit types, each of them designed for different drilling conditions (e.g., rock type). Drill bits are chosen given the underground rock formations expected to be encountered since to change bits can be a long process. The most classical method utilized is the reverse circulation hammer because it drills almost all geological formations. RC hammers are designed with an inner sample tube that extends through the center and into the top of the hammer bit.

RC drill rigs typically reach depths up to 500 m, although it can exceed that depth. The method is undergoing continuous technical development that will result in RC drilling being applied to deeper drillholes and more difficult geological conditions. In a comparison between RC drilling and core drilling, RC drilling presents two main issues. First, most of the RC drill rigs actually used have a depth constraint of about 500 m. Second, RC drilling offers obviously less information regarding the geological structure of the ore body. It is important to bear in mind that this aspect is very important when estimating the cost of extracting mineral from ore. Regarding the sampling process, RC drilling is mostly led to obtain mineral samples for analysis, so correct sampling equipment and practices are necessary when undertaking this type of drilling.

There are two main components to the sampling system: the cyclone (◘ Fig. 3.51) and the splitter. The cyclone serves mainly to separate the sample from the air, thus allowing it to be recuperated. A good cyclone will usually gather more than 99% of the sample, being the sample interval normally 1 or 2 m of drillhole. As one sample has been collected, another is being drilled and incorporating to the cyclone. The other mentioned component is the splitter. The purpose of this instrument is to cut the sample to a smaller size, which accurately represents the complete sample. The sample from 1 m drillhole is about 50 kg. This sample is currently in a bag that is sent to a laboratory for subsequent analysis.

Rotary Drilling

Rotary drilling using tricone bits is a noncoring method, being usually utilized for drilling through soft to medium hard rocks such as limestone, chalk, or mudstone. Rotary drilling uses different type of rotary bits although the most typical rotary bit is probably the tricone or roller rock bit (◘ Fig. 3.53) that is made with tungsten carbide insets. As the drill string is rotated, the bit cones roll along the bottom of the borehole and the rock chips are flushed to the surface by the drilling fluid for examination; in this method, advances of up to 100 m per hour are possible. It requires minimal air volume, and downhole costs are low. For this reason, it is a very economical method of drilling. Tricone bits are used in many drilling industry sectors. It is commonly applied to oil industry, with large diameter holes (>20 cm) and several 1000 ms depth.

◘ **Fig. 3.53** Tricone bit (Image courtesy of Atlas Copco)

Diamond Core Drilling

In diamond core drilling, a cylinder of solid rock, the core, is extracted from depth. It is commonly 27–85 mm in diameter (◘ Fig. 3.54), but larger diameters (up to 200 mm) are most useful but much more costly. The most common sizes used today for exploration drilling are 75 mm hole diameter. Due to the common hardness of the rocks and the time involved in translating the core from depth, the penetration in diamond core drilling is much slower than other drilling methods. Thus, diamond drilling is clearly more expensive than reverse circulation drilling.

◘ **Fig. 3.54** Different core diameter sizes

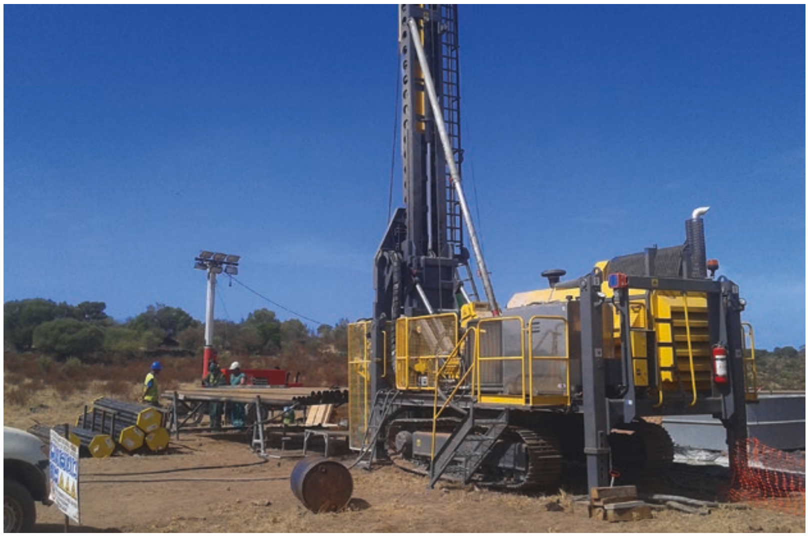


Fig. 3.55 Diamond core drilling operation (Image courtesy of Atlas Copco)

However, if core recovery is good, it has the benefit of carrying undamaged rock to the surface. Therefore, diamond drilling is usually accounted to offer the best quality of sample. Most advanced exploration uses a combination of diamond and reverse circulation drilling. In general, diamond drills are the most essential tool in the final exploration and evaluation of mineral projects because the study of the drill core yields a three-dimensional geologic picture of ore and host rock and the samples from drill core provide samples for chemical analysis, mineral recovery tests, and rock stability tests.

Nowadays, typical drilling operation includes a truck-mounted rig and a support truck to carry items such as the rods, casing, fuel, and water (Fig. 3.55). The method requires significant site preparation and rehabilitation. Diamond drilling machines utilized in mineral exploration commonly reach depth of up to 3000 m and extraordinarily up to 6000 m. In these situations, casing is installed in the upper levels to protect the walls from collapse. The rate of advance will depend of many factors (type of drill rig, type of bit, hole diameter, the depth of drillhole, and the rock type being drilled, among others). Drilling advance rates of up to 10 m an hour are common. The costs can range from USD 40 to USD 90 a meter in drillholes up to 300 m long and from USD 75 to USD 160 a meter for length up to 1000 m.

The quality and continuity of the core are crucial in the assessment of a potential mine, making the core bit a key component of a core drilling rig. The diamond drill bit comprises a cutting head using diamonds as the cutting medium. A variety of core bit types is available according to the diamond cutting elements used in their construction. In softer rocks (e.g., sedimentary formations), other cutting elements such as tungsten carbide and polycrystalline diamond compacts can be used. Diamonds used are fine to microfine industrial grade diamonds that are set within a matrix of varying hardness, from brass to high-grade steel. Other options include tungsten carbide (TC) and polycrystalline diamond composite (PDC) bits. TC core bits are utilized for drilling in non-consolidated formations and in overburden and for cleaning drillholes. PDC bits are an alternative to TC bits and surface set diamond bits when drilling in non-consolidated and medium hard rock formations (Black 2010).

As the drill bit advances, a cylindrical core of rock progressively fills a tube core barrel immediately above the drill bit. Core barrels are classified by the length of core they contain. They are usually from 1.5 to 3.0 m in length but can be as long as 6 m. It is important to note that to recover the core the barrel must be removed from the hole by pulling the entire length of drill rods to the surface, which is a time-consuming process. For this reason, the wireline system is now a standard practice (Box 3.7: Wireline System). Water is used in diamond core drilling as lubricant fluid and to remove crushed and ground rock fragments from the bit surface. Water can be used in combination with various clays or chemicals (Fig. 3.56).

Fig. 3.56 Chemical products used with water in diamond core drilling (Image courtesy of AMC)

Box 3.7

Wireline System

The Boart Longyear company introduced the wireline core retrieval technology to the mineral exploration industry in 1958. By the late 1960s, it was in almost universal use. Wireline core drilling is a special type of core drilling, most commonly used in mineral exploration. Before wireline drilling, the whole string of rods had to be pulled from the ground in order to recover core from each advance of the drill. Thus, in conventional rock coring, the entire drill stem and core barrel must be removed after each core run. This is a time-consuming operation on deep core holes, in addition to creating an inherent risk for collapse of the rock into the unsupported borehole. Moreover, as the average depth of hole continues to increase, the time and money saved by not having to remove the drill pipe in order to obtain a core is substantial. Consequently, wireline system is designed to recover rock core without removing the drill stem from the borehole after each core run (Fig. 3.57). Besides reduced tripping time and decreased cost, wireline core drilling system has the main following advantages: (1) with improved core recovery and quality, the purpose of the drilling project can be better satisfied; (2) logging instruments can be lowered by utilizing internal flush drill rod; (3) inner tube structure can be changed in accordance with the variation of rock layer; and (4) labor intensity of the operators can be reduced.

To obtain a core after the core bit is in place, the core barrel assembly is forced down the inside of the drill pipe using drilling mud pressure. When the core barrel assembly reaches the lower end of the drill stem, a locking device holds the barrel in place. The core barrel assembly consists of a cutter head, core catcher, core barrel, vent or inside pressure relief, locking device, and a retrieving head. During coring operations, the circulating fluid passes between the core barrel assembly and the drill collar. After the core has been cut, the core barrel assembly with its core is retrieved by lowering an overshot through the drill pipe (McPhee et al. 2015), or overshot, which is designed to engage the upper end of the core barrel. As the overshot is lowered over the upper end of the assembly, the locking devices are released,

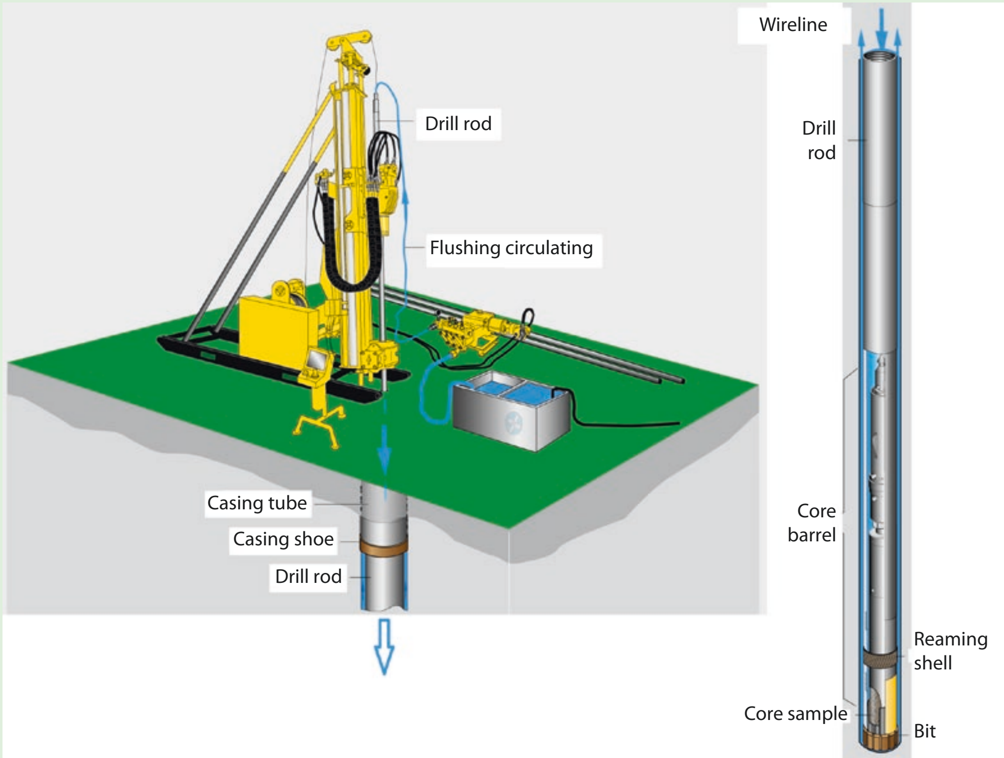


Fig. 3.57 Wireline system

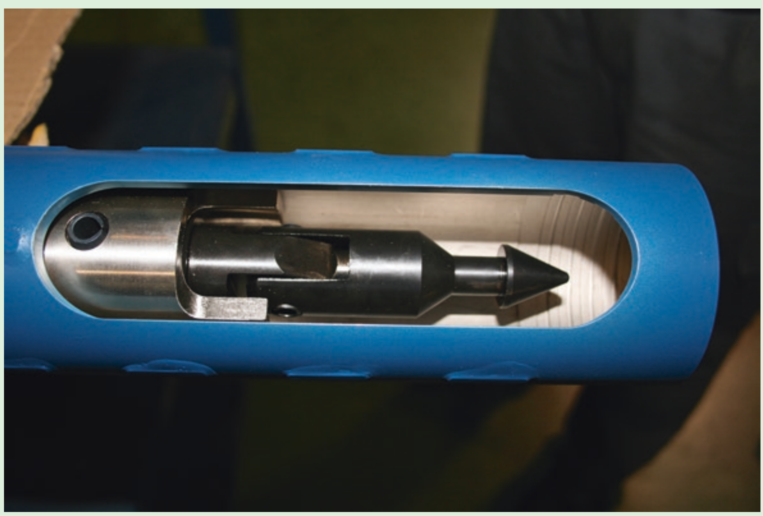


Fig. 3.58 Head assembly inside the overshot (Image courtesy of TECSO)

permitting removal of the entire assembly. Thus, overshots are a key component of wireline coring systems. In this method, the inner barrel containing the rock core is rapidly brought to the surface, leaving the outer core barrel and drill rods sill in position within the borehole. While the core sections are being removed from the inner tube and placed in special core boxes, a replacement inner tube is lowered into the hole so that drilling can recommence. The commonly used standard core diameters for wireline drilling are AQ = 27 mm; BQ = 36.5 mm; NQ = 47.6 mm; HQ = 63.5 mm; and PQ = 85 mm. Figure 3.58 shows the head assembly and the overshot attached.

Core recovery is essential in diamond drilling. This is a quantifiable measurement defined as the total linear amount of physical core sample extracted over the total linear advance in a borehole, expressed as a percentage. Low core recovery impedes quantitative interpretation of important properties, such as ore grade and ore boundaries. For example, Henley and Doyle (2005) reported an important bias in ore grade at Las Cruces (Spain) as a result of core loss. The problem was related to the presence of chalcocite in the mineralization as a friable and unconsolidated form. Very often, core recovery of more than 90% is stipulated with drilling contractors. Other strict rules must be agreed, such as careful extraction of the core and its packing in properly labelled core boxes and marking individual core runs. In this sense, drilling should be supervised by experienced geologists. Proper storage of core is needed for the duration of the project if the prospect is rejected and for the whole life of the resulting mine, if the deposit is feasible (◘ Fig. 3.59). Although onerous, storage is much cheaper than repeat drilling (Pohl 2011).

It is important to note that diamond core drilling is also carried out in underground mining development (◘ Fig. 3.60). Thus, underground core drilling is mainly accomplished to characterize new ore reserves and for the safety of the mines in establishing the position of possible gas or water intersections.

Other Drilling Methods

Other drilling methods used in mineral exploration include auger drilling and sonic core drilling, the latter being the most recent improvement in drilling technology (◘ Box 3.8: Sonic Drilling). Regarding auger drilling, rock is cut and broken with a simple blade bit that is mounted on the end of a rotating string of rods (◘ Fig. 3.61). The drill stem is shaped like a helical screw and is driven rotationally into the ground. Auger drilling is a useful method for quickly and cheaply collecting geochemical samples. On the other hand, this method is usually utilized to take samples in the reconnaissance stage of mineral exploration. Regarding the rate of penetration, it depends on the type of formation being drilled but commonly can reach depths of around 20 m. Obviously, augers are not capable of penetrating hard or consolidated rock. Auger drilling uses either a handheld power auger or one mounted on a small vehicle. Augers are available in various sizes. Thus, small augers mounted on trucks are often used for reconnaissance exploration projects while large augers are utilized for construction purposes.

◘ **Fig. 3.59** Proper storage of core (Image courtesy of Matsa, a Mubadala & Trafigura Company)

Fig. 3.60 Diamond drill underground exploration station (Canada) (Image courtesy of North American Palladium Ltd.)

Fig. 3.61 Auger drilling at Burkina Faso (Image courtesy of SEMAFO Inc.)

Borehole Surveying

In a drillhole, the orientation is fairly established by its azimuth (direction) and dip (inclination). It is common that borehole deviates away from the original direction because of many factors (Fig. 3.64). Borehole deviation is commonly defined as the angular change from vertical during the course of drilling. Some authors also call this process as borehole deflection. However, it is possible to distinguish clearly between deviation and deflection (Hartley 1994). Deviation indicates how the borehole changes path naturally whereas deflection points out where the driller deliberately changes this natural deviation by inserting some mechanical device or changing the rod string. The reasons to change artificially the borehole path can be various: (a) to create daughter boreholes to enable several intersections from the same collar, (b) to enhance or depress natural deviation to ensure the target is intersected, (c) to bypass difficult drilling conditions, (d) to obtain second intersection for improved recovery, and (e) to force the borehole path to those otherwise inaccessible locations.

Box 3.8

Sonic Drilling

Sonic drilling is a unique technology that generates vibrational frequencies, usually between 50 and 180 Hz (cycles per second), transferring the vibrations down the drill pipe to its tungsten carbide bit while rotating the pipe at the same time. This frequency range falls within the lower range of sound vibrations that the human ear is capable of hearing. Thus, the term «sonic drill» has been applied to this class of rotary-vibratory drilling machine. Sonic drilling technology was first applied over 40 years ago in Canada. In mineral exploration, sonic drilling (◘ Fig. 3.62) is typically used to provide continuous core samples of softer or even harder rock formation of mineral deposits. Instead of using a diamond bit rotating at the end of a drill rod, the sonic drill head sends high-frequency vibrations throughout the length of the entire drill pipe and onto the bit (◘ Fig. 3.63).

In sonic drilling, the head contains the mechanism necessary for rotary motion, as well as an oscillator, which causes a high-frequency force to be superimposed on the drill string. The drill bit is physically vibrating up and down in addition to being pushed down and rotated. These three combined forces allow drilling to proceed quickly through most geological formations including most types of rock. The operator is able to vary the frequency and drill bit weight to match the material he/she is going through, ensuring the best penetration rate and most accurate sampling are obtained.

The sonic drilling method can produce almost completely undisturbed core samples from both solid and unconsolidated materials with high percentage of core recovery rates; it is commonly greater than 90%, which gives rise to extremely accurate estimates of mineral distribution in the ore body. Sometimes, core sampling can be accomplished without any drill fluids (dry coring), although the casing is usually installed by using water or mud to flush cuttings. Sonic drilling can collect samples up to 300 mm in diameter and can drill down to 250 m in a vertical or angled hole. The environmental impact from sonic drilling is typically less than other drilling methods. Thus, having a small footprint and lack of need to introduce fluid into the hole, this is an ideal drilling method where contamination is potentially a problem.

In soft materials, sonic drilling is a penetration technique that strongly reduces friction on the drill string and drill bit due to liquefaction, inertia effects, and a temporary reduction of porosity of the material. The entire drill string is brought to a vibration frequency of up to 200 Hz, which causes a very thin layer of soil particles directly surrounding the drill string and bit to loose structure. Instead of the stiff mass that requires torque and weight to penetrate, the soil behaves like a fluid powder (in an unsaturated zone) or as a slurry or paste in a saturated zone.

The liquefaction and inertia effects enable to collect very long and continuous samples. In addition, the drill string stays extremely straight due to the vertical high-frequency movement, with a diversion

◘ **Fig. 3.62** Sonic drilling in iron ore mining (Image courtesy of Sonic Drilling Ltd.)

◘ **Fig. 3.63** Sonic drill bit (Image courtesy of Sonic Drilling Ltd.)

of commonly a few centimeters over the full length of the borehole. It makes sonic drilling an optimal technology for installing instrumentation and monitoring equipment. In alluvial material, vertical vibrations are generally enough to drive down a drill string for many meters without the injection of any water or air. On the contrary, liquefaction cannot take place in hard formations. In such cases, it is necessary to combine vibration with rotation to allow the tungsten carbide buttoned ring bits to cut through the harder formations. Because a sonic drill bit actually impacts the rock face, if a diamond drill bit was used with the sonic drilling method, it would shatter, so tungsten carbide bits are used instead. In order to keep the temperature of the drill bit down and lift the cuttings, foam injection is the best solution, but water or air is possible.

Fig. 3.64 Borehole deviation

If the orientation of the borehole is not known, the location of the sample is similarly unknown. Large discrepancies between planned and true drillhole locations can occur. Since the location of a borehole is just as important as the information itself, the error in location of drillhole due to borehole deviation can impinge significantly on resource/reserve estimation. These errors are unfortunately common. Deviation is commonly cumulative, and the bottom of a deep hole can be many tens of meters away from its straight-line course. For instance, a 200 m drillhole whose plunge is off by only 5° will have the end of borehole moved horizontally by about 17.5 m from its intended position. Consequently, the use of the wrong value of the end of the borehole sample in estimation procedures can originate serious and intense errors of resource and/or reserve estimation.

Although it is difficult to assign an order of importance, there are a number of features that cause a drillhole to deviate both in azimuth and inclination. The more important are hardness of rocks, rock strength anisotropy, anisotropic strength index, active length of drill rods, barrel length, hole size, bit type, and direction of rotation and wedges (Hartley 1994). Regarding the rock strength anisotropy, which is exhibited by rocks with planar texture features such as foliation and bedding, drillholes will tend to deviate so as to make a greater angle with the dominant foliation (usually bedding or cleavage) of the rock unless the drillhole is already at a very low angle to that foliation, in which case the drillhole will tend to deflect along the foliation. However, the absolute magnitude of deviation is related not only to rock strength but also to the relative strength in different directions. Thus, drillholes in well-foliated schists deviate at a much greater rate than through a normal shale, which will be greater than granite. Regarding drillhole size, greater deviation occurs in smaller holes, probably a function of greater flexibility of rod string.

Surveying the path of a borehole is referred to as a borehole orientation survey or a deviation survey. Borehole surveying must be an integral component of all drill programs. Downhole orientation surveys are commonly carried out by moving a probe along the drillhole and checking the movement of the probe relative to a reference. The references can include the Earth's gravitational field, magnetic field, or other inertial reference. There are differences among the numerous boreholes surveying devices used. These are based on the ability to operate inside steel casing, time-consuming, and complexity to operate. In any case, none of them are clearly perfect.

In general, a survey of a borehole must supply an accurate estimation of the path of the drillhole in three-dimensional space (X, Y, and Z coordinates) of every point along the path that is known. It should be obvious that the greater the number of known data points, the less extrapolation required and the more accurate the survey. The coordinates of points are not measured directly but are computed from measurements of the dip, azimuth, and length along the drillhole.

At present, there is a great variety of instruments for measure deviation, and borehole surveys are carried out routinely in all drillholes of the exploration project. Commonly used measuring devices are based on photographs of a bubble ring and related to an original orientation, such as single or multishot photos, magnetometer/accelerometer based tools, and/or small gyroscope devices, from which azimuth and dip measurements are taken. The probe is lowered into the hole, taking azimuth and dip measurements at prespecified intervals, typically every 20–50 m down the hole, and the data values are transmitted to the surface for processing. The measurements are later used to determine the X, Y, and Z location of each sample. Calculation of corrected positions at successive depths is a straightforward mathematical procedure, if both the location of the top of the drillhole and the initial drillhole inclination are known. In borehole geometry probe, the verticality section includes a triaxial magnetometer and three accelerometers and data from these are combined.

If the surveying is carried out in mostly nonmagnetic rocks and in open hole, then the standard magnetometer/accelerometer system is useful. On the contrary, a non-magnetic gyroscopic device must be necessary if magnetic anomalies are present. For instance, it is the case in ironstone mineralizations, ore with massive pyrrhotite, etc. This instrument uses an inertial navigation system to define the borehole path as it moves.

Logging

Considering the high costs of drilling, a maximum of information must be extracted. Thus, intense geological logging of core and drill cuttings is a common practice. Drillhole information is produced from many sources such as core, chips, down-the-hole geophysical measurements (e.g., caliper, natural gamma radiation, gamma-gamma density, magnetic susceptibility, or resistivity), data from instruments inside the hole such as MDW – measurements while drilling (e.g., pressure at the bit face, temperature, or rate of water flow) – and performance of the drilling machinery. All information related to each drillhole, including topography, drillhole deviation estimations, mapped geological features, and a copy of the data returned, should be available with a single folder for each drillhole (Rossi and Deutsch 2014).

Routine studies of drill cores consist of fracture spacing and orientation, core recovery (including the location of excessive core loss, >5%), lithological description (e.g., color, texture, mineralogy, rock alteration, and rock name), photographic documentation, description of the geological structures visible in the core, preliminary geological profile, rock properties for calculating geotechnical parameters (e.g., RQD), and content and distribution of mineral and ore components, including as possible in situ assaying of ore. Depending upon the objective of the site investigation, a secondary processing can include many other aspects such as the presence and content of clay minerals, total carbonate content, organic components, grain-size distribution, sediment matrix and cement, porosity, pore-size distribution, and many others. The description must be quantitative and systematic, avoiding as much as possible qualitative descriptions. Since structural features must be captured before split the core, the most useful way is to take photographs of the wet core previous the logging process with the objective of producing a permanent photographic record. In noncore drilling, descriptions must be again systematic and quantitative. The data from core and noncore observation are plotted on graphical core logs and utilized to help in interpreting the geology of the present and next holes to be drilled.

Regarding RQD, it is used as a standard parameter in drill core logging and forms a basic element value of the major mass classification systems such as rock mass rating (RMR) system and Q-system. In rock quality designation (RQD), the lengths of all sound rock core pieces that are greater than 100 mm in length are summed and divided by the length of the core run to obtain the final value in percentage. This parameter is commonly estimated where the rock has been altered and/or weakened by weathering. This procedure obviously penalizes if the recovery is poor, being useful since poor recovery commonly means poor quality rock.

Since geological logging is commonly a subjective process, this results in inconsistencies in the application of the logging codes. To solve the problem, it is desirable to use methods to objectively classify how mineralized is a sample, for instance, using portable XRF technology (Gazley et al. 2014). Thus, a clear, accurate, and standardized logging procedure is essential to promote uniformity of data through what is commonly a long data-gathering period. It is important to note that as geological information and concepts evolve with time, the context is likely to request the core be relogged (Sinclair and Blackwell 2002).

Although a great number of different logging methods are utilized in the industry, «there are three main logging forms for recording observations on drill core and cuttings: prose logging, graphical scale logging, and analytical spreadsheet logging» (Marjoribanks 2010). An interval is selected in prose logging, being identified by its downhole depth limits, and described in words. It is recommended that this type of logging must be only utilized in a special column (e.g., comments). Graphical scale log forms can include several mapping columns along with extra columns for recording digital data, sketches, verbal comments, etc. (Fig. 3.65). The important

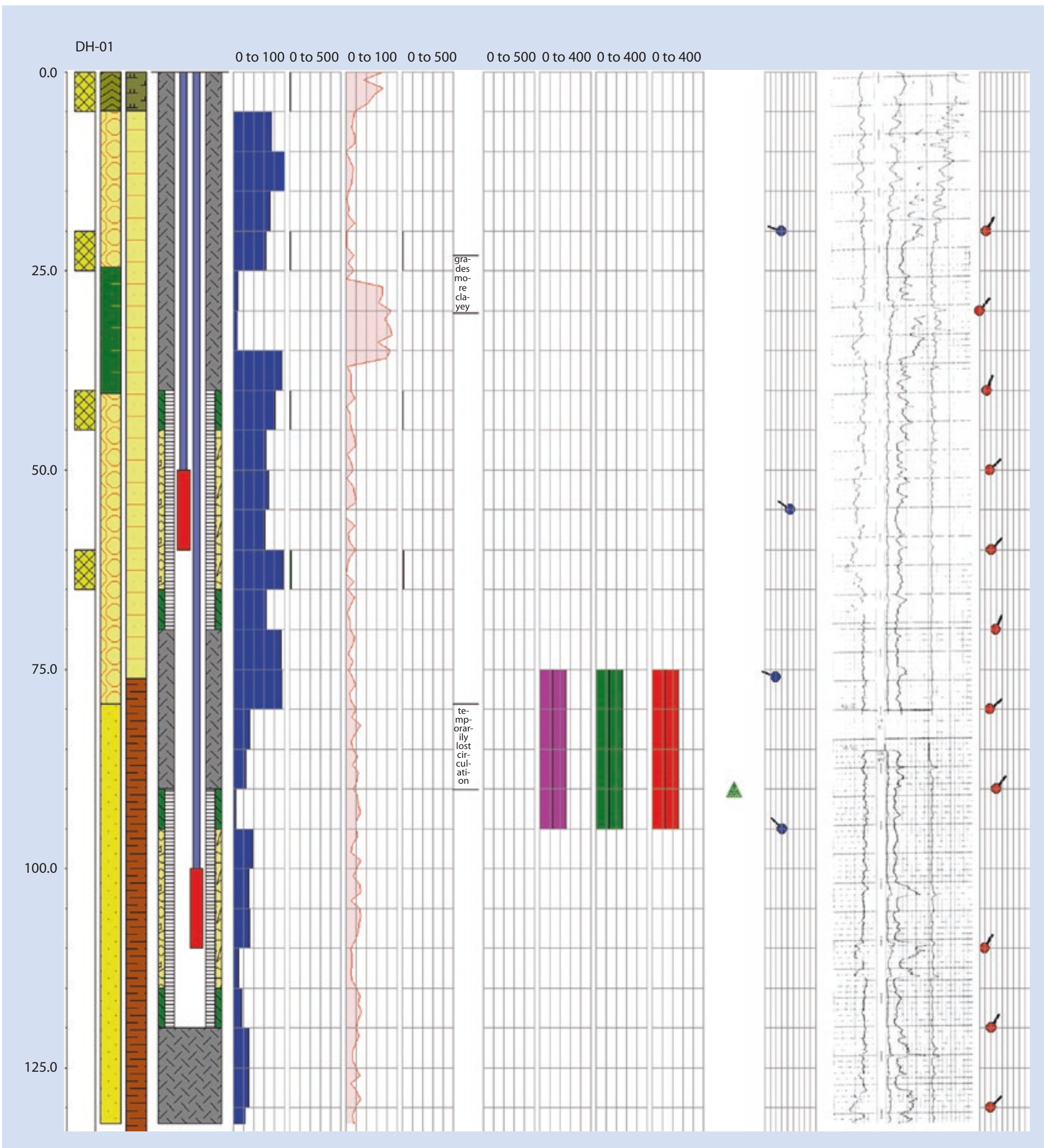


Fig. 3.65 Graphical scale log (Rockworks)

feature about all such logs is that they assemble many different types of geological observations on one form linked by a single down page scale. Finally, the use of spreadsheet logging is indicated in second-phase drilling programs (e.g., resource evaluation and definition) where the main geological problems associated with the ore body have been solved, and the aim of the logging is the routine recording of masses of reproducible data. Regarding the graphical scale logging form, it is usually separated into columns. The columns will be referred to in numbered order from left to right, for example, column 1 (hole depth), column 2 (core recovery), column 3 (sample no.), column 5 (assay results – it will be commonly necessary to devote several columns to insert all assay results), and so on.

3.5 Case Studies

Klaza Gold-Silver-(Lead-Zinc) Project Exploration: Courtesy of Rockhaven Resources Ltd.

The property lies 50 km due west of the town of Carmacks (Yukon, Canada), located 420 km from the year-round tidewater port at Skagway, Alaska (USA). Most of the property is underlain by mid-Cretaceous granodiorite. A moderately sized, late Cretaceous quartz-rich, granite-to-quartz monzonite stock intrudes the granodiorite in the southeast corner of the property and is thought to be the main heat source for hydrothermal cells that deposited mineralization along a series of northwesterly trending, structural conduits. The porphyry dykes are up to 30 m wide and commonly occupy the same structural zones as the mineralization. The dykes are coeval with or slightly older than the mineralization. Mineralization is dominated by gold-silver-rich structures associated with a zonation model ranging from weak porphyry copper-molybdenum centers, outward to transitional anastomosing sheeted veins, and lastly to more cohesive and continuous base and precious metal veins. The metals of primary interest at the property are gold and silver. These metals are intimately associated with lead, zinc, and copper in various forms and concentrations throughout the mineralizing system. The age of the mineralizing events is now considered to be Late Cretaceous. Depth of surface oxidation ranges from 5 to 100 m below surface, depending on fracture intensity, the type of mineralization, and local geomorphology.

The project exploration carried out by Rockhaven systematically advanced through soil geochemistry, followed by excavator trenching, geophysics (both magnetics and EM work very well), and finally through diamond drilling. Early soil sampling identified linear gold ± silver ± lead anomalies, which correspond to some of the known mineralized structural zones, and a large (2000 m by 3000 m) area of moderately to strongly anomalous copper-in-soil response, which partially defines the Kelly porphyry target in the southeastern corner of the property. Grid soil sampling performed from 2010 to 2012 expanded grid sample coverage to the west and north of the earlier grids and collected samples on a few contour-controlled lines in the northwestern part of the property. The grid to collect soil samples were established at 50 m intervals on lines spaced 100 m and oriented at 37°. Soil samples were obtained using a handheld auger and from 30 to 80 cm holes.

Soil Sampling

Effectiveness of soil sampling is often limited by thick layers of organic material and overburden and in many areas by permafrost. Despite these limitations, soil sampling has been an essential and effective surface exploration technique for detecting trenching or drilling targets. Results for gold from historical surveys and Rockhaven's sampling are illustrated in Fig. 3.66.

Historically, excavator trenching in geochemically anomalous areas has been the most effective tool for identifying near surface but non-outcropping, mineralized zones. Within the main areas of exploration, overburden generally consists of 5–20 cm of vegetation and soil organics covering a discontinuous layer of white volcanic ash and 50–125 cm of loess and/or residual soil, which cap decomposed bedrock. Trenching of 22,366 m was performed in 84 trenches between 2010 and 2015. Where possible, trenches were excavated in areas that had previously been stripped of soil and vegetation. The trenches were aligned at about 30°, which is perpendicular to the anomalous trends of the main soil geochemical anomalies. All rock samples (chip sampling) collected from the property were taken from excavator trenches, because there are no naturally outcropping exposures of these zones. Continuous chip samples were collected along one wall of the trench as close to

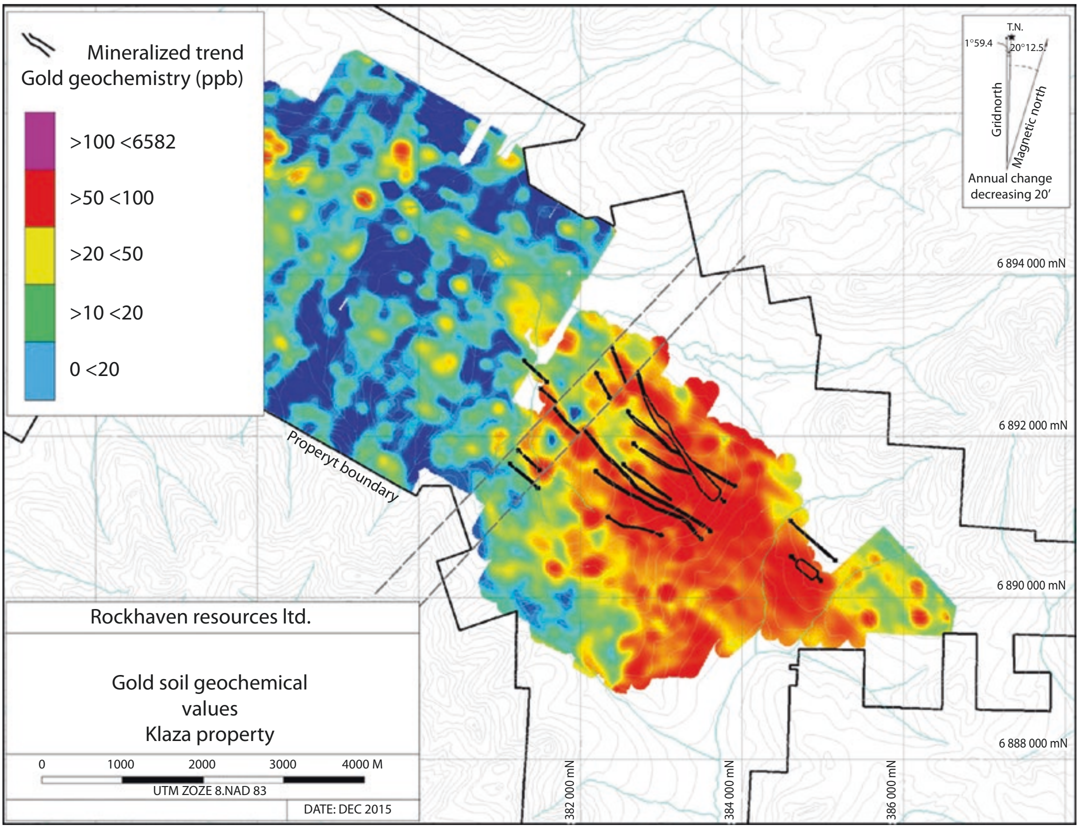


Fig. 3.66 Gold soil geochemical values in Klaza property (Data courtesy of Rockhaven Resources Ltd.)

the floor of the trench as slumping would allow using a geological hammer. Sample sizes averaged approximately 2 kg per linear meter sampled for intervals containing veins and about 1.5 kg per linear meter sampled for intervals comprised primarily of altered wall rock.

Geophysical Surveys

To date, four types of geophysical surveys have been completed on the property: (1) ground-based VLF-EM and magnetic surveys, (2) gradient array induced polarization survey, (3) high-sensitivity helicopter-borne magnetic and gamma-ray spectrometric surveys, and (4) high-resolution induced polarization surveys. The magnetic surveys identified a number of prominent, linear magnetic lows on the property. Subsequent trenching and drilling have shown that many of the northwesterly trending lows coincide with mineralized structural zones, while northeasterly trending breaks in the magnetic patterns correspond to cross faults. These relationships are consistent with the low magnetic susceptibility results that returned from core samples within the altered structural zones compared to higher values from surrounding unaltered wall rocks. Several of the magnetic lows extend outside the main areas of exploration and have not yet been tested by drilling or trenching. Figure 3.67 shows the first vertical derivative of the magnetic data overlain with the interpreted surface traces of the structural zones. Elevated potassic radioactivity is evident in the general area of the main zones in the eastern part of the property but does not specifically coincide with individual mineralized zones. Numerous porphyry dykes and frost boils containing porphyry fragments lie within this area, and they are the probable source of the elevated radioactivity.

The gradient array and pole-dipole IP survey covered a 1800 m by 1450 m area in the east-central part of the property. Readings were collected at 25 m intervals along lines spaced 100 m apart. This survey identified two main anomalies, both of which feature elevated chargeability with coincident resistivity lows. The most prominent anomaly is located in the southeastern corner of

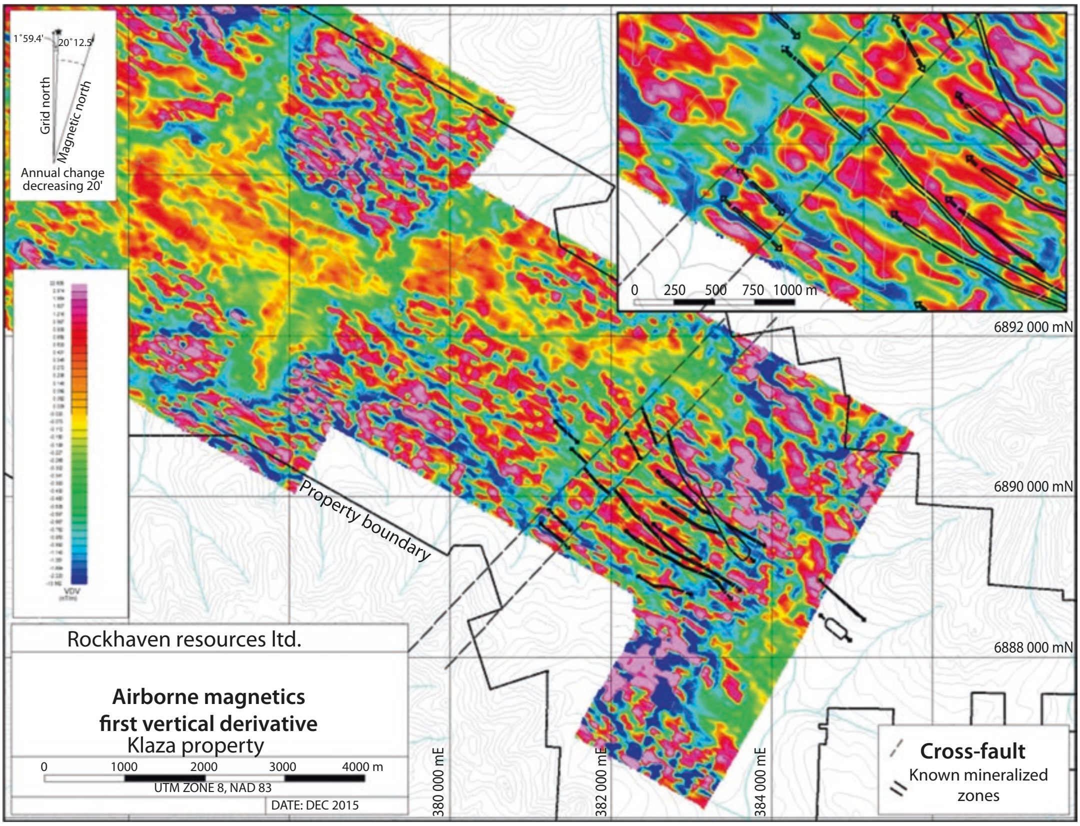


Fig. 3.67 First vertical derivative of the magnetic data in Klaza project (Illustration courtesy of Rockhaven Resources)

the grid and coincides with an area of weak to strong gold-in-soil geochemistry (25–100 ppb) and strong copper geochemistry (>200 ppm) as well as porphyry style mineralization that is part of the Kelly zone. The mineralized vein and breccia zones tested by geophysical surveys show up as resistivity lows that coincide with chargeability highs.

Drilling

Regarding drilling program, a total of 70,099.72 m of exploration and definition drilling was done between 2010 and 2015 in 295 diamond drillholes on the property. All diamond drillholes were collared at dips of −50°, and most of the holes had azimuths of 30–35° (north-northeast). Drilling was completed on section lines spaced roughly 50 m apart. Some of the 2015 drilling was done in part for geotechnical and environmental purposes. To monitor seasonal water levels and frost variations, vibrating wireline piezometers were installed in four holes and a thermistor was installed in one hole. Five diamond drillholes totaling 308.76 m were drilled vertically, peripheral to the mineral resource areas as water monitoring wells. In general, core recovery was good, averaging 95%, excluding the near surface portions of the holes where core recovery was poor. Final hole depths within the Klaza zone averaged 251.49 m, which included a maximum hole depth of 550.77 m. To determine the deflection of each drillhole, the orientation was measured at various intervals down the hole. Measurements taken and recorded were inclination, azimuth, temperature, roll angle (gravity and magnetic), as well as magnetic intensity, magnetic dip, and gravity intensity.

As an example of the results obtained in the main mineralized zones, the host rock of the mineralization in the central Klaza zone is an extensive complex of steeply dipping veins, breccias, and sheeted veinlets. The strongest veins are typically found along dyke margins. Pyrite, arsenopyrite, galena, and sphalerite are the main sulfide minerals in this subzone. Excellent results from

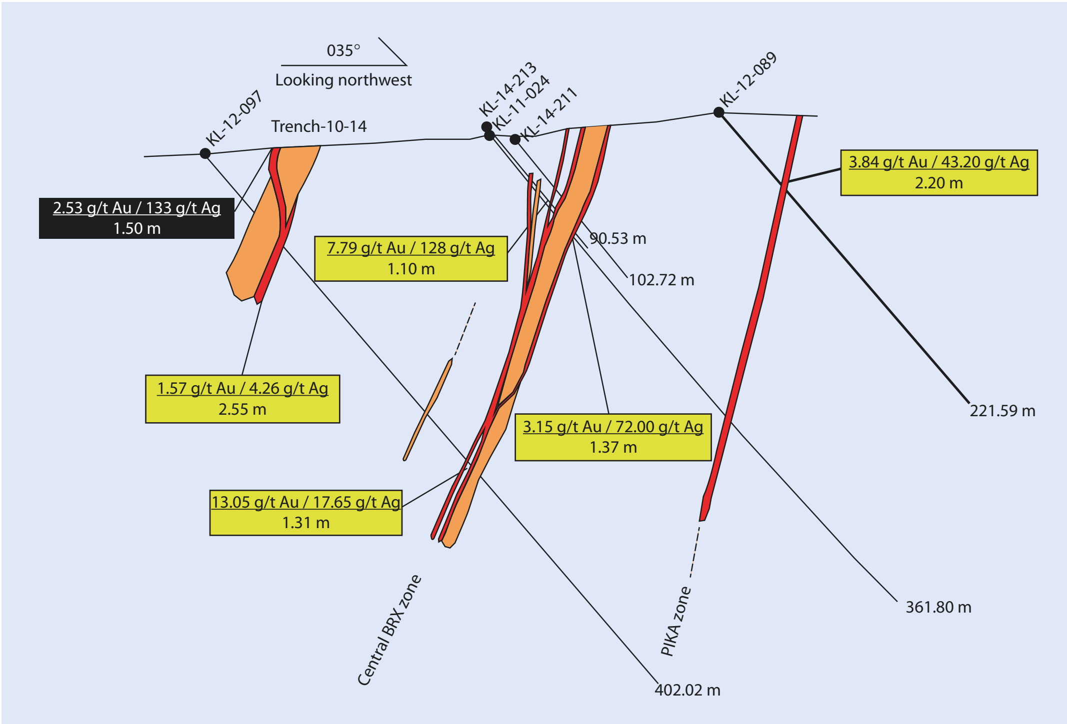


Fig. 3.68 Type section depicting the geometry of the mineralized veining relative to the dyke and the gold and silver grades values obtained in the samples (Illustration courtesy of Rockhaven Resources Ltd.)

this part of the Klaza zone were reported from an interval in KL-10-07, which graded 7.10 g/t gold and 259 g/t silver over 15.25 m, and an interval in KL-12-133, which graded 11.85 g/t gold and 5.24 g/t silver across 6.65 m. Figure 3.68 shows the results of central Klaza zone. Finally, a geotechnical log was carried out previous to geological logging and included determinations of core, rock quality designations (RQD), hardness, and weathering. In 2015, fracture frequency, joint sets, and joint set roughness, shape and infill were also recorded.

Ekati Diamond Project Exploration: Courtesy of Dominion Diamond Corporation

The Ekati diamond mine is located in northwest Canada, 200 km south of the Arctic Circle. Cold winter conditions are predominant in the region for most of the year. The area is a wildlife habitat, where human activities are limited to hunting and fishing. The geology of the Ekati project area consists mainly of Archean granitoids, intruded by metagreywackes and transected by Proterozoic mafic dykes. Bedrock is overlain by less than 5 m thick quaternary glacial deposits. The 45–75 Ma kimberlites of the Lac de Gras kimberlite field intrude both the granitoids and metagreywackes. The kimberlites are mostly small pipe-like bodies controlled by tectonic fissures and typically extend to depths of several 100 m below the land surface. The mineralization is mostly limited to olivine-rich resedimented volcaniclastics and primary volcaniclastics. Diamond grades from the kimberlites range from less than 0.05 cpt to more than 4 cpt.

Diamond exploration in the area started with heavy mineral sampling from fluvial and glaciofluvial sediments, which was followed by mapping of geomorphological features and field observations. Till sampling coupled with ground geophysics pinpointed the Point Lake kimberlite pipe, which was later investigated by core drilling and confirmed as diamondiferous kimberlite. Approximately 15,000 till samples were taken during the project exploration phase. They were also used to search for airborne geophysical anomalies. The extent and chemistry of the indicator minerals dispersion trains were evaluated to pinpoint drill targets.

Indicator Mineral

Kimberlite indicator mineral (KIM) compositions were outstanding in the exploration program leading to the development of the Ekati mine. Discovery of the first kimberlite at Point Lake was followed by the identification of over 150 kimberlite bodies within the Ekati areas. The use of KIM geochemistry was adopted to prioritize likely high-grade phases for follow-up bulk sampling and/or diamond drilling programs. The method involved selecting representative samples from the drilling and recovering a full suite of KIM's from each sample. The recovered grains (garnet, chromite, ilmenite, clinopyroxene) were analyzed by electron microprobe for major elements and by inductively coupled plasma mass spectrometry (ICPMS) for nickel.

Geophysical Surveys

The Ekati area was explored using helicopter-borne total field magnetic (TFM), electromagnetic (EM), and very low-frequency electromagnetic (VLF) surveys. Final exploration sweeps were carried out using an improved airborne EM system with tighter line spacing, reduced sensor height, and airborne gravity gradiometer (Fig. 3.69).

The ground geophysical surveys were used to gather more precise kimberlite/non-kimberlite target discrimination and estimates of pipe size. The surveys were completed on both the majority of the drill targets and all of the pipes with reported mineral resource estimates. A small core hole seismic survey was designed in the Koala pipe, this searching for detailed spatial information of the kimberlite body. The data proved that the borehole seismic technique could augment drillhole pierce points with seismically determined pipe wall contacts.

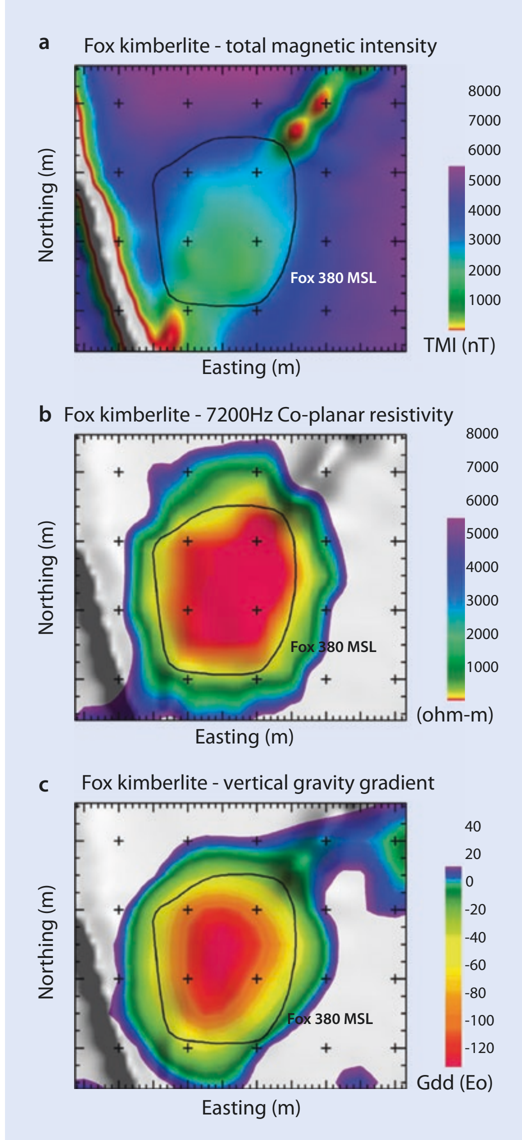


Fig. 3.69 Fox kimberlite airborne geophysical response; the Fox kimberlite has a weak and normal magnetization **a**, a strong conductive response **b**, and a very strong gravity response **c** (Illustration courtesy of Dominion Diamond Corporation)

Drilling

Drilling lasted from 1991 until 31 July 2016 and included 1389 core holes (254,490 m), 111 sonic drillholes (2596 m), and 513 RC holes (106,547 m). Core drilling using synthetic diamond-tipped tools and/or carbide bits contributed to define the pipe contacts, wall-rock conditions, and internal geology. Prior to 1995, the diameter of drillholes ranged from 27 to 71 cm; from 1995 to 2008, the holes' diameter was standardized to between 31 and 45 cm. In order to obtain larger samples, drillholes' diameters for the 2015 and 2016 programs ranged from 45 to 61 cm. Core drilling was also used for gathering geotechnical and hydrogeological data. Figure 3.70 shows the location of the drillholes. Forty kimberlite occurrences were subsequently tested for diamond content using reverse circulation (RC) drilling and/or surface bulk samples.

Sonic drilling was used to core both soil and bedrock in Ekati. The primary objective of sonic drilling was to characterize the nature and

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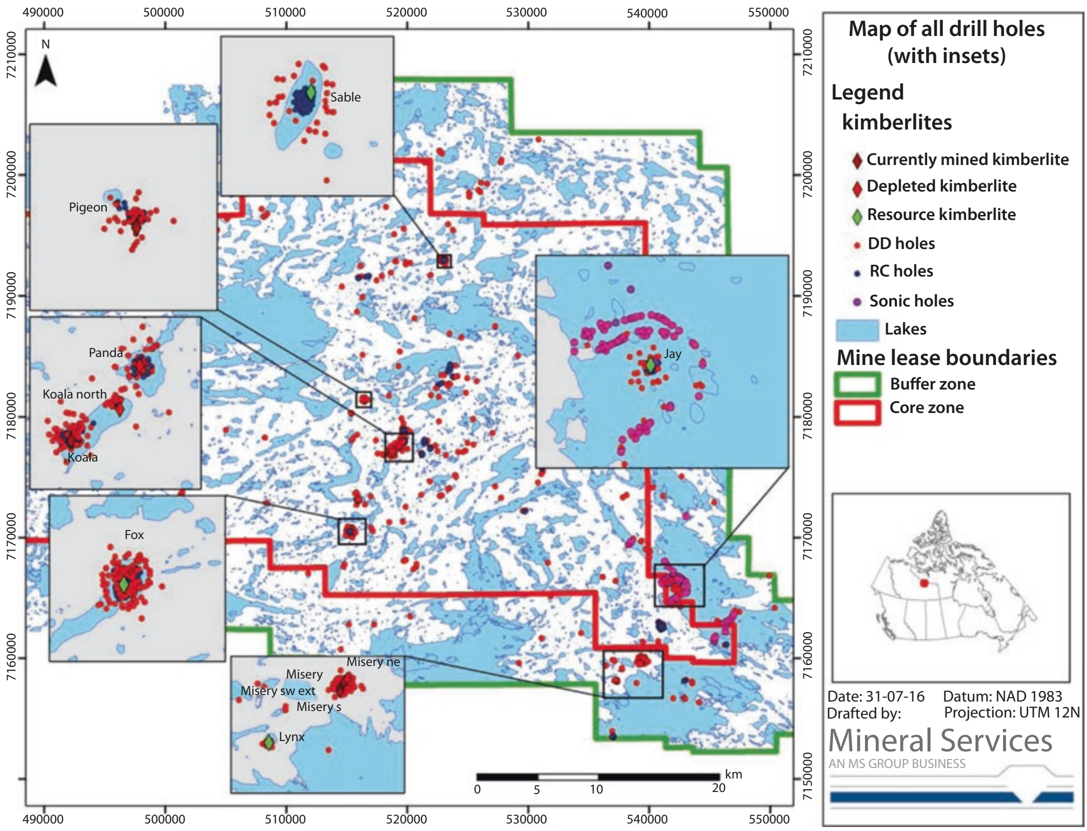

Fig. 3.70 Map showing location of all drillholes with insets for pipes with reported mineral resources for the Ekati project area; approximately 350 geophysical and/or indicator dispersion targets were drilled, with a total of 150 kimberlites discovered (Illustration courtesy of Dominion Diamond Corporation)

variation of the soil layers to determine the depth to bedrock. In addition, recovered soil was logged and geotechnical laboratory testing was performed on selected samples. After reaching the final depth of investigation at each borehole location, in situ hydraulic conductivity testing was carried out.

Since core recovery was largely a function of the hardness of the kimberlite, recoveries of 95–100% for both core and RC drillholes were common within wall rock. In kimberlite, the core recoveries were as low as 20% and as high as 95% but were more typically in the 75–85% range. For RC drillholes, kimberlite recoveries ranged from 50% to over 100% in cases of in holes sloughing. All core and RC drillhole collars were surveyed with GPS instruments prior to and after drilling in order to ensure that the drillhole collar location error is minimal. For core holes, downhole surveys were done with industry standard instruments. Three tools, including the tool for gyroscopic deviation surveying, the three arm caliper, and the tool for conductivity induction and natural gamma readings, were used on all RC holes.

Oriented core was used for geotechnical investigation of the wall rocks but not in kimberlite. The following geotechnical parameters were determined for all core drillholes: (a) percentage core recovery, (b) rock quality designation (RQD), (c) fracture frequency, (d) point load strength index, and (e) joint condition and water.

Digital geological and geotechnical logging was completed and the core photographed before being stored in appropriate building. Color photographs were taken of delineation drill core and used to verify significant contacts and lithologies as well as provide a permanent record of the drill core. Geological logging used digital logging forms for both wall-rock lithology, kimberlite/wall-rock contacts, and internal kimberlite lithology. Kimberlite cores were examined macroscopically and using a binocular microscope to determine concentration of macrocrystic olivine, matrix composition, abundance and type of country rock xenoliths, approximate abundance of indicator minerals, rock fabric, color, and alteration. Samples were taken from core holes for determination of dry bulk density and moisture content of host rock and kimberlite. In the opinion of the responsible QPs, the quantity and quality of the lithological, geotechnical, density, collar, and downhole survey data collected in the drill programs were sufficient to support mineral resource and mineral reserve estimation.

Matawinie Graphite Project Exploration: Courtesy of Nouveau Monde Mining Enterprises Inc.

The Matawinie project is spread over an area of approximately 70 km by 50 km, being the center of the most important block (Tony Block) positioned approximately 120 km north of the city of Montréal (Canada). The Matawinie property lies in the southwestern portion of the Grenville geological province. The belt hosts the only currently producing crystalline flake graphite mine in North America. The area includes a great variety of rock types such as paragneiss and calc-silicates. Granitic and pegmatitic intrusions also occur and are located occasionally on the property. The graphite mineralization is hosted in paragneiss horizons and appears as disseminated graphite flakes. The graphitic paragneiss occurs as layers a few centimeters to several meters thick and can often be followed along strike over tens to hundreds of meters. This rock type visually contains approximately 0.5–3% disseminated crystalline graphite. The graphitic horizons are interbedded with garnet paragneiss units displaying low graphite content and ranging from a few centimeters to tens of meters in width. The mineralized zones are limited by garnet paragneiss on the exterior side of the main circular conductive anomaly while charnockite granitic gneiss (hypersthene granite) occupies the internal portion of the circular anomaly.

Geophysical Surveys

Given the contrasted physical properties of the graphite mineralization sought after, geophysics was the key component of the exploration program and in particular time domain electromagnetic techniques (TDEM). The exploration strategy was twofold. First, it implied large regional airborne surveys with a wide line spacing to detect pluri-kilometric conductors, which were then followed up with ground prospecting/sampling. Second, areas with mineralization of interest were then flown again at the local scale with a tighter line spacing to define areas with better potential for thick and continuous mineralized envelopes. These prioritized areas were then surveyed with ground-based TDEM technology to allow high-resolution delineation of the sub-outcropping parts of conductors. This information was used to quickly plan trenching and drilling for efficient sampling of the mineralization.

With graphite being significantly more conductive than host rocks, the first step in the exploration stage was to carry out an airborne time domain electromagnetic (TDEM) survey. Thus, a regional heliborne magnetic and TDEM survey was carried out over an area of 55 km × 72 km at a 1 km line spacing. Several anomalies were detected and ten local areas were selected for detailed surveying using the same configuration, but at a 100 m line spacing. The surveys were successful in outlining several large-size conductors (Fig. 3.71). In order to assess the multiple

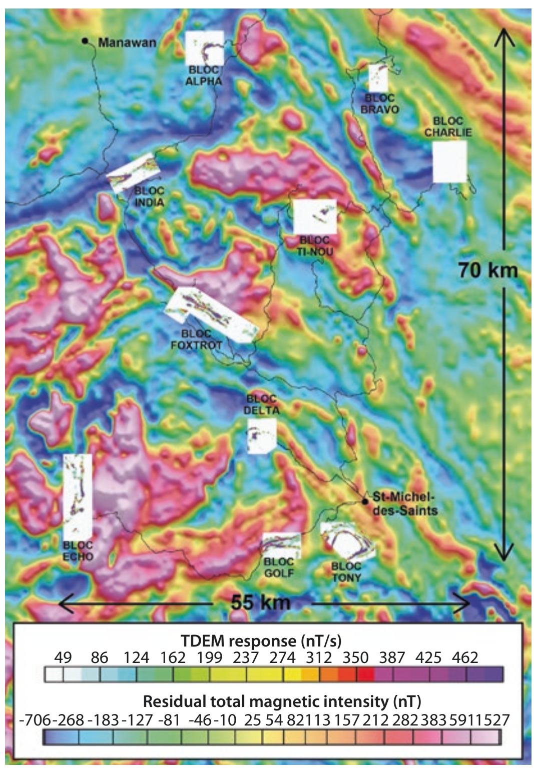

Fig. 3.71 Local areas flown with heliborne MAG-TDEM (Illustration courtesy of Nouveau Monde Mining Enterprises Inc.)

resulting targets in an efficient manner, a quick prospection campaign was deployed. It was supported by the use of a very small EM device with a penetration capability estimated at 1 m. This effort resulted in the collection of 35 grab samples grading between 5% and 17% graphitic carbon. Based on these preliminary results and the potential size of the conductors estimated from the airborne surveys, several areas were selected for further assessment.

In order to get an accurate image of the sub-outcropping portion of the conductors, a ground TDEM system was used. The system has a limited penetration depth estimated in the order of 10–15 m but offers high spatial resolution, being the unit equipped with an integrated GPS. The survey was carried out along existing roads and trails, along the 100 m spaced network of lines cut for other geophysical techniques, and finally along a local 20 m spaced set of lines specifically designed for this instrument. The overall results obtained with this system proved very useful to map the sub-outcropping conductors. This type of high-spatial-resolution information enables a significant gain for understanding the geometry of ore bodies close to surface. It served as a guide for strategically locating exploratory trenches and drillholes, especially in this geological area that underwent strong deformation. Everywhere a trench was dug or a hole drilled based on these results, graphite and/or sulfide mineralization was found and could explain the anomalies. The overburden encountered in the drillholes of the area varied from 0.4 to 5.5 m, with an average of 3.5 m.

Other classic geophysical techniques were used to better define the conductors. A horizontal loop EM (HLEM) survey was performed every 25 m with a 100 m cable using three frequencies. With its estimated penetration depth of 50 m, the information provided by this survey was especially useful at locating conductors with significant vertical extensions and those with their top located deeper than the penetration depth of the previous system. It also enabled some estimation of the dip, conductance, and depth to top of conductors. A resistivity/induced polarization (IP) survey was also carried out at a 12.5 m station spacing (for increased resolution) with ten receiving dipoles using the pole-dipole configuration (for increased penetration depth). The conductors identified with this electrical method conformed well with those detected using EM methods. Some poorer, more subtle, conductors were also outlined. Furthermore, the chargeability model highlighted some areas where disseminated graphite and/or sulfides may occur in addition of the conductive occurrences. In addition, the 2-D section models of the IP data were especially useful for drillhole planning.

Magnetic data was also gathered in an effort to try to discriminate weakly magnetic conductors, likely relating to low sulfide graphite occurrences, from strongly magnetic conductors for which higher sulfide concentration may occur (◘ Fig. 3.72). In one instance where a drillhole had intersected a graphite-rich horizon at a depth much greater than expected (85 m instead of about 20 m), a borehole Mise-À-La-Masse survey (MALM) was carried out with a 12.5 m spacing to verify which of the sub-outcropping conductors this deep intersection was connected to. The MALM survey proved that the deep graphite occurrence was connected to shallow conductive units further to the northwest rather than nearby, indicating some local discontinuities.

The 2014 and 2015 ground TDEM surveys delineated wide conductive areas over each of the targeted mineralized zones. As a result, four trenches were excavated in 2014 and five in 2015 (◘ Fig. 3.73). Trenches were oriented roughly perpendicular to the foliation of the paragneiss units and mineralized horizons with the exception of one trench, which was at about 45° to the foliation because of terrain constraints. In 2014, the trenching program aimed at sampling only mineralized material along the trenches in order to determine the potential of the mineralization, while in 2015 channel sampling usually started 2 or 4 m (1–2 sample lengths) outside the visible mineralized area and was collected in a continuous manner as to prevent any sample bias. Trenches were approximately 1.5 m in width and varied from 0 to 4 m in depth. In some instances, large boulders, the accumulation of water and prohibitive depth prevented the excavation and/or sampling of portions of the planned trenches.

▪▪ Drilling

Drilling on the Tony Block targeted wide conductors on each of the main conductive areas outlined by the 2015 ground TDEM survey. A total of 70 holes were drilled for a total of 10,479 m. As an example, the drilling on the southeast zone of the south deposit consisted of nine holes for a

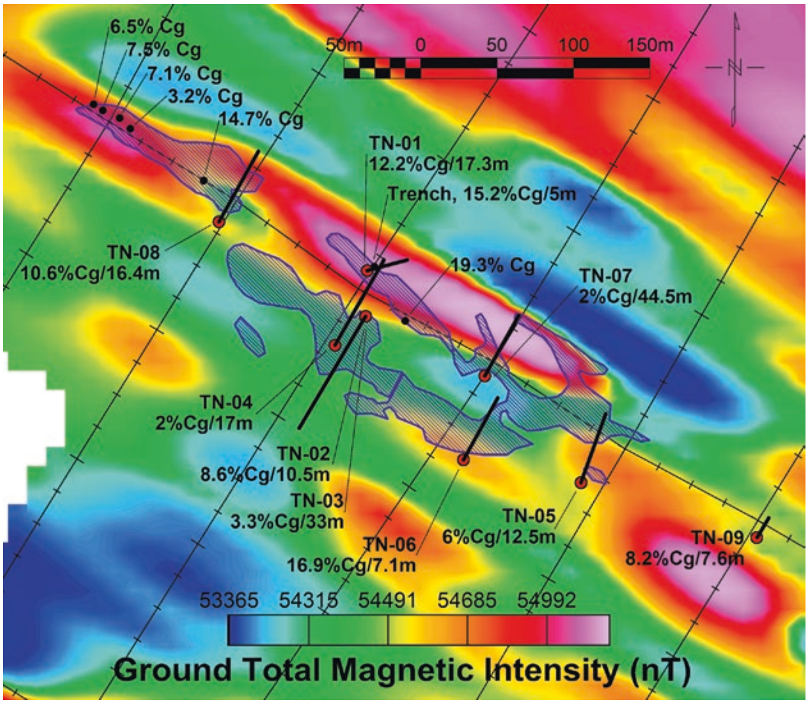

Fig. 3.72 Ground magnetic survey results (Illustration courtesy of Nouveau Monde Mining Enterprises Inc.)

Fig. 3.73 Part of a trench including a channel sampling (Image courtesy of Nouveau Monde Mining Enterprises Inc.)

total of 1552 m drilled. Mineralization was intercepted 13 times by drilling here resulting in the interpretation that the southeast zone is composed of two main mineralized horizons (S1 and S2). From Section S2600 to Section S2900 (300 m length), the mineralized horizon ranges from 116 to 159 m true width, with grade varying from 3.18% to 3.61% Cg. The drilling on the southwest zone of the south deposit consisted of 22 holes for a total of 2617 m drilled. Mineralization was intercepted 57 times by drilling here resulting in the interpretation that the southwest zone is composed of two main mineralized horizons (S1 and S2). The highlight of southwest zone is a first graphitic horizon (S1) about 29 m thick. It is followed by a mainly barren interval between 24 and 62 m thick and finally a second graphitic horizon (S2) around 40–50 m thick, with both graphitic horizons varying from 2.79% to 5.29% Cg.

Coringa Gold Project Exploration: Courtesy of Anfield Gold Corp.

The Coringa gold project is located in north central Brazil in the mining friendly state of Para, 65 km south of Novo Progresso. The area occurs in the southeastern part of the Tapajós mineral province where past production is estimated at 30 million ounces of gold. The claims are underlain by Proterozoic granites and rhyolitic volcanics, and the main structural trends are northwest and north-northwest. The Coringa shear-vein system (high-grade gold mineralization is hosted in a series of narrow quartz-sulfide veins that range in thickness from 0.15 to 4 m) is coincident with the north-northwest trend (345°) and dips 70–90° to the northeast. The main shear is 7 km long and five zones of vein mineralization occur along it. Many other mineralized structures are also present.

Geophysical Surveys

The exploration program initially focused on determining drilling targets. These targets were identified through artisanal workings, geological mapping, airborne geophysics, and ground IP surveys, along with rock and soil sampling. An airborne magnetic-gradiometric and gamma-ray survey was carried out during 2007. The airborne survey covered 549 km^2 with a 200 m grid spacing and at an altitude of 100 m. A 34 km line induced polarization (IP) dipole-dipole geophysical survey over two zones and a 70.7 km line IP (induced polarization) dipole-dipole geophysical survey over approximately 7.0 km of gold-bearing structures were later completed. A time domain electromagnetic geophysical survey of 860 km line was flown to cover all identified pan concentrate and gold-in-soil anomalies.

Rock and Soil Sampling

Rock and soil sampling were carried out in several phases. Gold and 34 other elements were assayed in the samples. Soil sampling was carried out using a 100 m by 25 m sampling grid together with 18 trenches. For soil sampling, a baseline was set up perpendicular to the soil line orientation. The topsoil (between 0.3 and 0.5 m deep) was removed and a 0.5–0.7 kg sample was collected from the following 0.5 m below the topsoil. Samples were placed in a plastic bag and tagged. A brief description that included color of the sample, percentage of gravel, sand, and silt was carried out. All field information was controlled by the geologist in charge of the soil survey and digitized into the data base before sending the sample to the laboratory for gold analysis. In trench sampling, a start point was located with a handheld GPS, and azimuth and trench length was estimated with a compass and tape. Trenches were hand dug to a depth of 1 m. In these trenches, approximately 2–3 kg chip-channel samples were collected at 1–1.5 m intervals. Finally, a stream sediment sampling program was also carried out, being collected a total of 756 samples.

Drilling

In drilling program, four drilling phases have been completed on targets identified at the project site for a total of 24,093 m of HQ core in 160 exploration holes. In the first phase of drilling, 1774 m in 22 holes was carried out for early stage exploration, being drilled under the main artisanal workings («garimpos») (Fig. 3.74). The second phase drilling includes 5032 m in 44 holes, and

Fig. 3.74 «Garimpo» or artisanal working (Image courtesy of Anfield Gold Corp.)

Table 3.5 Drilling summary by phase

Phase	Holes	Holes with downhole survey	Meters	Samples	Meters sampled
1	22	0	1774	1922	1717
2	44	42	5032	1711	1370
3	15	12	1979	434	333
4	79	66	15,308	5227	4752.83
Total	*160*	*120*	*24,093*	*9294*	*8172.83*

Data courtesy of Anfield Gold Corp.

Fig. 3.75 Sampling procedure (Images courtesy of Anfield Gold Corp.)

the aim was further defining of the resources in several blocks. Regarding the third phase, drilling was 1979 m in 15 holes; the main objective was to define the resource in one block and test two of the IP targets. Finally, the fourth phase of drilling covered 15,308 m in 79 holes), being the goal to test the continuity of the Mae de Leite structure at depth and along strike as well as the continuity of the Meio zone along strike to the north and south.

Table 3.5 summarizes the drillholes completed within each phase of drilling.

The sampling procedure in holes (Fig. 3.75) includes the continuous sampling of the core at intervals of approximately 0.5 m (mineralized zones) to 1 m (non-mineralized zones). In this process, a cutting/splitting guide line is marked on the core by the geologist to ensure that the mineralized structure is equally divided, each box is photographed to provide a visual record of the core, and then one-half of the core is returned to the core box, while the other half is placed in the numbered and tagged sample bag.

Atacama Copper Project Exploration: Courtesy of Arena Minerals Inc.

Atacama copper property consists of approximately 920 km^2 (92,000 hectares) in Chile's Antofagasta Region, approximately 40 km northeast from the city of Antofagasta. The property has been almost exclusively explored and exploited for industrial minerals, primarily iodine and/or nitrates. These industrial minerals are found within overburden covered areas and generally within 20 m from surface. As a result, the exploration activities within the property focused on shallow exploration methods, ranging from trenching to short RC drilling in more recent years, which targeted sedimentary layers within the overburden. Most of the Cu porphyry deposits of the region belong to the Paleocene-Early Eocene world-class Cu-Mo porphyry belt which extends from southern Peru to northern Chile for a distance of over 1300 km. Mineralization is associated with a complex of granodioritic to quartz-monzonite stocks with accompanying Paleocene dykes dated at 57 million of years.

A basic outline of the exploration program in place is as follows: (1) data compilation; (2) desktop analysis and target selection; (3) initial ground work and prospecting; (4) prospect generation and selection; (5) follow-up ground work, including additional geology, geochemistry, and ground geophysics; and (6) drill program design, RC drilling in two phases: 2 km grid drilling followed by 1 km grid infill based on results. Thus, the exploration program during 2013–2014 started with an initial phase of target selection using satellite imagery (ASTER) to identify exposed alteration zones and main structural features and trends. This work was combined with regional geological and geochemical data to provide a selection of priority targets for field follow-up with prospecting and sampling. Twenty-nine target areas were selected of which 11 have exposed alteration of various compositions read from the ASTER images. The other targets that do not have surface alteration detectable by ASTER imaging lie under cover or have only small outcrop expressions (less than the 25 m pixel limit). These areas were selected based on their copper-molybdenum geochemical anomalies from the short RC exploration holes carried out by SQM while looking for industrial minerals, known reported or inferred lineaments forming structural corridors and other geological information.

The initial target selection was reviewed again once the company acquired regional airborne magnetic and radiometric data that had been flown as part of a multi-client survey in 1999 (Fig. 3.76). The reinterpretation of this database led to the identification of an additional 23 targets that had not been selected during the initial selection process. The remaining 17 of the reinterpreted magnetic/radiometric targets correspond with previously selected ASTER/geochemical targets.

All alteration zones or mineralized outcrops were systematically sampled, and a preliminary geological map was made of all areas of interest. A total of 1450 rock chip samples were collected during this phase of exploration, and several areas with potential for both copper porphyry and epithermal gold mineralization were identified for additional ground follow-up surveys. Based on the results of the initial fieldwork, five areas were selected for ground magnetics coverage: Cerro Barco, Cerrillos, Quebrada Honda, La Paloma, and Paciencia. A total of 3647 line km of surveying was completed by this survey. Following the ground geophysics, additional mapping and in some cases multispectral analysis were done on selected alteration areas to get a better definition of the targets and define trenching and drilling targets.

The ground magnetic surveys were conducted on north-south lines with a line spacing of 100 m. Readings were carried out with an approximate station spacing of approximately 0.5–1.5 m. A checkpoint was measured twice daily with all the magnetometers. Repeatability of the corrected magnetic readings was within 1 nT, and the GPS UTM coordinate repeatability was within 2 m of the average value. In summary, several large anomalies that may be indicative of large hydrothermal alteration systems were observed, being the magnetic data effectively mapping lithology.

Regarding the exploration results and interpretation, the comparison between the smoothed analytical signal data and what is known of the local geology from either the regional map sheets or the arena mapping allowed for a definition of a set of characteristics for the different lithologies, subject to variations caused by things like burial

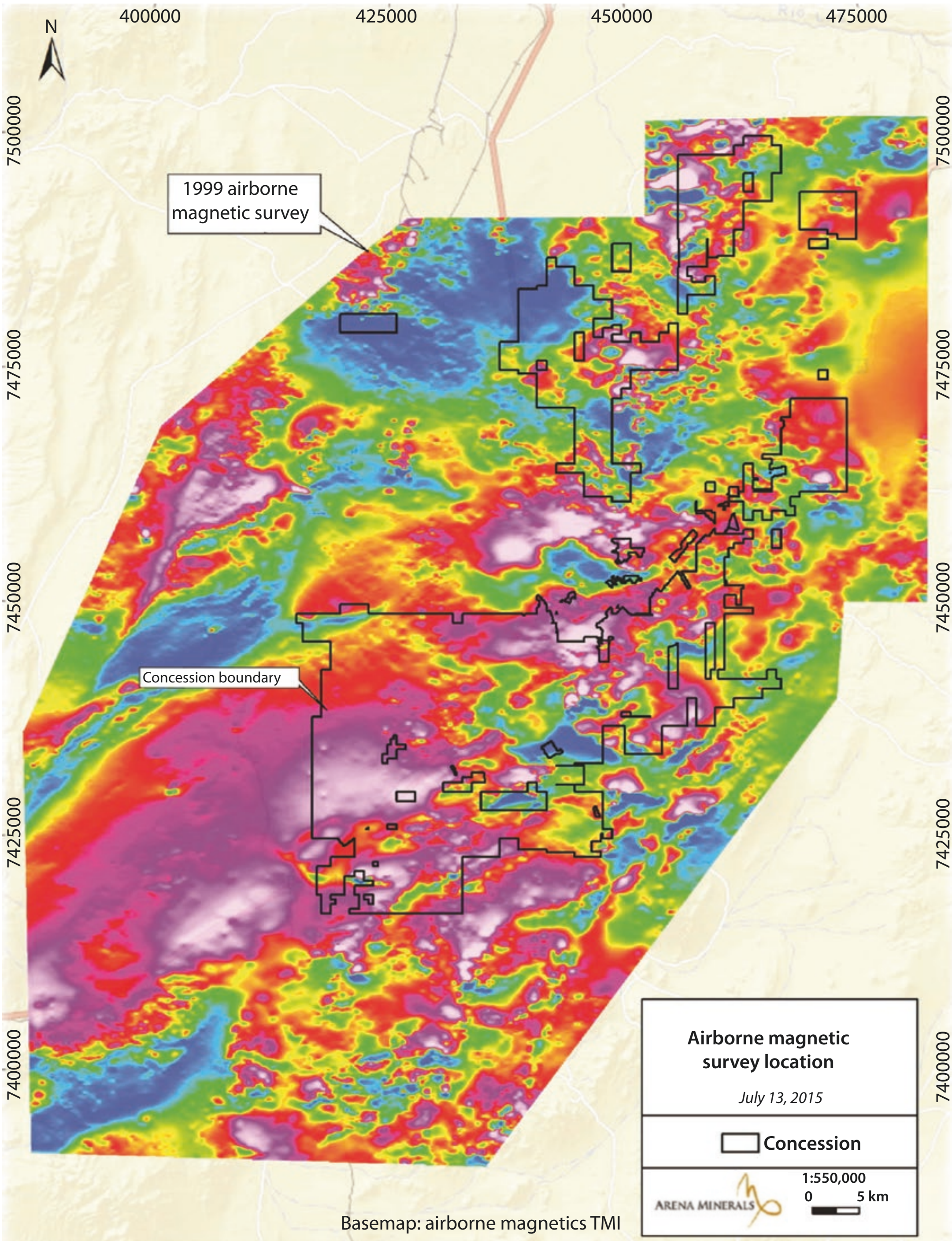

Fig. 3.76 Airborne magnetic survey location (Illustration courtesy of Arena Minerals Inc.)

depth and well-known magnetic characteristics of igneous and sedimentary rocks. Typically, mafic intrusives are more magnetic than mafic and intermediate volcanics that have been subject to oxidation, and in turn, these are more magnetic than granites. In several locations, the lithologies were revised based on magnetic characteristics where the regional mapping indicated lithologies (e.g., Cretaceous granites) that are not consistent with high magnetic gradients measured in the

Fig. 3.77 Crustiform quartz from Paciencia prospect (Image courtesy of Arena Minerals Inc.)

new survey and where the geological map sheet shows Holocene cover and an absence of outcrop.

Pampa Paciencia prospect results, as an example of the exploration program, are the following. The Pampa Paciencia prospect is located approximately 10 km north and northeast of the Sierra Gorda and Spence mines, respectively. The detailed magnetic survey has allowed the interpretation to refine the position of the faults and to recognize that most of the lithologies are fault-bounded in a broadly north-south elongation direction. Two distinct mineralized areas, approximately 2 km apart, have been discovered within altered dioritic and granodioritic intrusive rocks. The mineralization consists of quartz vein outcrops and subcrops of angular quartz fields that align with east-west to west-northwest lineaments. The quartz vein material exhibits well-developed crustiform-colloform textures (Fig. 3.77) and is associated with gold, silver, and base metals anomalies.

The epithermal quartz field consists of a 500- to 800-m-long area of quartz float concentrated along a west-northwest axis immediately south of a large granodiorite outcrop. Several quartz chip samples taken from this area originated anomalous values of gold up to 6.82 g/t of gold. Seventeen other samples from this zone also generated anomalous values ranging from 0.5 to 3.85 g/t of gold. The second area of interest is located 2 km southwest of the epithermal quartz field, consisting of veins with associated quartz-amethyst. Anomalous gold values from chip sampling range from 0.3 to 2.07 g/t and are associated locally with anomalous base metal values in copper (up to 0.12%), lead (up to 0.41%), zinc (up to 0.15%), and silver (up to 154 g/t). The next stage of work on this target should comprise of trenching, sampling, and mapping of the two vein areas prior to exploration RC drilling.

Preston Uranium Project Exploration: Courtesy of Skyharbour Resources Ltd.

The Preston uranium property is located in northwestern Saskatchewan, Canada. The property comprises 121,148 ha and is approximately 32 km long in a northerly direction. Outcrop exposure is limited, generally 5%. Vegetation, weather conditions, and seasons are typical of northern Saskatchewan. The Preston uranium project is located 30 km southwest of the southwest margin of the Athabasca Basin, which is interpreted to have been filled over a 200 Ma period in four major depositional sequences coalescing into a single basin. No significant zones of uranium mineralization have been identified on the property to date but the Athabasca Basin arguably hosts the world's largest and richest known uranium deposits.

Airborne Geophysical Surveys

A 5162 line km combined versatile time domain electromagnetic (VTEM^{plus}) and aeromagnetic survey was completed over six blocks of the Preston property. The survey areas were flown at 200–300 m line spacings with tie lines at 1000 m. Over 300 km of conductor segments, some approaching 10 km in length, occur in the combined eastern blocks of the Preston VTEM coverage. Basement aeromagnetic trends in the furthest western block are oriented northwest-southeast, while those of the eastern blocks are E-NE which is similar to the dominant basement strike orientation at Fission's Patterson Lake South high-grade uranium discovery area. Cross-cutting structural features and flexures affecting the conductor traces were identified to be of particular interest as prospective follow-up targets.

A Goldak high-resolution radiometric survey was flown to locate uranium boulder trains, in situ uranium mineralization and alteration associated with uranium mineralization. The airborne radiometric, magnetic, and VLF-EM survey was flown over one large block extending up to 60 km east-west and up to 36 km north-south flown at 50 m above surface. A total of 8273 line-km on 200 m line spacing was flown on lines at 155°/335°. The airborne radiometric spectrometer coverage mapped a significant number of enhanced radioactive locations that were classified into contributions from uranium, thorium, and potassium sources. Interpretation of the radiometric data identified areas with elevated uranium counts that can be correlated along and between multiple lines potentially indicating the presence of radioactive boulder trains or in situ uranium mineralization. These radiometric features, particularly when coincident with prospective EM conductors, were given high priority for follow-up ground work.

Geological outcrop mapping and identification of boulders and/or boulder terrains were completed over geochemical survey grids (at 200 m line spacing) and on prospecting traverses while ground truthing geophysical anomalies. Geological traversing and mapping and sampling of the various rock types were aided by ground radiometric surveying. Areas with high topography were chosen for geological mapping traverses based on coinciding airborne radiometric anomalies and strong EM conductors. Geological outcrop and structural mapping was completed at a scale of 1:5000 in selected areas. The dominant lithology was moderately to steeply dipping, northeast trending, weakly to moderately foliated granite. Further to the northeast, to the extent of the Preston tenure boundary, diorite-to-gabbro and granite-to-granodiorite outcrops are mapped along the same intermediate airborne magnetic northeast trend. Radioactive pegmatites (>2000 cps) intrude granite to granodiorite to the northeast.

Water, Sediment, and Soil Sampling

Lake-bottom water and sediment sampling were regularly collected together at the same site. Samples of lake sediment were collected using a tubular steel instrumentation, fitted with a butterfly valve that opens an impact with the sediment and closes as the sample is retrieved, and trapped the containing sediment. The sampler is designed so that once retrieved, it can be inverted and the contained sediment poured into a sample bag. Sample control was by GPS with sub 5 m accuracy. Thematic plotting was completed for As, Au, Co, Cu, Li, Mo, Pb, U, Th, Y, and Zn and assessed for spatial associations with known geological, radon, and geophysical features. Statistics for select elements of interest are tabulated in Table 3.6. For

Table 3.6 Select lake sediment statistics

n = 260	U_ppm	Pb_ppm	Pb206_ppm	Co_ppm	Au_ppb	Y_ppm
Max	*2.60*	*19.74*	*4.66*	*42.90*	*7.30*	*39.49*
Min	0.05	0.55	0.13	0.50	0.10	0.65
Average	0.63	3.31	0.83	8.10	0.50	8.01
Stdev	0.46	2.31	0.57	6.47	0.78	7.09
50‰	0.50	2.81	0.70	6.20	0.30	5.66
78‰	0.80	4.02	1.03	11.01	0.70	11.08
90‰	1.20	5.35	1.40	16.19	1.10	16.84
95‰	1.71	6.60	1.62	20.98	1.51	22.47
99‰	2.20	12.66	3.12	29.42	3.30	33.79

Data courtesy of Skyharbour Resources Ltd.

the uranium lake-bottom sediment results, a total of 7 out of 260 samples collected in 2013 are above the 99th percentile. This cluster of samples is also strongly anomalous in Co, Cu, Nb, Y, and Zn.

Regional soil sampling grids were completed, for the most part, between 200 and 400 m line spacing and 100–200 m sample spacing orthogonal to EM conductors and/or radiometric anomalies. Over 700 B-horizon samples were collected with sampling generally avoiding muskeg. The soil profile comprises 0–15 cm of moss or pine needles covering a thin 0.1–1 cm organic humus layer, then into a generally beige- to white-colored unconsolidated pebbly sand. The B-horizon selected for sampling was identified in the field as an abrupt transition from the above beige or white sand to a brown or orange sand typically occurring between 15 and 85 cm depth. Thematic plotting was completed for Ag, As, Au, Ce, Co, Cu, Li, Mo, Pb, U, Th, Y, and Zn and assessed for spatial associations with known geological, radon, and geophysical features. Uranium anomalies in soils are generally limited to one or two adjacent station anomalies. Two of the most significant multi-station soil anomalies in the north-west to north central fin area are spatially associated with mapped granitoid outcrops with significant topographic relief. The highest U value for 2013 came from the west central portion of the Swoosh target, adjacent to the projected map extension of pelitic sediments. This sample returned 7.90 ppm U with >95th percentile values for Cu and Y and greater than 80th percentile As and Pb and positive Pb isotope systematics.

Biogeochemical Sampling

Regional biogeochemical sampling was completed on geochemical survey grids in conjunction with soil sampling. Black spruce was selected as the preferred vegetation medium due to its proven ability to concentrate many elements and widespread availability in both well-drained and poorly drained areas. Previous studies also identified Jack pine as a suitable biogeochemical medium. These species was selected as a secondary target vegetation type, due to its widespread distribution in the property area. Thus, twigs with attached needles were collected from around the circumference of an individual tree within 20 m of each soil sampling site. Numerous field parameters were collected including tree height, twig length and diameter, soil moisture conditions, slope, aspect, and any other factors that would affect sample quality. The three different tree species have differing background values on an element by element basis, so it is critical that plots showing biogeochemical results be levelled to account for these differences.

In Situ Radon-in-Soil

In situ radon-in-soil measurements were taken adjacent to the site of soil sample (hole). A hand-operated auger was used to drill a hole approximately 2.5 cm in diameter to a depth of approximately 65 cm. Net radon results are given in counts per minute (cpm). Radon-in-soil analysis was completed at a total of 181 sample sites, most of which have corresponding soil sampling completed for ICP analysis. Values for radon ranged between 0 and 26 counts per minute. In most areas, the spacing and sample density were too low to establish significant anomalies when viewing the radon-in-soil data alone. Other samples such as lake-bottom water samples were also collected and measured.

Ground Gravity Surveys

The targets for land-based gravity surveying were selected based on favorable geology and structure, coincident geochemical survey (lake sediment, radon-in-water, radon-in-soil, and/or biogeochem), and airborne geophysical survey results from the 2013 exploration program. Prioritization was given to discrete sub-kilometric ovoid gravity lows potentially associated with desilicification, clay alteration, and other alteration typically found in uranium deposits. The 2014 ground-based gravity survey consisted of gravity stations collected on survey lines spaced at 400 m with a station spacing of 50 m. A horizontal loop electromagnetic survey (HLEM) was later carried out. The targets were selected for HLEM surveying to more accurately define airborne VTEM conductors of interest refined by the geological, geochemical, and gravity results.

Drilling

Finally, two diamond drilling programs were carried out in 2014 and 2015. The drill core was descriptively logged by the geologist on site for lithology, alteration, mineralization structure, and other geological attributes with the pertinent data entered into a database. Handheld spectrometers were used to measure the radioactivity of the drill core and aided in the selection of zones for sampling. The core was sampled based on radioactivity, alteration, and structure of the core with sample intervals typically 0.5–1 m in length.

Ilovica-Shtuka Gold-Copper Project Exploration: Courtesy of Euromax Resources

The Ilovica property is located in the southeast of Macedonia, about 16 km to the border with Bulgaria. Ilovica is a porphyry copper-gold deposit, situated in a northwest-southeast striking Cenozoic magmatic arc that covers large areas of central Romania, Serbia, Macedonia, southern Bulgaria, northern Greece, and eastern Turkey. It is more or less 1.5 km in diameter, being associated with a badly exposed dacite-granodiorite plug and emplaced along the northeastern border of the northwest-southeast elongate Strumica graben. The exact location of the deposit is controlled by major north-south crosscutting faults and minor northwest-southeast faulting, parallel to the faulted border of the graben. Alteration related to tertiary magmatic activity at Ilovica is variably present over an area of approximately 8 km^2. Pervasive alteration is largely confined to a roughly 1.5 km^2 area in and adjacent to the main intrusive complex. Smaller areas of pervasive and structurally-controlled alteration extend somewhat asymmetrically to the south and east of the intrusive complex.

Regarding the mineralization, the main sulfide mineral at Ilovica is chalcopyrite followed by pyrite and secondary copper sulfides such as chalcocite, covellite, and bornite. Molybdenite, galena, and sphalerite are present in minor amounts, and occasional traces of sulfosalt minerals such as tetrahedrite-tennantite and tellurides of gold and silver are observed. High-temperature oxide mineralization, such as magnetite, dominates at depth associated with pyrrhotite and chalcopyrrhotite in what is interpreted as the core of the system. A variety of iron hydroxide group minerals are largely developed within the oxidation and cementation zones. Very occasionally gold nuggets are observed at the base of the oxidation zone.

Field Mapping, Rock Chip Sampling, and Soil Geochemistry Survey

Detailed geological mapping was completed on 1:2000 and 1:5000 scales and comprised observations with respect to petrology, style of alteration, and mineralization. Rock chip samples were collected from the outcrops which were identified as having potential to host mineralization.

In total, three phases of soil sampling have been undertaken on the property, resulting in a total of 540 sampling points arranged on a 100 m × 100 m grid. The total area covered by the soil geochemistry sampling was approximately 5000 m^2. The soil sampling targeted the subsoil horizon, which is generally at a depth of 20–30 cm (the «B» horizon of the soil profile), as this unit generally contains the accumulated minerals. The soil surveys were completed by initially removing the humus topsoil layer with a spade, before taking a 2–3 kg sample of the subsoil. The remainder of the soil was restored to the sampling location and rehabilitation of disturbed areas was performed.

Results of soil sampling over the property indicate significant copper anomalies (>200 ppm copper) to the northwest, southwest, and south of the mineralized intrusive (Fig. 3.78). These anomalies are believed to represent down slope dispersion of the copper from the central area of mineralization. In contrast, significant gold (>0.10 ppm) and to a lesser extent molybdenum (>20 ppm) show less down slope dispersion and more accurately delineate the underlying mineralization.

Geophysical Surveys

A total magnetic intensity survey was carried out and 24 east-west lines spaced 100 m apart were surveyed with readings taken every 10 m. The aim of the survey was to outline the lateral and vertical extension of stockwork zones with secondary magnetite enrichment intersected in several drillholes. Magnetic susceptibility measurements were taken at an average interval of about 10 cm on core from these holes using an electromagnetic inductance bridge. A high amplitude magnetic anomaly was outlined; the magnetic susceptibility measurements demonstrated that the only magnetic rocks in the area are the secondary magnetite enrichment stockwork zones that are the source of the magnetic anomaly. The magnetic models indicated that the magnetic stockwork zone trends north-northeast along an 800 m strike length and is approximately 300 m wide, though inherent ambiguities in the interpretation process may have underestimated the width of the body.

A high-resolution pole-dipole array survey was carried out using dipole lengths of 300 and 150 m and n spacings of 0.5, 1, 1.5, 2, 2.5, 3, 3.5, 4, 4.5, 5, and 5.5 for the array with dipole length of 300 m and $n = 1, 2, 3, 4, 5, 6,$ and 7 for the 150 m dipole length. The IP or resistivity survey identified a number of intense IP anomalies, interpreted to be related to sulfide and magnetite mineralization previously intersected in drillholes (Fig. 3.79). The resistivity

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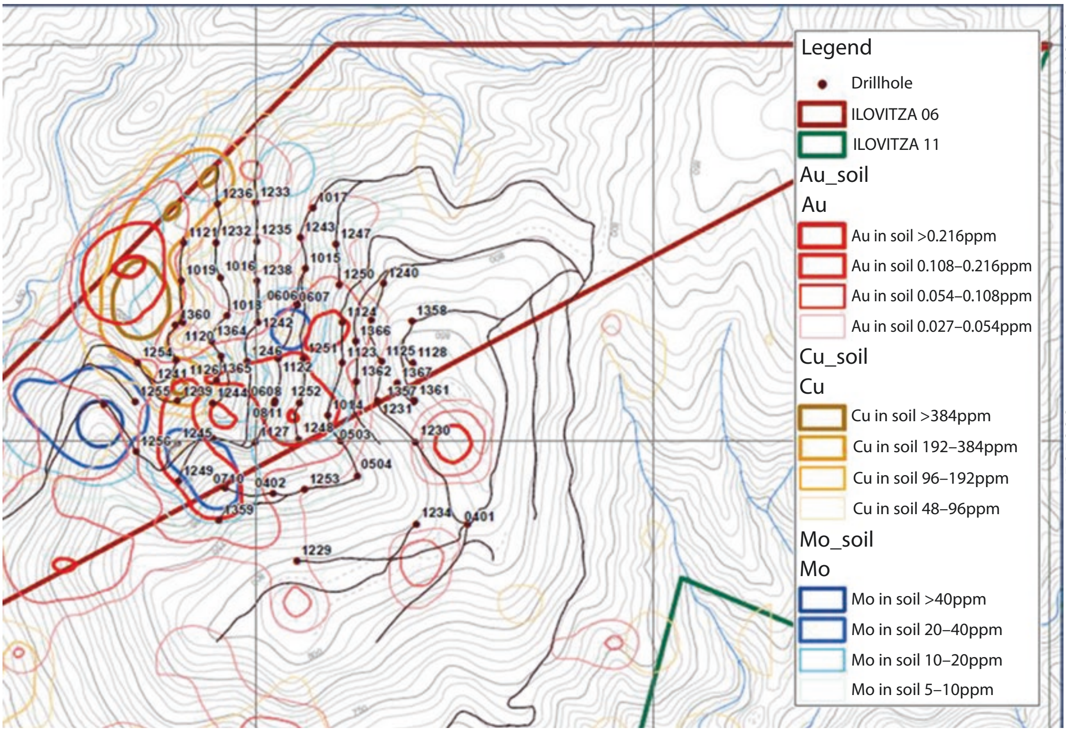


Fig. 3.78 Excerpt of the soil geochemistry anomalies map (Illustration courtesy of Euromax Resources)

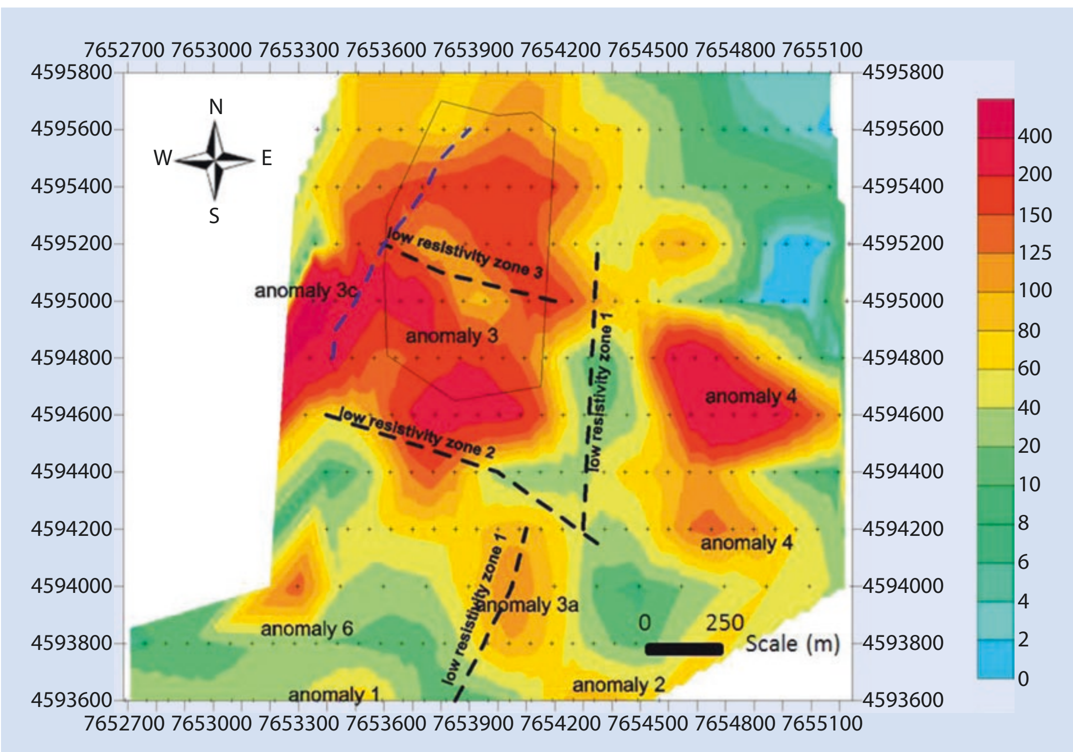


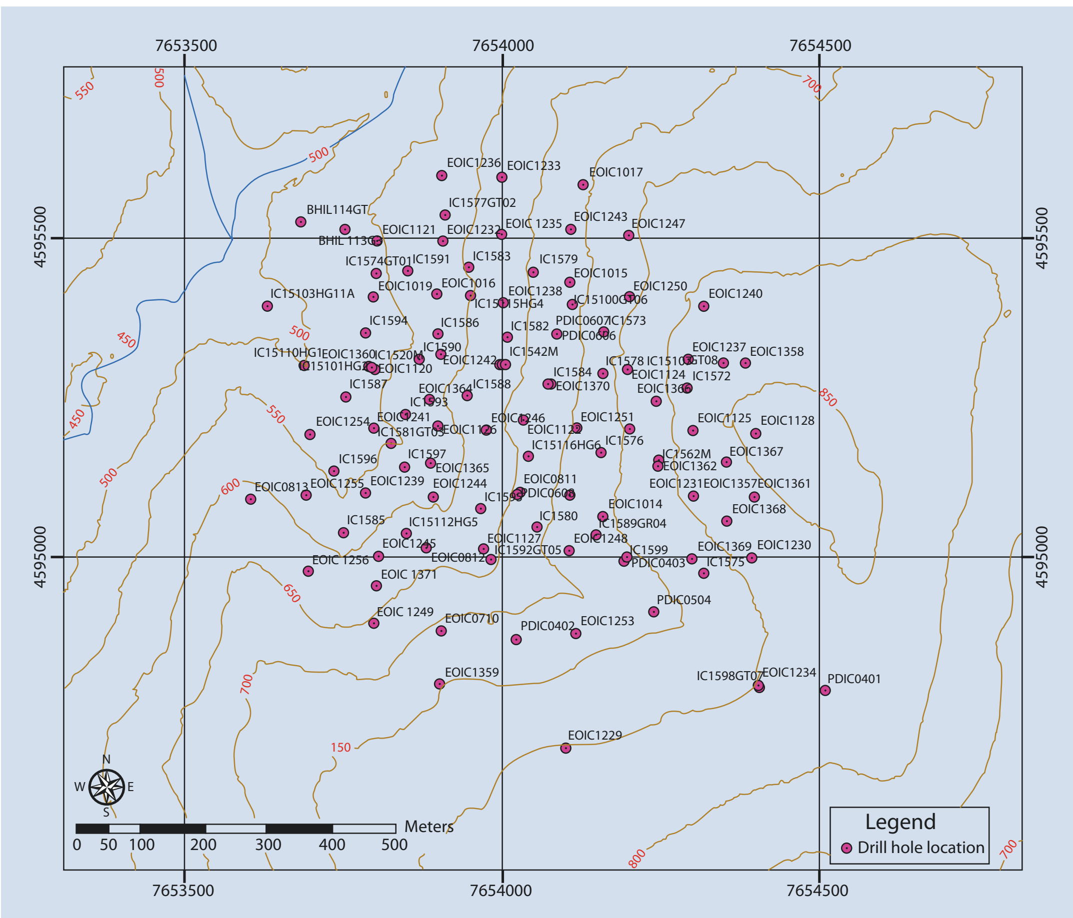
Fig. 3.79 2D IP Inversion model on level 350 m from surface (Illustration courtesy of Euromax Resources)

models revealed the presence of linear, almost vertical low resistivity features, interpreted as fault zones. The most prominent IP anomaly coincided spatially with the magnetic stockwork zone defined previously by the magnetic survey and tested by several drillholes. The high IP intervals correlated with high total sulfide values of up to 3–5%, though while the copper mineralization in drillholes coincides with high sulfide concentrations, it was not possible to distinguish between anomalies related to a barren pyrite halo and IP anomalies associated with porphyry copper mineralization.

Several IP anomalies form a discontinuous annular zone around the interpreted core of the system, probably related to the pyrite halo. The resistivity model indicates the presence of near to horizontal low resistivity layers to the west of the core of the system interpreted to reflect the presence of intensive stockwork zones with copper mineralization. A further observation is that the area of low resistivity correlates with low-grade copper and gold in the Ilovica block model, and in fact, grade appears to increase as the higher resistivity zone is intersected.

▪▪ Drilling

A total of 130 holes have been drilled over 10 campaigns (42,032 m): 20 were drilled for geotechnical investigation, 15 were carried out for hydrogeological investigation, and 95 were drilled for mineral resource determination. The drillholes are generally vertical or steeply dipping, with 95 of the drillholes being vertical and the remainder being between 55° and 75°. The drill locations are illustrated in ◘ Fig. 3.80. All of the holes were drilled using rotary diamond coring techniques. Drillholes were collared with PQ diameter (85 mm core) and then advanced with HQ (61.1 mm core) and then occasionally NQ (45.1 mm core) diameters. A wireline system was



◘ **Fig. 3.80** Drillhole locations (Illustration courtesy of Euromax Resources)

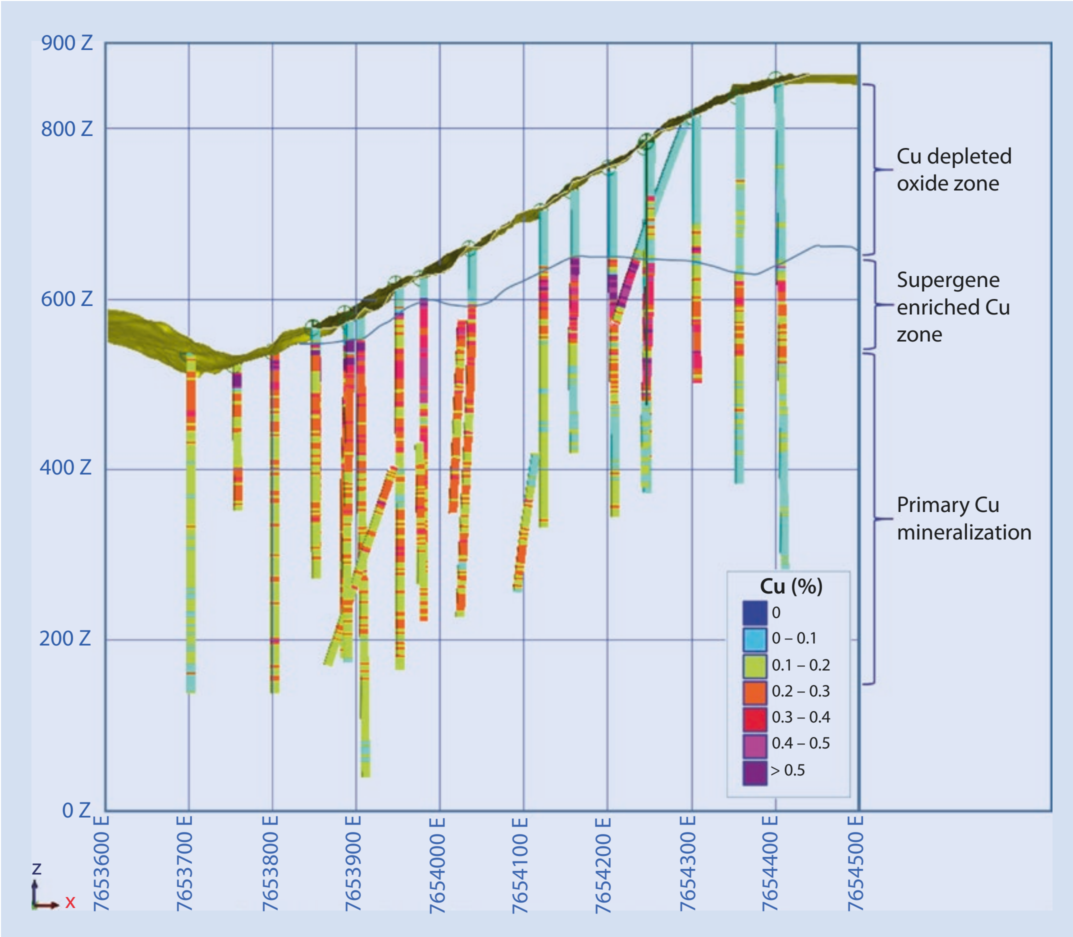


Fig. 3.81 Typical section with copper assays (%) (Illustration courtesy of Euromax Resources)

used to hoist the core tube to surface to allow the drill core to be extracted.

Logging included observations relating to lithology, alteration, mineralization, structure, recovery, and rock quality designation (RQD). Drill core recovery is very good, generally greater than 95%, throughout the deposit. Within the oxide zone, the core is general highly fractured, and as such the RQD is low; however the overall core recoveries remain high. Generally half of the core samples were taken and processed for analysis. Where density samples were taken, one quarter was collected for density determination and another quarter was taken for assaying. Downhole surveys were completed using a digital survey instrument (JKH-R magnetic single shot inclinometer) with readings taken every 50 m. Generally the drillholes show very low deviation from the planned hole paths; deeper holes show up to five degrees variance from design for both dip and azimuth. The cross section presented in Fig. 3.81 illustrates the interpretation of the drilling results in relation to copper depletion in the oxide materials and supergene enrichment beneath. The gold assays show a similar but less pronounced distribution.

Gold Springs Gold Project Exploration: Courtesy of TriMetals Mining Inc.

The Gold Springs gold project is an advanced exploration stage gold project located along the Nevada-Utah border in the USA. The project is situated in the southeastern portion of the basin and range physiographic province, which is characterized by northerly trending mountain ranges with closed internal drainage basins that resulted from extensional tectonism and associated volcanism during the tertiary period. The Gold Springs

Fig. 3.82 Stockwork veining (Image courtesy of TriMetals Mining Inc.)

project lies within the Indian Peak volcanic field, which is a broad tertiary volcanic field that straddles the Utah-Nevada border and contains several nested, collapsed calderas and resurgent dome features that formed as part of a major Oligocene-Miocene «ignimbrite flare-up cycle.» The oldest rocks in the region consist of Proterozoic through lower Mesozoic sedimentary sequence that became folded and thrust-faulted eastward during the Cretaceous Sevier orogeny and were subsequently overlain by tertiary sedimentary deposits.

Gold mineralization at Gold Springs is hosted by complex sheeted veins, breccias, and stockwork vein systems (Fig. 3.82) that are laterally extensive and locally form resistant ledges and ribs that protrude up to 10 m above the surrounding ground surface and surrounding areas of mineralized wall rock. The veins contain quartz, adularia, and bladed calcite with minor sulfides (<2%) and represent a low sulfidation, epithermal gold-silver vein system. Gold and silver mineralization are hosted in quartz and quartz-calcite veins, breccias, and stockwork/sheeted vein zones surrounding the main vein systems. True thickness of the mineralized zones reaches up to 150 m wide with the strike length of the vein systems extending up to several kilometers.

Sampling

The work program collected 2409 rock chip samples, 2964 soil samples, and 323 stream sediment samples. The majority of the samples were collected on a reconnaissance basis from both outcrop and float material. Sampling was also conducted within the target areas where outcropping mineralization was sampled perpendicular to structural trends where possible. Grab samples were collected to help define background geochemical levels within the various rock units and to evaluate metallic ion distribution and chemical zonation in areas of new exploration. Select samples were also collected from mine dumps and vein exposures to determine if there were any specific geochemical signatures and to characterize the ability of the system to contain high-grade gold values.

Rock chip results ranged from <5 ppb gold to a high of 145.68 g/t gold. Silver shows some correlation with gold and values ranged from <0.1 ppm to a high of 252.9 ppm. Gold geochemistry from the soil samples ranged from <0.5 ppb to a high of 1.3 g/t while silver ranges from <0.1 to 11.6 ppm. Results from the stream sediment sampling show a variation in gold values from a low of <5 ppb to a high of 1.28 g/t. Preliminary analysis of some of the downhole geochemical data suggests that there are at least two different signatures for the various target areas. Moderately anomalous arsenic and local molybdenum values are associated with gold mineralization with a surrounding zone that shows a relative depletion in calcium, potassium, and sodium.

Then, a detailed follow-up sampling and mapping were conducted on several of the target areas. This work included detailed structural analysis and channel sampling as well as detailed vein sampling in the main trenches. In addition, a series of channel sample lines were completed over the exposed vein zones. These channel sample lines generally consist of a series of 2 m long, continuous chip-channel samples across outcropping exposures of the various vein-stockwork zones.

Geophysics

A 470 line-km ZTEM and aeromagnetic helicopter survey was completed. Previous ground surveys revealed a positive correlation between the known epithermal gold systems and buried subvertically dipping high resistivity features. The ZTEM results correlated very well with known geology, in particular the presence of all the known epithermal centers. The helicopter-borne geophysical survey in Gold Springs project included a Z-axis Tipper electromagnetic (ZTEM) system and a cesium magnetometer. Ancillary equipment included a GPS navigation system and a radar altimeter. The survey was flown in an east to west (N 90° E azimuth) direction, with a flight line spacing of 200 m. Tie lines were flown in a north to south (N 0° E azimuth) direction, with a flight line spacing of 1900 m.

TD (total divergence) imaging converts the ZTEM tipper crossover data into peak responses which assists their interpretation in plan. TD low (conductive) areas signify areas with sulfides or possibly conductive clays, and the TD high (resistive) areas represent resistive rocks which show an excellent correlation with known gold occurrences as would be expected in the low sulfidation environment. Subsequent to the 2-D inversion, a 3-D inversion of both the ZTEM conductivity data and the magnetics was carried out. The resulting models for the ZTEM and magnetic inversions provided a 3-D conductivity model of the earth that honors the ZTEM and magnetic data to a specified level of fit. The modeling correctly considered 3-D topographic effects which can significantly influence the data. The inversion modeling was unconstrained by geologic and physical property information. The primary outcome of these studies was the development of a clear correlation with the location of surface gold mineralization and gold intersected in drillholes particularly with the margins of the high resistivity features. This correlation can be seen in the «depth-slice» presentation of the data (Fig. 3.83). Where the high resistivity is shallow, a strong correlation between the margins of the high resistivity and gold intersected in drillholes exists. Where the high resistivity is deeper, gold mineralization is found both at the margins and over the top of the resistivity features. This correlation is interpreted as relating to the heat source for the «hot spring» style mineralization seen at Gold Springs.

Drilling

The last exploration work in 2014 focused on completing a 38-hole RC and 4-hole core drill program. The four core holes were completed to collect material for metallurgical testing and to start to collect geotechnical data for rock quality designation (RQD), fracture analysis, and lithologic and alteration data. On the other hand, downhole surveys are conducted using a gyro deviation survey instrument at or near the termination of the hole. These surveys provide detailed downhole data on the azimuth and dip of the hole over the length of the hole.

The 2012–2014 drilling programs were conducted by wet reverse circulation drilling method. Emphasis was placed on quality control and the proper handling and numbering of all samples. The reverse circulation drill cuttings are collected as they come out of the drillhole from an industry standard rotary wet splitter provided by the drilling company, which delivers the material to three collection points. Samples are collected on 1.52 m intervals. Every 20 samples, standards and blanks are inserted into the numbering sequence of the drill cuttings. The material from the second sample point is retained as a duplicate sample for future testing if needed. The material from the third sample point is geologically logged on site and put into chip trays that are labeled with sample numbers and footage intervals from which the sample was taken.

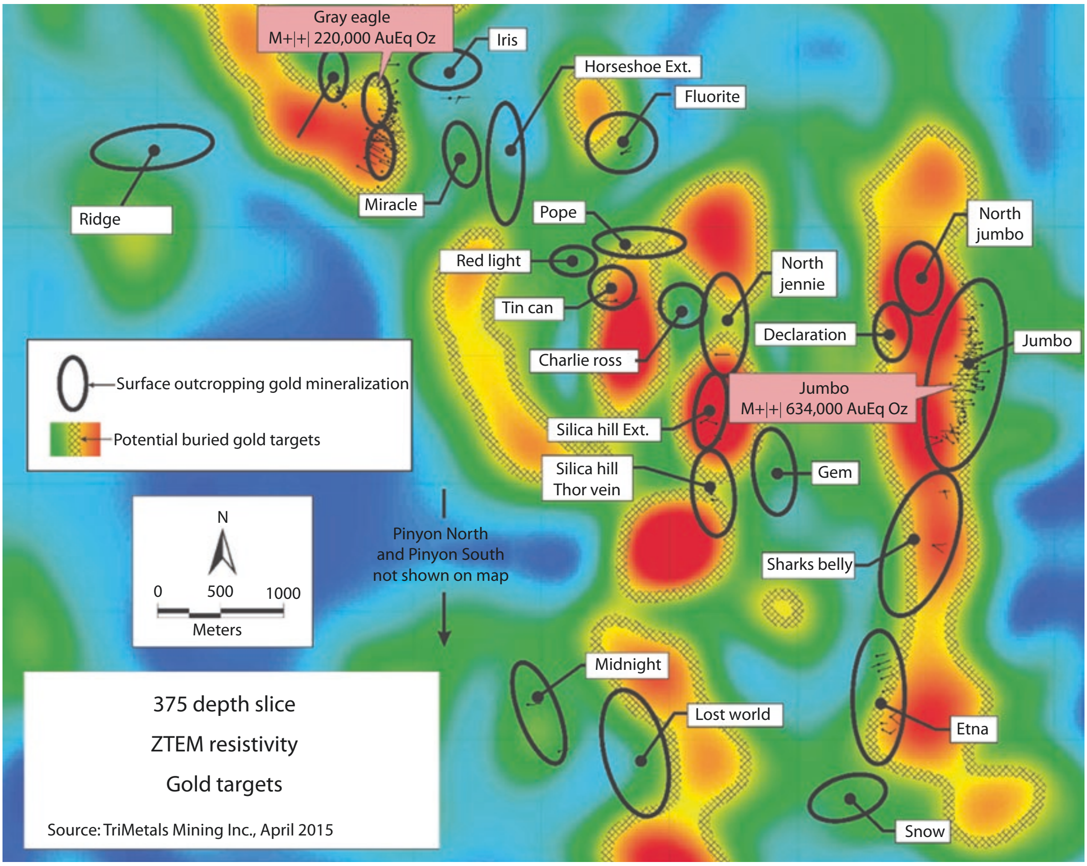


Fig. 3.83 ZTEM high resistivity and correlation with outcropping gold-bearing rocks in the Gold Springs project area (Illustration courtesy of TriMetals Mining Inc.).

3.6 Questions

Short Questions

- What is mineral exploration?
- Define the concepts of «juniors» and «majors» mining companies.
- What greenfield and brownfield exploration programs mean? Explain the advantages and disadvantages of both.
- What are the main mineral resource exploration stages? Explain briefly each stage.
- What is a mineral deposit model?
- What is the Landsat program?
- Explain the differences among diamagnetic, paramagnetic, and ferromagnetic minerals.
- What are the airborne geophysical surveys? List the main types of measurements carried out in these methods.
- Explain the importance of borehole geophysical logging.
- Explain the differences between primary and secondary halos in geochemical exploration.
- List the main types of multivariate statistical methods used in geochemical interpretation.
- Bring out the main disadvantage of reverse circulation drilling.
- What is the wireline system in diamond core drilling?
- Define the concept of borehole deviation. What is the difference between deviation and deflection?
- Explain the parameter RQD in logging and the method of calculation.

Long Questions

- Explain the three-key-factor process in selection of drilling methods.
- Describe the electrical methods used in mineral exploration.

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Mineral Resource Evaluation

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Summary

Mineral resource evaluation should provide a basis on which economic decisions can be taken. At least, four aspects can be identified if a mining project is evaluated, technical, economic/financial, social, and political; this chapter introduces the first two. The technical aspects include all matters related to the geological setting of the deposit, characteristics of the mineralization (grade, tonnage), and technology that determines the production system. A general introduction to geostatistics from the conceptual viewpoint is provided as well as the main classical methods used in mineral deposit evaluation. As regards the economic/financial aspects, they cover the economic inputs and outputs in the project and the amount, type, and cost of capital forthcoming for a project. On this subject, net present value, internal rate of return, payback period, as well as risk analysis are included. Some case studies are presented to illustrate the main methods used in mineral resource evaluation.

4.1 Introduction

Mineral resource evaluation should provide a basis on which economic decisions can be taken. There are several steps needed to ensure a logical progression of a mining project from the initial scattered prospection data to final resource/reserve valuation that meets the needs of potential investors and bankers. Thus, a mine will come into existence if it generates and sells something valuable (Scott and Whateley 2006). At least, four aspects can be identified if a mining project is evaluated: technical, economic/financial, social, and political.

The technical aspects include all matters related to the geological setting of the deposit, characteristics of the mineralization (grade, tonnage), and technology to establish the system of production. The economic and financial aspects cover the economic inputs and outputs in the project and the amount, type, and cost of capital forthcoming for a project. The latter will be defined partially based on financial environment at the moment the investing is carried out. Regarding the social aspects, they include the social costs and benefits originated in a mining project. The infrastructure development, the use of local work and commodity resources can afford positive contributions to society, but conversely mines generate tailings and effluents that can produce negative impact on the environment. Finally, political aspects mean the mineral, fiscal, foreign exchange, and employment policies of the country governments where the deposit is situated. They are especially noticeable to governments contributing in mineral projects. In this chapter, technical and financial aspects will be described, whereas everything else is left to more specific texts.

The relative importance of each type of evaluation of a mineral project at any point in time depends on the stage of development. Thus, target selection or drilling draw relies mainly on the geological sciences, while the later stage (feasibility study) depends more on the engineering sciences and economics. The socioeconomic evaluation is carried out preferentially where development of a mineral deposit is considered. Moreover, rather than being independent of one another, these types of evaluation are interrelated and they are often carried out in parallel. The results of the technical evaluation serve as important input to the economic evaluation, and together, the technical and economic evaluations serve as a starting point for the socioeconomic evaluation. In addition, these evaluations are constantly revised in the light of new information.

4.2 Sampling

On any deposit delimitation program, sampling is an essential step to establish the limits, volume, mass, and grade of the mineral deposit. Thus, the main goal of sampling is to generate values about the mineralization (e.g., assays of metal grades) that are the fundamental information to be utilized in carrying out resource and/or reserve estimations. Therefore, sampling of an ore deposit is a process of approximation, and the objective is to arrive at an average sample value that closely depicts the true average value for the ore body (Readdy et al. 1982). Sampling is also important to study several geotechnical properties of the overburden and the host rock of the mineralization during the prospecting stage of the mining project. These include properties (strength or degree of weathering, among others) that are essential in

■ **Fig. 4.1** Fully automated sample plant-taking samples; the process is completely hands off and uses robotics to perform the analysis (Image courtesy of Anglo American plc.)

designing a mine (e.g., size of the underground chambers or different pit slopes).

Sampling determines the day-to-day of any operation in the mine. Since inappropriate sampling procedure can originate incorrect estimation of present production and future potential, the mine department commissioned of resource/reserve estimations and mine sampling should be monitored by qualified and experienced professionals with technical backgrounds qualifying them to obtain precise data (Tapp 1998). In sampling an ore body to estimate grade, the geologist is mainly concerned with the reliability of his estimate as measured by its accuracy and precision. Accuracy, the close correspondence of an estimate to the «true» value, is achieved by obtaining unbiased results through appropriate sampling, sample preparation, assaying, and data analysis (■ Fig. 4.1). To avoid bias, the geologist must control issues such as salting (e.g., Bre-X affaire; ■ Box 1.4) or nonrepresentative samples. On the other hand, precision is the closeness of a single estimate obtained by sampling and ore body or other geologic entity to the estimates that would be obtained by repeated sampling of the ore body.

4.2.1 Significance of the Sampling Process

The sampling of metalliferous and industrial mineral deposits is undertaken for a variety of reasons and at various stages in their evaluation and exploitation. During the exploration phase, the sampling is largely confined to the analysis of drill cuttings or cores and is aimed at the evaluation of individual, often well-spaced, intersections of the deposit. During the exploitation phase, sampling is also used to define assay hanging walls and footwalls together with the grade over mineable thicknesses. Sampling is much more intense in this situation and is undertaken to allow the assignment of overall weighted grades to individual ore blocks or stopes. Also at this stage, sampling will be used to extend existing reserves and attempt to prove new ore zones accessible from existing developments (Annels 1991). Perhaps one of the most important applications of sampling during the exploitation phase is in grade control (■ Fig. 4.2) (e.g., bench grades in an open-pit mine) since it determines the boundaries of mineralization and waste (see ▶ Chap. 5).

Fig. 4.2 Taking samples for grade control (Image courtesy of Alicia Bermejo)

It is important to remember that to take a sample means that the information obtained from the analytical data of the sample will be finally utilized to someone who will use the information contained in the analytical result to make a decision. These decisions can involve immense capital engagements to open or close a mine or marginal process costs that include the decision if a batch of mineralized rock must be sent to the beneficiation plant or to the tailings dump (Minnitt 2007). For this reasons, the process of sampling is among the most essential activities in mining operations because the possibility always exists for large occult costs to accumulate in mineral development due to sampling errors. These hidden costs arise due to misunderstanding of the principal factors that affect the size of sampling errors (e.g., amount of the sample, the consequences of dividing a sample to reduce the amount, or the notorious impact of the particle size in the mineralization). Items such as sample procedure, sample reduction, assaying methods, and obviously geological data collecting and modeling are critical for a high-quality estimation of the resources and/or reserves. This is because many times data collection techniques are not of adequate quality to correctly define a mineral deposit.

All the processes involved in sampling must be checked continuous and appropriately. Obviously, there will constantly be a difference among the content of the lot, the sample obtained, and the sample for assay since the comparatively large amount of a sample is reduced to a small subsample of some grams that are needed for the final chemical analysis. This discrepancy is termed the sample error. Attention to the matters cited above reduces the errors and improves the quality, which is essential for interpretation of geological data and modeling and consequently the quality of resources/reserve estimation. The so-called sampling due diligence, which carries out an authentic geological resource evaluation, requires a validation process of many components, including, among others, (a) adequacy of samples, (b) sample representation, (c) accuracy of laboratory assays, (d) insertion of blank and standards, and (e) quality assurance and quality control protocols; these are the currently famous QA/QC (Box 4.1: QA/QC in Coringa Gold Project).

QA/QC includes duplicate analysis and standard analysis. The precision of sampling and analytical data are estimated by analyzing twice the same sample utilizing the same methodology (duplicates), being the variance between the two data an estimation of their precision. Precision is affected, as aforementioned, by mineralogical factors such as grain size and distribution but also by errors in the sample preparation and analysis processes. Regarding standard samples (or reference materials), they are samples with a known grade and variability. These are commonly used to assess analytical accuracy and bias by comparing the assay results against the expected grade of the standard. In these sense, managers and consultants always insist that standard and duplicate samples are invaluable items to measure the accuracy and precision of commercial analytical laboratories. Moreover, they can ensure there can be a realistic confidence in the data by correctly utilizing these measurements of data quality to quantify the future risk of the mining project.

Box 4.1

QA/QC in Coringa Gold Project: Courtesy of Anfield Gold Corp.

The QA/QC program included the insertion of two standards, two duplicates, and one blank every 42 samples. Table 4.1 shows a summary list of control samples. Company created sample «duplicates» on site utilizing a method that creates a disparity in the duplicates. The procedure employed was to place a one-half core split in a plastic bag and to then crush the core with a hammer. The resulting crushed material was mechanically hand-mixed and then divided into equal portions for shipment. The use of this procedure introduces a bias in the sample since they are divided at a very coarse particle size. The bias is compounded by the fact that the majority of the mineralization is represented in discrete veins which are typically represented by a fraction of the particles at the large particle size. Due to this procedure, comparison of the duplicates created by Magellan is poor. In contrast, the laboratory preparation duplicates compare well with each other showing the sampling program was unbiased. Figure 4.3a shows a comparison graph of the laboratory duplicates, and Fig. 4.3b shows a comparison of the laboratory repeat assays which also compare well. Regarding the assay results of the blanks used in the QA/QC program, a total of 26 blanks returned detectable gold values. In all cases, the assay result was below 100 ppb Au. These discrepancies do not greatly affect the resource calculation since the average grade of the resource is 3.92 g/t Au or 47 times the highest gold value returned for a blank sample. Thus, analysis of the results from the blank insertion indicates that there was no contamination apparent.

In addition to blanks and duplicates, standards were used to check the accuracy of the assay results. A total of 12 different standards were used. The gold values of the assay standards cover the variation of the average gold grade for the resource, from 0.081 ppm Au to 14.89 ppm Au. The 14.890 ppm Au standard shows that all of the 12 samples report above the certified standard value which may indicate that the lab was overreporting the gold grade. The certificate of analysis shows that gold grade for the 14.890 ppm Au standard was determined by laboratory consensus which represented the average of eight-subsample sets analyzed by 11 different laboratories. It indicates acceptable accuracy performance of the standard despite the fact that all samples return assays higher than the certified value. Overall, the QA/QC program for sample assays indicates acceptable performance of all standards and blanks with only a few minor discrepancies.

Table 4.1 Summary QA/QC program

Type sample	Description	Number of samples
	Total number of samples	9139
	Number of control samples	1922 (21.03%)
Sampling	Duplicates	315 (3.44%)
	Standards	421 (4.60%)
	Blanks	212 (2.32%)
Assaying	Lab – duplicates	353 (3.86%)
	Lab – repetition	275 (3.00%)
	Second lab checking	346 (3.78%)

Data courtesy of Anfield Gold Corp.

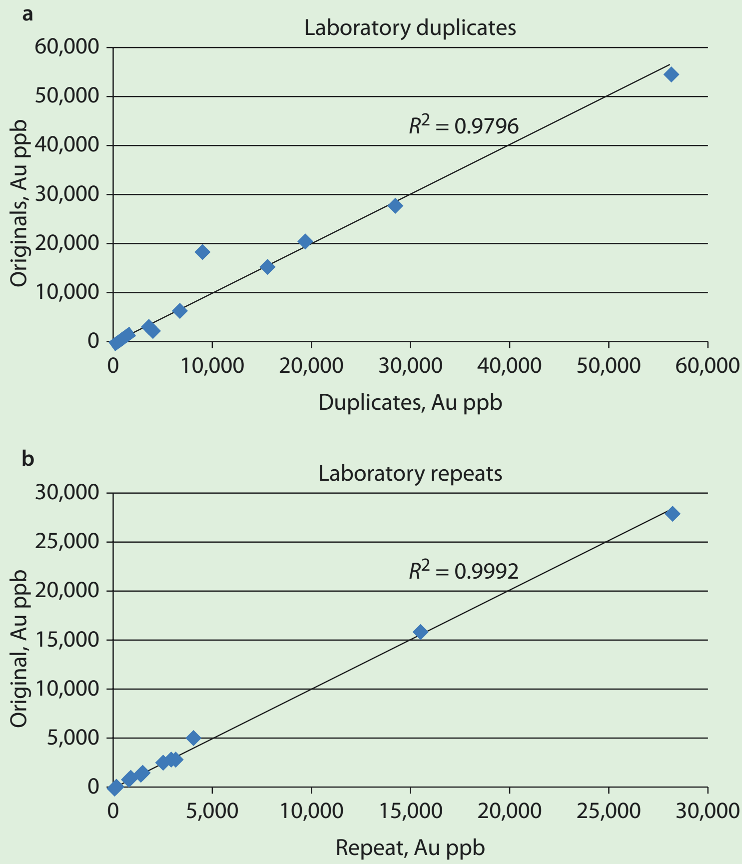


Fig. 4.3 Laboratory duplicate comparison **a** and laboratory repeat comparison **b** (Illustration courtesy of Anfield Gold Corp.)

4.2.2 Definition of Sample

From a practical viewpoint, it is impossible to gather all the components of a population for study unless the population itself is very small. For this reason, it is essential to resort what is commonly known as «sample.» There are many definitions of sampling, but the concept is quite elementary. For example, a sample is «a representative part or a single item from a larger whole, being drawn for the purpose of inspection or shown as evidence of quality,» and it is «part of a statistical population whose properties (e.g. physical and chemical) are studied to gain information about the whole» (Barnes 1980). Another definition of sampling is «the operation of removing a part convenient in size for testing, from a whole which is of much greater bulk, in such a way that the proportion and distribution of the quality to be tested (e.g. specific gravity, metal content, recoverability) are the same in both the whole and the part removed (sample)» (Taggart 1945).

Both definitions are very similar being essential that the sample be representative (Fig. 4.4). It is the key to a successful process of sampling. If the samples are not representative of the deposit, the rest of the evaluation is useless. There is no point in geological interpretation and modeling is carried out correctly if the initial data are wrong. Thus, the accuracy of a mineral resource or reserve calculation depends on the quality of the data gathering and handling processes used (Erickson and Padgett 2011). Large amount of sampling is carried out in the mineral industry, but little attention is given to ensure representative sampling. The responsibility for sampling is often tasked to people who do not take into account the significance of sampling, with cost being the main factor rather than the representative of the sample. The quality of the subsequent analysis is

Fig. 4.4 The sample must be representative

undermined, and mineral companies are exposed to enormous potential financial losses.

The successive steps of sampling must be therefore tested continuously, although it is important to bear in mind that the condition of representativeness for the sample obtained from a whole is never fulfilled where heterogeneous materials are sampled, unless the sample includes all the mineralization. Thus, «an orebody is a mixture of minerals in proportions that vary in different parts of the mass. As a consequence the proportion of contained metals also varies from place to place. Therefore, a single sample taken in any particular place would not contain the same proportion of metals as does the orebody as a whole except by a highly improbable coincidence. The probable error, which would be very large if only one sample were taken, decreases with the number of samples, but it never disappears completely unless the samples are so numerous and so large that their aggregate is equal to the orebody itself, in which case the orebody would be completely used up in the process of sampling» (McKinstry 1948).

Random and systematic errors involved in the collection, preparation, analysis, and evaluation of samples must be recognized and accounted for. In fact, this is not a problem but rather an incentive. In this sense, Sarma (2009) affirmed that a good sample design must:

1. result in a really representative unit;
2. lead to exclusively a small error;
3. be cost-efficient;
4. be one that monitors systematic bias; and
5. the results of the sample study can be utilized for the population with a fair degree of confidence.

The samples must also be representative from a spatial viewpoint, which means that the spatial coverage of the deposit is adequate. Thus, the samples can be taken roughly in a regular or quasi-regular sampling grid (Fig. 4.5), representing each sample a similar volume of mass in the valuable mineralization. Furthermore, the most important norm for an accurate sampling is that all components of the mineralization or other raw material must have the same probability of being sampled and constituting part of the final sample for the assay. The logic of sampling is to collect a minimal mass (grams, kilograms, or tons) that equals a certain parameter (e.g., gold content) of a much larger mass (hundreds or thousands of tons) (Pohl 2011). It is necessary to take into account that finally only a tiny portion of the mineral deposit is collected and that often less than one-millionth of the total mass of a deposit is being drilled; it is quite easy to obtain this datum estimating the volume of drillholes, the volume of an entire deposit, and dividing both data.

The type and number of samples collected depend on a range of factors which include (1) the type of mineral deposit and the distribution

Fig. 4.5 Sampling grid in blasting

and grain size of the valuable phase; (2) the stage of the evaluation procedure; (3) whether direct accessibility exists to the mineralization; (4) the ease of collection, which is related to the nature and condition of the host rock; and (5) the cost of collection, funds available, and the value of the ore (Annels 1991). It is clearly incorrect to take over that many samples remove any errors in the sampling procedure. To obtain unbiased samples, the location of the sample in relation to the mineralization and waste is just as important. In fact, the accuracy of a sampling procedure is only known where all the mineralization is mined and later milled and processed.

Obviously, the cost of intense sampling of a low-grade or low-value deposit (e.g., aggregates for construction) can be prohibitive. For instance, the mode of occurrence and morphology of a mineral deposit has considerable impact on the type and density of sampling and on the amount of material required. Indeed, sampling of vein deposits, where many veins are narrow, is quite different than sampling of stratiform deposits where mineralization tends to be thick (e.g., up to 30 m). Thus, a mineral deposit classification with sampling as one of the main goals has been proposed taking into account the geometry, the grade distribution, and the coefficient of variation (Table 4.2; Carras 1987).

Table 4.2 Mineral deposit classification based on geometry, grade distribution, and coefficient of variation (Carras 1987)

Type A – low coefficient of variation
Type A1 – simple geometry and simple grade distribution
Examples: Coal, iron, bauxite, lateritic nickel and stratabound (stratiform?) copper.
Type A2 – simple geometry and complex grade distribution
Examples: disseminated copper, gold stock-works, Witwatersrand gold
Type B – complex geometry and simple grade distribution with a low coefficient of variation
Examples: Basemetal deposits, e.g. skarn copper deposits (Craigmont, BC)
Type C – complex geometry and complex grade distribution with a high coefficient of variation
Examples: Archaean gold (e.g. Kalgoorlie and Canada)

4.2.3 Steps in Sampling

To acquire accurate analytical data for resource estimation, it is indispensable to carry out a correct process of collecting samples (methodology, sampling pattern, and sample size), including a study of the ore with particular attention to the particle size distribution and the composition of the particles in each size class. Samples of several kilograms or even some tons are later cut to several grams, the so-called assay portion, which are further assayed for valuable elements; theoretically, this final aliquot must still replicate targeted properties of the original large mass. The reduction in weight is around 1,000 times with a kilogram sample and 1,000,000 with a sample mass of one ton. This process obviously involves errors, and Gy (1992) established a relationship between sample particle size, mass, and sampling error. Analytical errors are ascribed to laboratories and commonly take place from the selection of the portion for analysis. As aforementioned, these errors must be considered with and external control by submitting to the laboratory duplicate samples and reference materials of similar composition of the unknown samples.

To reduce errors in sampling, one solution is to divide the mineral deposit and the mineralization into distinct parts (a previous step in the sampling process). Thus, to take samples of the previously defined different types as separate units instead of as only one large sample can minimize natural variation and maintain the sample weight in a minimum. This method is the so-called stratified sampling, and it is very important if the separate types of mineralization need different mineral beneficiation techniques. Regarding the different steps in the sampling process, sampling, sample preparation, analysis, and interpretation in the final stages of exploration and mining are planned and carried out by a staff of geologists, chemists, statisticians, and engineers, who contribute their expertise to the interpretation of the sampling data. The importance of thorough, joint planning and interpretation is obvious because they form the basis for an economic and technical evaluation of the mineral prospect and because of the large financial commitment that the development of a potential ore deposit requires (Gocht et al. 1988).

4.2.4 Sampling Methods

Sampling methods are as different as the mines in which they can be utilized. The most suitable type of sampling and the combination of methods used depend to some extent on the type of deposit being evaluated. For instance, to conduct an unbiased sampling in vein gold deposits presents particular challenges because the features of the mineralization and host rocks are extremely complex. Variance however can be diminished by carrying out a well-planned procedures of sampling as well as careful collection of samples as possible. The mine geologist or engineer devoted to sampling process must select a method of sampling, test in a specific area, and later critically evaluate the results obtained. If these outcomes are sufficiently accurate within the economic limits determined by the mining company, then the methodology can be embraced as a general rule in the project and/or mine.

In general, there are three hand sampling methods: channel, chip, and grab sampling. Other sampling techniques include pitting and trenching or drill-based sampling (diamond drilling and in some cases rotary percussive drilling are the main sampling techniques available to the geologist in the exploration of a mineral deposit). In fact, the most satisfactory method should ensure that the sample properly represents the deposit at the smallest cost. It is very important to bear in mind that whether the samples are collected on surface or underground is not in itself a significant factor, that is, the same process must be assigned to sampling a core drill in surface drilling and in underground drilling.

Channel Sampling

Channel samples (◘ Fig. 4.6) are suited particularly to outcrops, trenches, and underground workings. The method consists of cutting a relatively precise narrow channel of constant depth and width across the exposed width of the mineralization, typically a vein ore. The cut can be either horizontal, vertical, or perpendicular to the dip of the ore. In the case of strongly preferred orientations (e.g., bedding), channels must be guided across the layering. The samples are collected across the full width of the vein, or at some uniform fixed length in wide; in complex veins, any identifiable subdivisions should be sampled

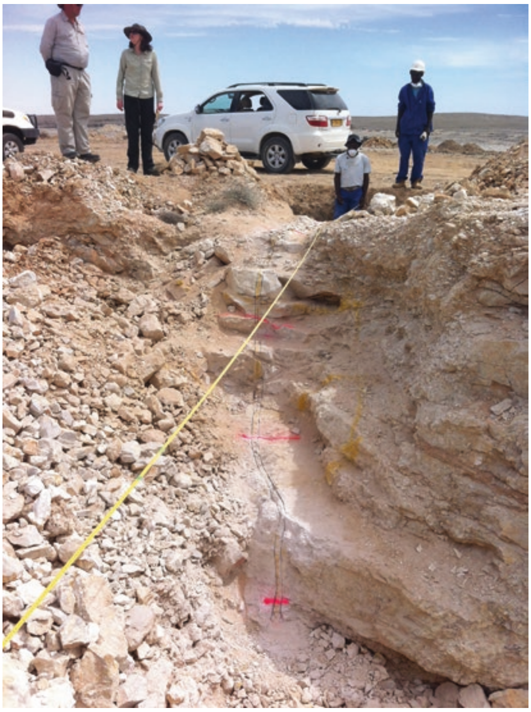


Fig. 4.6 Channel sampling (Image courtesy of Martin Pittuck)

Fig. 4.7 Channel sample obtained with a hammer and chisel (Image courtesy of Martin Pittuck)

separately. In theory, if the channels were continuous, and uniform, the channel sample would be similar to a drill core.

As far as possible, the channel is kept at a uniform width (e.g., 3–10 cm) and depth (e.g., 5 cm), although the spacing and length depend on the inhomogeneity in the distribution of the ore or the amount of material needed for analysis. The channel is best cut at a right angle to the ore zone, but if this is too difficult, the channel can be taken horizontally or vertically. As an example of the procedure, in the Cornish tin mines, a standard practice was to collect channel samples at 8–10 m intervals at the face on every other bench up the dip of the stope. Approximately 2 kg of material was collected to represent a length of channel not exceeding 50 cm (Annels 1991).

Samples are usually collected by hand and can be cut with a hammer and chisel (Fig. 4.7) or an air hammer. The chips are set out on a plastic sheet laid out the floor of the working area, from which it is collected and bagged. Accessibility and rock hardness determine the applicable sampling tools. If the quantity is large, it can be quartered before being placed in the sample bag. In hard rock, it is quite difficult to achieve the ideal channel unless a special mechanical diamond-impregnated disk cutter is used, so that a reasonable approximation is generally accepted to be satisfactory. The working area to be sampled must be cleaned thoroughly employing a wire brush or water, among others. This is done to reduce the potential for contamination of the sample by loose fragments on the face being sampled.

The main problem of channel sampling is related to the presence of soft minerals since they can commonly be broken preferentially. Thus, soft mineralization can be overrepresented in a sample, which imposes a high bias on the grade results. On the opposite, soft gangue minerals can be overrepresented and produce an undervaluation of grades. This problem may be partially resolved by taking large samples or taking separate samples from soft and hard zones, if possible. Channel has commonly a maximum length of 1.5 m and the samples must be divided into smaller parts in longer samples. This subdivision is carried out based on the structures in the mineralization, changes in rock types, or differences in rock hardness. Although channel sampling possibly originates the best method of delimiting and extracting a sample, the process is expensive, laborious, and time-consuming.

Fig. 4.8 Chip sampling (Image courtesy of Gold One Group Limited)

Chip Sampling

Chip sampling (Fig. 4.8) is a modification of channel sampling utilized where the rock is too hard to channel sample economically or where little variation in the mineral content shows that this type of sampling method will provide results comparable to those originated by channel sampling. Chip sampling sometimes is applied as an inexpensive method with the objective to control if the ore is really valuable and allows the implementation of the more expensive channel sampling technique. It is the most common method used for underground grade control sampling. Since the advantage of chip sampling is its high productivity, the method is rapid and easy way to get information about the mineralization, but samples are less representative than in channel sampling. For this reason, this method should not be used for quantitative ore reserve calculations.

Chip samples are taken by chipping over the whole area or a portion of the face, for example using a grid laid out on the face of an exposed outcrop. Where a line is sampled, rock chips are taken over a continuous band across the exposure approximately 15 cm wide using a sharp-pointed hammer or an air pick. This band is usually horizontal and samples are collected over set lengths into a cloth bag, usually 15 cm by 35 cm, and equipped with a tie to seal it. At Sigma Mine, Val d'Or (Canada), rock chips are taken at intervals of 0.25–0.5 m along horizontal lines marked on the face. Each line is spaced at 0.75 m from its neighbor and provides between 3.5 and 5.0 kg of material which is sent for assay (Annels 1991).

A general requirement is to collect small chips of equal size or in some cases coarser lumps at uniform intervals over the sampling band or area. The distance between any two points, horizontally or vertically, must be the same on any one face and can vary with the character of the ore. The recommended number of points depends on the variation of the ore: 12–15 for uniform to highly uniform deposits, 20–25 for nonuniform deposits, and 50–100 if mineralization is extremely uneven (Peters 1978). The possibilities for unintentional or intentional bias due to variable chip sizes and the oversampling of higher-grade patches or zones are high. Effort should be made to keep relatively constant sample volume proportional to the widths of the ore, and care must be taken to collect approximately the same size chips across the zone being sample; chip points should also be as regularly spaced as possible. Often, a composite sample is commonly obtained to establish the average grade of the ore present.

Grab Sampling

Grab sampling is usually performed as the inexpensive and easy option, but it is the least preferred sampling method and consisting of already broken material (Fig. 4.9). The method involves collecting large samples from the stockpile at a face or at a drawpoint or from the trucks or conveyor belts transferring the mineralization from these points. The accuracy of this sampling

Fig. 4.9 Grab sampling

method is frequently in doubt and sampling bias is known to be large. Care must be taken that the sampler is not selective and does not tend to select only large or rich-looking fragments; some correlation usually exists whereby the larger fragments are enriched or depleted in the critical component of value. Impartiality is rather difficult to achieve unless rigorous precautions are taken, and this is one of the disadvantages of the method (Storrar 1987).

However, if the grab sample is composed of enough fragments and if taken over a large enough area, it can sometimes represent the grade of the mineralization in that area. Thus, in disseminated or massive deposits where the ore limits are outside the available site, a composite of several pieces from a freshly blasted face can be the most successful sample. In general, grab sampling is not considered reliable since many independent variables can affect this type of sampling process. For example, if the ores occur in the softer fraction and a proportional amount of the resulting fines are not sampled, the results are clearly erroneous. Because of the lack of significant dimension and the commonly biased collecting procedure, grab sampling can neither be used to volumes estimation nor utilized in mineral deposit evaluation.

It is commonly accepted that the value of a grab sample is only applicable to the aliquot that was assayed. Thus, a grab sample from a stockpile gives information just on the sample itself and is unsuitable for any accounting purposes. The main problem «is that the material in stockpiles or the material loaded into trucks is rarely sufficiently mixed to be representative of the block of ground from which it was drawn; also, material collected will be from the surface of the pile and rarely from its interior» (Annels 1991). Grab sampling «works better in more homogeneous low-nugget effect mineralization types such as some disseminated base metal deposits, while in heterogeneous high-nugget effect types (e.g., gold, especially if coarse gold is present), strong bias is expected» (Dominy 2010). In brief, nugget effect means error.

One of the greatest problems with grab sampling is related to the size of the sample that is needed, being the amount of individual samples ranging between 1 and 5 kg. These few kilograms of sample that are obtained over a pile are therefore commonly inadequate which leads to a large error. In most cases, it is likely that tons of materials are required for each sample. One approach to stockpile sampling is that employed at the gold mines in Val d'Or, Quebec, where the «string and knot» method is used. According to Annels (1991), «the broken ground from each blast at the face is transported to surface and spread over a concrete pad; three of four strings, with knots at 0.5 m intervals, are then placed over the pile at 3 m intervals and, at each knot, a sample is taken and its weight recorded, along with the position of the knot; each sample is assayed and the result

Fig. 4.10 Bulk sample plant where kimberlite samples are being treated (Image courtesy of De Beers)

weighted by the relevant weight to obtain the overall grade.»

Bulk Sampling

Bulk sampling is a usually utilized term to outline the method of the removal of large quantities of ore for the purpose of testing mineral contents. Before taking a decision to develop a mine, an explorer can extract a bulk sample of the material to be mined for further metallurgical or chemical testing and refinement of the proposed mining procedures. Thus, bulk sampling is carried out only in a much evolved exploration if making the decision to mine is required. Bulk samples are also used for developing beneficiation flow sheet and maximizing the recovery efficiency in mineral processing. Moreover, in parallel with the bulk sampling and geological appraisal work, the geomechanical and mining features of the mineral deposit commonly can be studied in more detail.

Extraction of a bulk sample (e.g., 100 tons) commonly involves excavation of a small pit or underground operation. Samples are dispatched for analysis in strong bags or in steel drums. The primary purpose is to collect a representative sample and to reliably determine the grade for comparison with the resource estimate; this aspect is essential for advanced mineral projects with a nugget problem (e.g., gold mineralization). Therefore, an integral part of a bulk sampling program is the verification of the geological interpretation used for a resource estimate, for example, where the grades of diamond drill core or reverse circulation drilling chips are suspect due to poor drilling conditions. A typical bulk sampling and sample preparation protocol relies on several stages of comminution, each followed by mass reduction through splitting (Fig. 4.10). While expensive, bulk sampling provides relatively cheap insurance against a failed mine investment as part of a pre-feasibility or feasibility study. Many minerals and metals, especially industrial minerals, also require testing for the quality of the concentrate or mineral produced. In these cases, large-scale samples of the concentrates or products may be needed by the customer.

Bulk sampling is also typically used in exploration of diamond-bearing kimberlites. The bulk samples are the first stage to establish the economic parameters of the kimberlites with the objective of obtaining information that leads to the decision of a more detailed program of drilling to determine kimberlite size, morphology, geology, and grade distribution. In these deposits, the economic evaluation is usually carried out in four stages, and at the third stage, a limited bulk sampling program (order of 200 tons) must be carried out to provide the grade of the diamond expressed as carats per ton (1 carat = 0.2 g). A bulk sampling procedure in diamond kimberlites (bulk samples typically 50–200 tons) costs usually several USD 100,000 if not several USD millions (Rombouts 2003). If macro-diamonds are present, only a mini-bulk sample is necessary, being obtained either from drill core or localized pit sampling. Typical sample sizes of these mini-bulk samples range from 500 kg to several tons.

Pitting and Trenching

If the soil is thin in a mineralized area, the definition of bedrock mineralization is commonly carried out by the examination and sampling of outcrops. However, in locations of thick cover, it is imperative a sampling program using pitting or trenching (or drilling). In these methods, heavy equipment is utilized to clear surface soil and expose the bedrock. Hereafter, trenches or pits are excavated into the rock to expose ore zones for sampling (◘ Fig. 4.11). Despite their relatively shallow depth, pitting and trenching have several benefits in comparison with drilling such as the comprehensive geological logging that can be delineated and large and undisturbed samples obtained. Pits and trenches can be dug by bulldozer, excavator, or even by hand, being excavators commonly much quicker, inexpensive, and environmentally less harmful than bulldozers.

◘ **Fig. 4.11** Trenching in progress (Image courtesy of Petropavlovsk)

In general, pitting and trenching can often be regarded as special cases of bulk sampling. The advantages of pits and trenches are that they permit the accurate sampling of mineralized horizons and they facilitate the collection of very large samples, which is particularly important in the evaluation of some types of mineral deposits such as diamondiferous or gold deposits. If the terrain is unfavorable for trenching or if greater depth of penetration is required, drilling techniques must be employed. In some cases, the pit can be sunk not including wall support, but correct safety procedures are crucial if there is any possibility of the sides caving or of rocks being moved from the sides (MacDonald 2007).

Pitting is usually employed to test shallow, extensive, flat-lying bodies of mineralization, being buried heavy mineral placers an ideal example. In tropical regions, thick lateritic soil constitutes optimal conditions for pitting, and if the soil is dry, pits to 30 m in depth can be safely extracted. The sinking of 1 m diameter pits through the overburden into weathered bedrock has been a standard practice in Central Africa, where exposure is poor due to the depth of weathering. Circular pits, 5–10 m apart, are sunk to depth of 10–15 m along lines crossing the strike of geochemical anomalies to allow the geologist to cut sampling channels in the pit wall and to identify the bedrock type, structure, and mineralization, if present (Annels 1991). Pitting is a slow, labor-intensive exercise, and the depth of penetration can be limited by a high water table, the presence of gas (CO_2, H_2S), or collapse due to loose friable rubble zones in the soil profile and hard bedrock.

With regard to trenches, they are commonly utilized to expose steep-dipping bedrock buried below shallow overburden, being useful for further channel sampling where bulk sample treatment facilities are not available (◘ Fig. 4.12). Excavated depth of up to 4 m is common in trenches, and they can be cut to expose mineralized bedrock where the overburden thickness is not great (<5 m). Most trenches are less than 3 m deep because of their narrow width (<1 m) and their tendency to collapse.

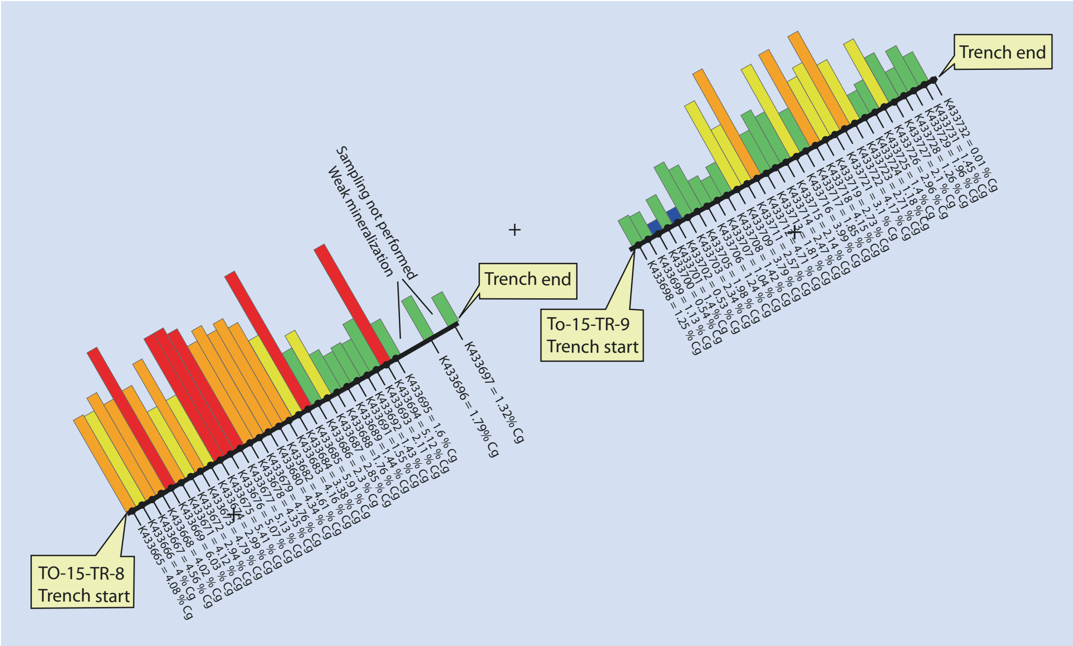


Fig. 4.12 Results of channel sampling in a trench (Illustration courtesy of Nouveau Monde Mining Enterprises Inc.)

Sampling Drillholes

Although expensive, diamond drilling has many advantages over other sampling techniques in that:

1. a continuous sample is obtained through the mineralized zone;
2. constant volume per unit length is maintained; this is very difficult to achieve in both chip and channel sampling;
3. good geological, mineralogical, and geotechnical information can be obtained as well as assay information;
4. problems of contamination are minimal for the core has good clean surfaces; where contamination does exist, the core can be easily cleaned using water, dilute HCI, or industrial solvents; and
5. drilling allows samples to be taken in areas remote from physical access (Annels 1991).

These methods are now utilized routinely, especially for evaluation of large ore where profuse data are needed from what would otherwise be inaccessible parts of a deposit. Mining geologist tends to play only a supervisory role in chip, channel, and grab sampling in a mine, but a direct involvement in the logging and assaying of drill cores will be essential.

Either solid rock core or fragmented or finely ground cuttings are brought to surface by drilling and sampled for assay (Fig. 4.13). Cuttings are either sampled invariably by machine as it reaches the surface or piled up that must be later subsampled. Samples are collected at depth intervals of 1 m or more, depending on the variability of the mineralization. In this sense, the quantity of cuttings from a single drillhole can be huge and the sampling problem is not unimportant (Sinclair and Blackwell 2002). Drill cuttings generally can be generally reduced in mass by riffling to generate samples of handy size for further subsampling and analysis. In this sampling method, it is essential that as much of the mineralization as possible for a specific drilled interval is obtained. The RC drill recovers broken rock ranging from silt size up to angular chips a few centimeters across. The total mass of cuttings produced in each drilled interval is then collected from the cyclone and the material should be routinely weighed, being the common weigh of a 1 m interval of about 25–30 kg.

In diamond drilling, core recovery should be 80% or more for an accurate evaluation, although even at this level of recovery, it is needed to establish whether losses are random or whether specific types of mineralization or gangue are lost

Fig. 4.13 Samples with cuttings

Fig. 4.14 Half and quarter core as samples (Image courtesy of Pedro Rodríguez)

preferentially, yielding a systematically biased result. Once the core has been brought up from underground, it should be washed and then examined to ensure that all the sections of core fit together and that none have been misplaced or accidentally inverted in the box. After the core is in the correct order, the core recovery is measured throughout the mineralized interval, and where losses have occurred, an attempt is made to assign these to specific depth ranges in the core boxes. Core is commonly split along the main axis, one-half being maintained for geologic information and the rest generating material for analysis. The decision to utilize mainly half or quarter (Fig. 4.14) as a sample for assay is based on the requirement for a sample size adequate to overcome any nugget effects. In general, half-core split lengthwise is the most common amount taken for assay. Core splitting can be done with a mechanical splitter or with a diamond saw (Fig. 4.15), being sawing the standard and preferred way to sample solid core. Thus, the core is sawn lengthways into two halves using a diamond-impregnated saw. The diamond saw also gives a flat surface on which the mineralization can be examined with a hand lens and on which intersection angles of bedding or vein contacts can be measured with ease.

Fig. 4.15 Core cutting with a diamond saw (Image courtesy of Euromax Resources)

Half-core must be stored safely because it is a crucial background material with which to create new ideas of both geologic and grade continuity as understanding of a deposit evolves (Vallée 1992). For this reason, to take photographs of the split core in the core boxes is one of the most used procedures to preserve evidence of the character of the core and is especially needed if all of the core is consumed for assaying or testing milling procedures. The next stage is the subdivision of the split core into sample intervals. There are many criteria that could be taken into account, and a decision has to be made as to what information is most important and what may be lost without too great an impact. To some extent, the method that will be used to compute the ore reserves will also play a role in the final decision (e.g., classical methods or geostatistics – see ▶ Sect. 4.4.6) (Annels 1991).

4.2.5 Sampling Pattern and Spacing

The sampling pattern is a consequence of the sampling method, the accessibility of the site, the objectives of the project, and the further requirements for statistical analysis of the data. For this reason, uniform grid sampling is preferred for deposits of any appreciable size so that optimal statistical coverage can be obtained. In practice, the final pattern is generally a compromise between what is preferable and what is convenient or economical. Since most of the sampling methods are necessary, the main goal in optimizing a sampling pattern is to produce the exact number of samples required for representing the grade and dimensions of an ore body. It is essential to take enough ore samples to obtain an estimate sufficiently precise to guide evaluation of mining but also to avoid the expense of taking unnecessary samples.

A relatively widely spaced sampling pattern can be useful for the delineation of the mineral deposit and to calculate the resource estimates or where a geologic model can be provided with precision. More closely separated control data are needed for local estimation, especially where the block size for estimation procedures is clearly smaller than the drillhole spacing at a first step of prospection (Sinclair and Blackwell 2002) (Table 4.3). A systematic grid of samples taken normal to the ore zone is commonly the preferred pattern because it originated a good statistical

Table 4.3 Drilling grid spacing used for exploration and development in nickel laterites

Stage	Drill hole spacing (m)
Reconnaissance	300 × 300
Deposit outline	100 × 100
Detailed definition	$33\frac{1}{3} \times 33\frac{1}{3}$
Mine planning and grade control	$16\frac{2}{3} \times 16\frac{2}{3}$ or staggered $33\frac{1}{3} \times 33\frac{1}{3}$

Data courtesy of Sherritt International Corporation hedge. Sampling patterns progress as the mineral deposit evaluation process evolves through successive steps, and large and relatively uniform ore deposits may be effectively sampled at intervals as great as 100 m or even 200 m. In less regularly mineral deposits, for instance, in gold deposits, the following general guideline can be used: a drillhole spacing between 25 and 30 m is required for measured resources, about 50 m for indicated, and rarely inferred resources are informed if the spacing is more than 100–120 m.

Perhaps the most worrying question to answer is whether a deposit is being under- or over-drilled. The best sampling interval is commonly based on understanding of the nature of the deposit and on empirical studies of predicted and realized grades in blocks of ground. Different statistical methods have been used in an attempt to resolve this problem such as those based on variation coefficient (Koch and Link 1970), correlation coefficient (Annels 1991), Student's *t*-distribution (Barnes 1980), or successive differences (De Wijs 1972), among others. In fact, the coefficient of variation serves not only to guide the number of ore samples to be taken in order to obtain a specified precision of an unsampled ore deposit. It serves also as a guide to the form of statistical distribution that is likely to be appropriate for data analysis and as a measure to control the quality of sampling (Koch and Link 1970).

There is no doubt that the semivariogram is the best estimator of sampling interval where sampling is done by drilling (see ▶ Sect. 4.4.6.2). According to the range of the semivariogram, which is a measure of correlation among samples, a critical distance can be outlined, that is, the optimum spacing between drillholes or sample locations in this particular direction would be indicated by the range of the semivariogram; samples taken at a greater distance would miss significant correlation. It is important to bear in mind that more drillholes do not always imply more precision on reserve estimates. Figure 4.16 (Annels 1991) is a good example of this assertion since the relationship between drilling grid size, number of holes drilled, and the precision of reserve estimates is not linear and the maximum precision is not strictly related to the maximum number of drillholes. In other words, further drilling improves the confidence only to a certain extent and marginally.

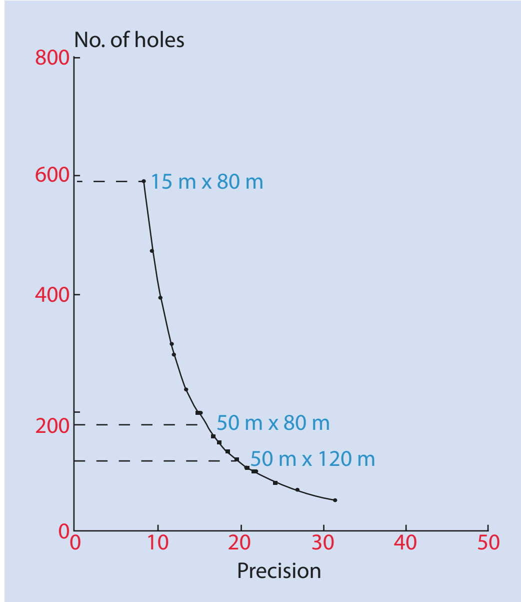


Fig. 4.16 The relationship between drilling grid size, number of holes drilled, and the precision of reserve estimates for the Offin River placer, Ghana (Annels 1991)

4.2.6 Sample Weight

A long recognized pitfall of ore reserve estimation is the dependence between sample size and assay distribution, often referred to as the volume/variance relationship. Mathematically, samples are treated as point values without dimensions, but in reality samples are taken at many different support sizes. It is clearly observed that as the support size increases, the variance of the assay will reduce. Thus, it is crucial in sampling to estimate the smallest simple mass to guarantee that a sample is representative of the whole. The initial weight of a sample must be representative, but not too big since reducing the bulk of a sample for chemical analysis is time-consuming and expensive. The appropriate weight is influenced by the following factors:

1. The distribution of the ore: the initial weight can be smaller on deposits with a regular distribution of useful minerals such as massive and banded structures.
2. The size of the ore fragments: the coarser the useful minerals, the higher the initial weight of the sample should be and conversely.
3. The specific gravity of the mineralization: the higher the specific gravity of a useful mineral,

Table 4.4 Minimum permissible sample weight for a given particle size in aggregates (EN 932-2)

Maximum particle size (mm)	Minimum permissible sample weight (g)
1	100
2	200
4	500
8	800
16	1,000
32	2,000
63	10,000

the larger the initial weight of the sample must be.

4. The mean grade of the ore: the lower the average content of useful mineral, the larger the initial weight of the sample must be.

From an empirical point of view, many tables to calculate the minimum sample weight are present in the literature. For instance, Table 4.4 illustrates the data from EN 932-2 (1999) used to select the minimum permissible sample weight in aggregates for a given particle size. On the other hand, there are several formulas to estimate the initial weight of the sample such as, for instance, the Richards-Czeczott formula (Kuzvart and Bohmer 1978), the Royle formula (Royle 1992), or the Page formula (Page 2005).

Thus, the necessary weight of sample (Q) can be often determined using the Richards-Czeczott formula:

$$Q = k \times d^2$$

where d is the size of the largest grain of useful mineral and k is a constant expressing the qualitative variation of the deposit. This constant ranges from 0.02 for deposits with uniform distribution of the economically valuable component (e.g., large stratabound sedimentary deposits) to 1.0 for deposits with extremely irregular distribution of the useful mineral (e.g., diamond or gold deposits).

Another way to establish the initial weight of the sample is to apply the Royle formula (Royle 1992). A simple expression to give a minimum safe weight (MSW) of sample can be derived from the expression: weight of metal in the largest mineral particle divided by MSW equals maximum contribution made by this particle to the analysis. If the largest mineral particle in a deposit contains A grams of metal and the grade is expressed in percent metal and if this particle is not to contribute more than G% to the analysis, then:

$$\frac{A}{\mathrm{MSW}} = \frac{G}{100} \quad \text{or} \quad \mathrm{MSW} = \frac{100A}{G}$$

For example, if the largest galena grains in a mineralization are spherical and are 2 cm in diameter, then the weight A of contained lead in such a grain is 27.2 g. If G is set to 0.2% for example, then:

$$\mathrm{MSW} = \frac{100 \times 27.2}{0.2} = 13.6\ \mathrm{kg}$$

In the Page method, the size of the sample is such that the largest particle is diluted by the bulk of the sample to the same extent. Therefore:

$$V_\mathrm{s} = V_\mathrm{lp} \times V_\mathrm{d}$$

Where V_s is the volume of sample, V_lp is the volume of largest particle, and V_d is the volumetric dilution. Dilution is the inverse of concentration, so the reciprocal of the grade measures the dilution of the mineral by the country rock. Mineral-volumetric grade is a suitable way of expressing grade proportion. An example is 2.5 $\mathrm{cm}^3/\mathrm{m}^3$ native gold meaning that 2.5 cm^3 of native gold are likely to be found in 1 m^3 of country rock. Again continuing the example, for the mineral-volumetric grade 2.5 $\mathrm{cm}^3/\mathrm{m}^3$ = 2.5 cm^3 per 10^6 cm^3 the corresponding dilution is $10^{-6}/2.5$ = 400,000. Thus, considering spherical particles of 2 mm in size and a density of 16.5 g/cm^3 in a country rock of density 2.75 g/cm^3 where the mineral volumetric dilution is 400,000, then the sample mass will be 4,608 g.

4.2.7 Sample Reduction and Errors

Most field and mine samples need to be reduced in size for laboratory assay. In general, some grams of homogeneous very fine material at 100–150 µm size are needed by the laboratory for chemical analysis. This process of reduction is achieved by progressive comminution to ensure that the reduced volume of the largest valuable particle, if included in or excluded from the reduced sample size, does not cause an unacceptable difference in the assay result; the process is also called subsampling (e.g., Pohl 2011). It designates procedures that reduce the total mass sampled to the few grams of powder in a small bottle that is all a modern laboratory requires for analysis. Thus, the reduction value is around 1,000 times in a kilogram sample and 1,000,000 with a ton sample, if 1 g of sample is required to analysis. However, the final weight of a sample is chosen at 0.5–1 kg because a certain number of samples are deposited as duplicates in the chemical laboratory and in the mining company. Regarding the size of the particles, in practice grinding would be continued to pass sieve 200 µm for fire assay and even finer where chemical dissolution is involved. The normal result of an inadequate sample reduction system is a large random error in assays, including sampling plus analytical error. Obviously, these large errors contribute to a high nugget effect.

There are two main forms to generate errors in sampling process: (a) related to the inherent properties of the material being sampled and (b) from inappropriate sampling procedures and preparation. Errors can be introduced at many stages during sampling of an ore deposit and also during crushing and splitting of the sample in preparation for analysis. In the first case, the sample taken can be too small to be truly representative of the large block of ground to which its value will be assigned, or, in the case of diamond drill sampling, the two halves of the core can contain different concentrations of mineralization. In general, sampling errors can be classified into four main groups: (a) fundamental error, which is due to the irregular distribution of ore values in the particles of crushed ore to be sampled; (b) segregation and grouping error, which results from a lack of thorough mixing and the taking of samples; (c) integration error, which results from the sampling of flowing ore; and (d) operating error, which is due to faulty design or operation of the sampling equipment, or to the negligence or incompetence of personnel (Assibey-Bonsu 1996). Sampling protocols must be designed, so they will minimize the errors introduced through improper procedures (second to fourth group). The fundamental error is the only error that cannot be eliminated using proper sampling procedures because it will be present even if the sampling operation is perfect.

The preparation of samples depends on their size, physical properties, and on the analytical method to be used. Samples are reduced by crushing and grinding, and the resultant finer-grained material is separated by halving or quartering into discrete mass components for further reduction. For this reduction, a relationship between the sample particle size, mass, and sampling errors was established (Gy 1979, 1992). It has been widely accepted and sometimes criticized. The Gy relationship gives an expression for the relative variance (error) at each stage in the sampling reduction process (fundamental error). Therefore, it is possible to either calculate the variance for a given sample size split from the original or calculate what subsample size should be used to obtain a specified variance at a 95% confidence level.

In any reduction system, the most sensitive pieces of equipment are the crushers and grinders. Each one works efficiently within a limited range of weight performance and size reduction. Depending of the primary size of fragments, the sample must be crushed in jaw crushers and then ground and pulverized to the final analytical size in rotary mills or disk mills. The reduction of sample weight is carried out by riffle division method or by coning and quartering method. In the riffle division method, the sample shall be mixed well and placed with a uniform thickness into the riffle tray and divided into almost two equal parts (Fig. 4.17). Either of the two divided samples shall be selected at random each time the sample is reduced.

The sample splitters are commonly called riffle or chute splitters and consist of a series of chutes that run in alternating directions and producing a randomly divided two equal-sized fractions. One of the fractions can then be split again, and the process can be reiterated until a sample of the desired size is generated. If a material is recurrently split into smaller fractions using a riffle, the errors from each procedure of splitting will be added together, resulting in increasing variance between samples. The rotary, or spinning, riffle is the best method to use for dividing material into representative samples. In these riffles, the material to be sampled is fed to a feeder, which drops

Fig. 4.17 Riffle division method (Image courtesy of Alicia Bermejo)

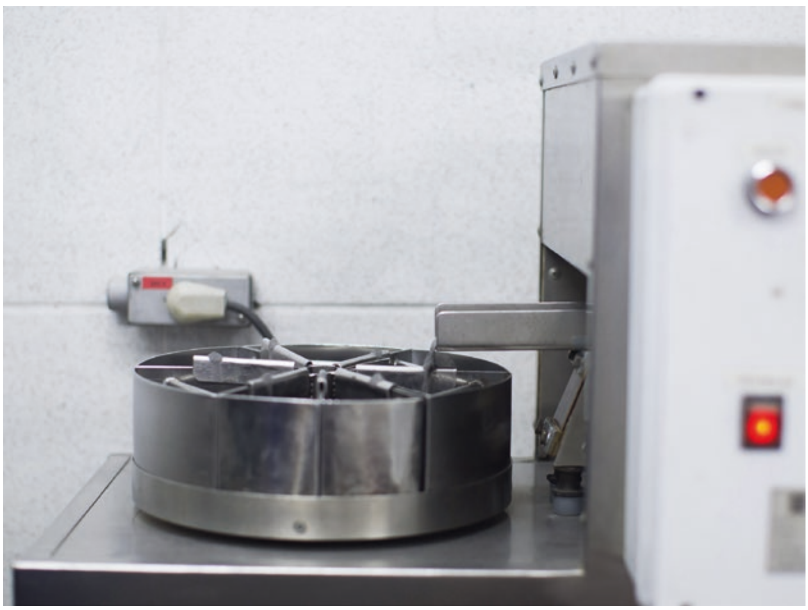


Fig. 4.18 Rotary riffle (Image courtesy of Anglo American plc.)

the material at a uniform rate into a series of bins on a rotating table (Fig. 4.18).

Where a mechanical splitter is not available for separating finely crushed material or where the fragments in a bulk sample are too large to be handled, the sample can be reduced by the method of coning and quartering (Fig. 4.19). In this method, the crushed ore shall be well mixed up and then scooped into a cone-shaped pile. After the cone is formed, it shall be flattened by pressing the top of the cone with the smooth surface of the scoop. Then it is cut into quarters by two lines, which intersect at right angles at the center of the cone. The bulk of the sample is reduced by rejecting any two diagonally opposite quarters.

A simple rule in sample reduction is that all fragments must be crushed to such a size that the loss of any single particle would not affect the analysis. This rule without numbers depends on the

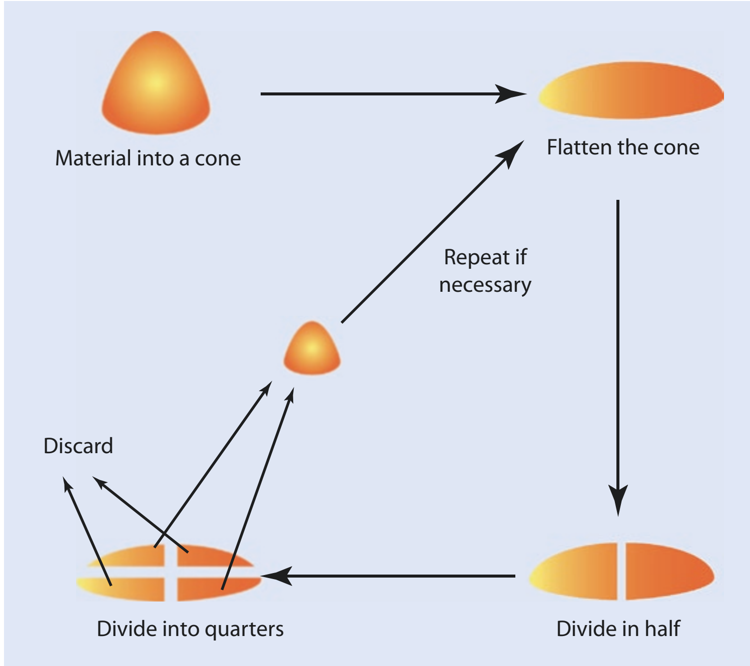


Fig. 4.19 Coning and quartering method

Table 4.5 Empirical guidelines for the maximum allowable particle size in respect to approximate sample weights (Peters 1978)

Weight of sample (kg)	Size (diameter) of largest piece (cm)
250	5.0
60	2.5
40	2.0
22	1.5
10	1.0
3	0.5
1	0.3

accuracy required, the contrast in value between ore and rock particles, and the size of the sample. Empirical guidelines for the maximum allowable particle size in respect to approximate sample weights are shown in Table 4.5 (Peters 1978). For a very homogeneous ore, somewhat larger particle size would be acceptable. A sequence of crushing and splitting in which each step is selected according to some values determined by a variant of Richards-Czeczott formula (see previous section) can be outlined (Kuzvart and Bohmer 1978).

Regarding analytical errors, assaying can be done by a commercial or company laboratory. In any case, a certain percentage of the samples, usually a minimum of 10%, should be assigned a new sample number and resubmitted for a repeat analysis to provide a check on the analytical precision of the laboratory. It is also recommended practice to send a percentage of the samples to a different laboratory for accuracy comparison (Fig. 4.20). Should there be any doubt as to the accuracy of the particular laboratory used, a few standard samples, including a blank, should be submitted for analysis. Control samples can be included in the sampling stream, before shipment to the assay laboratory.

Three types of errors can occur when making measurements in a laboratory: «(a) random errors, which are usually due to an inherent dispersion of samples collected from a population; as the number of replicate measurements increases, this type of error is reduced; (b) instrument calibration errors, which are associated with the range of detection of each instrument; uncertainty about the calibration range varies; and (c) systematic errors or constant errors, which are due to a variety of reasons such as biased calibration-expired standards, contaminated blank, interference (complex sample matrix), inadequate method, analyte instability, among others» (Artiola and Warrick 2004).

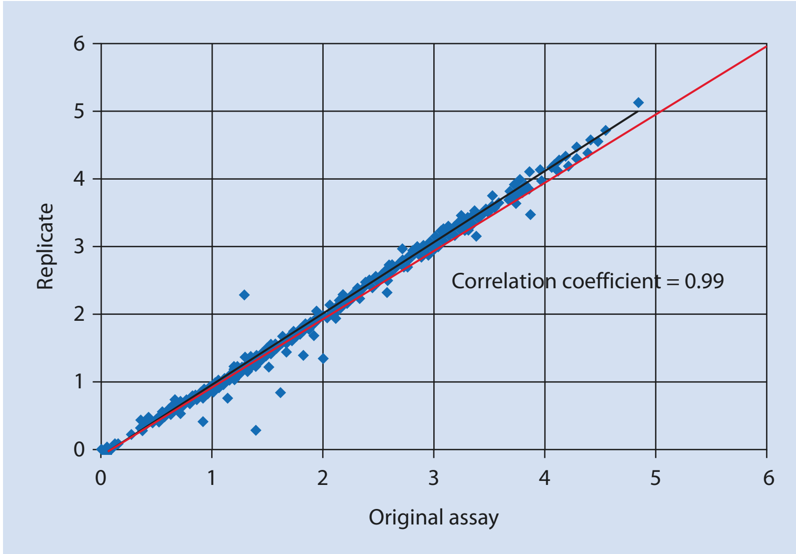


Fig. 4.20 Samples analyzed in a different laboratory for accuracy comparison

4.3 Determination of Grades

Evaluation of grade distribution and estimation of overall grades are the first quantitative analyses of the grade data and are basic tools to provide inputs to the resources/reserve estimation. The grade of ore on a portion of a mine or on an entire deposit is estimated by averaging together the assay returns of the samples that have been taken. The process involves basically two methods of estimation: weighting techniques and statistical techniques (mean, median, geometric mean, and Sichel's *t* estimator). The first ones are commonly applied to estimation of grades in drillholes, whereas statistical estimators of grade require the samples are randomly, but uniformly, distributed throughout the area being evaluated and that the values are far enough apart to be independent variables.

4.3.1 Weighting Techniques

Grade estimations involving assay intervals in drillholes are enough for a general estimate of a potential mineral deposit in the early steps of prospection. One of the most frequent calculations is to compute a grade value for a composite sample (e.g., the average grade of a channel sample from data intervals of several lengths) developing a weighted average for unequal sample lengths and/or widths. Thus, each sample grade in an intersection of a deposit can be weighted in a variety of ways (Annels 1991). The first is simply by length-weighting, in which the sum of the products of intersected length and grade are divided by the sum of the intersected thickness. This method can be expressed mathematically as follows:

$$G = \frac{\sum_{i=1}^{n}(G_i \times L_i)}{\sum_{i=1}^{n}(L_i)}$$

where G indicates weighted grade, n is the number of samples combined, and G_i and L_i are the grades and lengths of each sample, respectively. Sometimes a thickness × grade (metal accumulation) values is computed and utilized to estimate minimum mining width.

All these calculations assume that there is no significant difference in the specific gravities of different types of material and thus that equal volumes represent equal weights. The assumption is usually not far from the truth, but if certain portions of the ore body consist of material that is considerably heavier or lighter than the average, it can be necessary to weight the samples not only for volume but for specific gravity. It often occurs in vein deposits where massive sulfide and disseminated mineralization are present together. So previous equation should be modified as follows:

$$G = \frac{\sum_{i=1}^{n}(G_i \times L_i \times \mathrm{SG}_i)}{\sum_{i=1}^{n}(L_i \times \mathrm{SG}_i)}$$

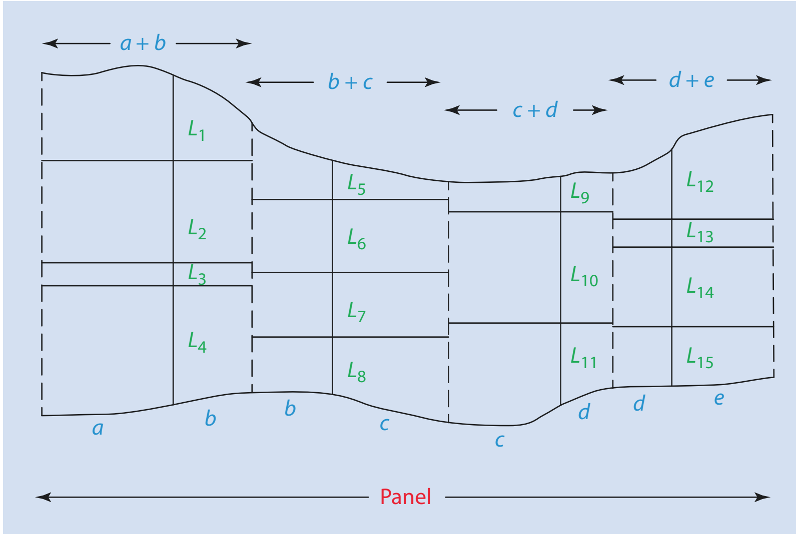


Fig. 4.21 Face sampling and zones of influence (Annels 1991)

4

where SG_i is specific gravity of each sample. Precise application of the principle of weighting for specific gravity would require specific gravity determination for each sample, a practice which is not common and, ordinarily, is hardly warranted. In some ores, the specific gravity is closely related to the assay value so that it is feasible to construct a curve based on a limited number of determinations and then read off the specific gravity corresponding to any given metal content.

Another weighting method is the frequency weighting. It was originally developed for the evaluation of the reserves of Witwatersrand gold ores (Watermeyer 1919). It requires the production of a frequency histogram or curve from a large assay data base which is assumed to be representative of the deposit from which the intersection has been made. For each assay value (G_i) obtained during the sampling, the corresponding frequency of occurrence (F_i) is read off and used to weight the assay as follows:

$$G = \frac{\sum_{i=1}^{n}\left(G_i \times L_i \times F_i\right)}{\sum_{i=1}^{n}\left(L_i \times F_i\right)}$$

Very high assay values, which only occur infrequently, are thus assigned a very low frequency weighting factor and their tendency to bias the overall grade is reduced. For this reason, this technique is applicable where abnormal assays (outliers) are present (see ▶ Sect. 4.2.3).

Finally, where a face which has been sampled by vertical channels at irregular intervals and by samples of variable length must be evaluated (◘ Fig. 4.21; Annels 1991), weighting by zone of influence is applied. In this situation, the weighted grade assigned to the panel is thus calculated by multiplying the grade of each sample by its area of influence, which is based on the sum of half the distances to the adjacent channels (the *ZOI*) times its sample length. According to ◘ Fig. 4.21, estimation of grade would be

$$\begin{aligned} G_p &= \Sigma\left(L_i \times ZOI_i \times G_i\right) / \Sigma\left(L_i \times ZOI_i\right) \\ &= \left[L_1 \cdot G_1 + L_2 \cdot G_2 + L_3 \cdot G_3\right. \\ &\quad + L_4 \cdot G_4\left(a+b\right) \\ &\quad + \left(L_5 \cdot G_5 + \ldots + L_8 \cdot G_8\right)\left(b+C\right) \\ &\quad + \left(L_9 \cdot G_9 + \ldots + L_{11} \cdot G_{11}\right)\left(c+d\right) \\ &\quad \left. + \left(L_{12} \cdot G_{12} + \ldots + L_{15} \cdot G_{15}\right)\left(d+e\right)\right] \\ &\quad / \left[\left(L_1 + L_2 \ldots + L_4\right)\left(a+b\right)\right. \\ &\quad + \left(L_5 + \ldots + L_8\right)\left(b+c\right) \\ &\quad + \left(L_9 + \ldots + L_{11}\right)\left(c+d\right) \\ &\quad \left. + \left(L_{12} + \ldots + L_{15}\right)\left(d+e\right)\right] \end{aligned}$$

Compositing

Raw data in a mineral deposit are usually matched in such a way as to generate composites of roughly similar support, being composites combinations of samples. The term compositing, where used in mineral resource evaluation, is applied to the process by which the values of adjacent samples are

Table 4.6 Results of a statistical validation of composited intervals

Domain	Mean Au grade (g/t)		Number of intervals		Interval length (m)	
	Composite	Raw[a]	Composite	Raw	Composite	Raw
A	0.75	0.75	92	165	91.7	91.8
B	7.64	7.62	14	32	11.5	11.7
C	9.21	9.17	28	91	23.7	23.7
D	9.96	9.92	42	132	36.7	36.9
E	11.14	11.07	38	148	50.4	50.7
F	4.15	4.15	23	77	28.7	28.7
G	2.54	2.45	10	27	9.4	9.8
H	7.47	7.44	34	113	37.7	37.9
I	4.03	4.03	4	44	12.1	12.1

[a]The mean value was weighted by interval length

matched so that the value of the longer intervals can be evaluated. Thus, compositing is a numerical process that includes the estimation of weighted average grades over larger volumes than the original samples (Sinclair and Blackwell 2002; Hustrulid et al. 2013). Data are composited to standard lengths to due to many reasons such as:

1. Reduce the number of samples.
2. Provide representative data for analysis where irregular length assay samples are present.
3. Bring data to a common support; for example, to combine drill core samples of different lengths to a general length of 1 m.
4. Reduce the effect of isolated high-grade data.
5. Produce bench composites, that is, composites extending from the top of a bench to the base in an open-pit; such composites are especially helpful if two dimensional evaluation procedures are utilized in benches.
6. Incorporate dilution (e.g., in mining continuous height benches in an open-pit exploitation).
7. Provide equal-sized data for geostatistical analysis.

After compositing, the composited drillhole dataset is commonly validated (Table 4.6).

Since compositing is linear in nature, a substantial smoothing effect (reduction in dispersion of grades) results because compositing is equivalent to an increase in support. It should be considered that compositing can also be performed for values of variables other than grade. Downhole composites are computed using constant length intervals that generally start from the collar of the drillhole or the top of the first assayed interval. These composites are used where the holes are drilled at oblique angles (45° or less) to the mining benches and bench composites would be excessively long (Noble 2011). Bench compositing has the advantage of providing constant elevation data that are simple to plot and interpret on plan maps. For large and regular mineral deposits where the transition from ore to waste is gradual, the compositing interval is often the bench height and fixed elevations are selected. This bench compositing is nowadays the procedure most generally utilized for resource modeling in open-pit mining (Hustrulid et al. 2013).

In the process of compositing, the starting and ending points of each composite is recognized, and the value of composite grade is estimated as a weighted average by matching the samples included within these limits (Fig. 4.22). In the case of a sample that crosses these limits, only the part of the sample that falls within the mineralization is included in the calculation. If density is extremely variable, for example, in massive sulfides, compositing must be weighted by length times density.

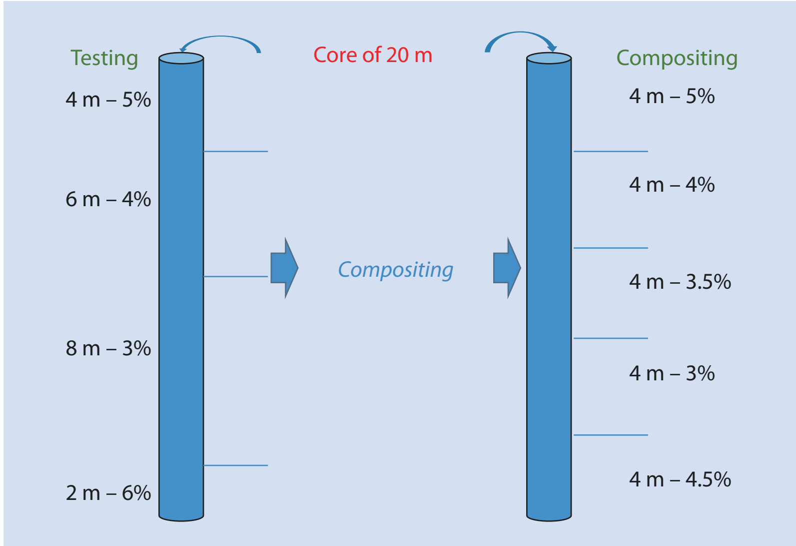


Fig. 4.22 Compositing

4.3.2 Statistical Estimation of Grades

Statistical estimators of the grade of a deposit require that the distribution of grades be Gaussian or normal. This probability density function is the common bell-shaped curve, which is symmetric about the mean value of the distribution. Normal curves can be adjusted to an unbiased histogram to prove the probability that the variable is normally distributed (Fig. 3.42). The first stage in the process is therefore the production of histograms or frequency curves so that an overall impression of the nature of the assay distribution can be obtained. The approach to normality of this population can also be assessed by producing a cumulative frequency diagram and a probability plot. Once the arithmetic mean and associated variance or standard deviation are calculated, then the shape of the assay distribution can also be described in terms of skewness. This value measures the departure from symmetry for a population. A positive value indicates a positive skew (e.g., excess of high values compared to a normal population), while a symmetrical distribution should approach zero.

The coefficient of variation C, expressed as standard deviation divided by mean, is also used to describe the variability of assays in a deposit. For a data population to be considered as normal, the coefficient of variation should be less than 0.5, and larger values indicate either lognormality or an erratically distributed data set (Koch and Link 1970). Other values cited are less than 1.0 (Carras 1984) and less than 1.2 (Knudsen 1988). Where there is any doubt of the normal distribution of the grades, a chi-square test can also be carried out since this test is used to determine mathematically how closely the natural distribution can be compared to a normal distribution. Thus, the «closeness» of the approximation is tested (Fig. 4.23). Chi-square test compared observed data (e.g., grade values) with data awaited to obtain using a specific hypothesis (normal distribution) and «decided» if the observed data can be adjusted to a normal distribution according to a predefined level of confidence.

Normal Population

If the data conform to a normal population (the simple assumption of a normal distribution occurs only rarely for geological data), the sample mean (X), arithmetic mean or average value, or the 50 percentile (median) value is conceptualized on the central tendency of distribution parameters around it is distributed. This value is calculated by the sum of the values of all observations within the population divided by the number of samples, and it is used as average grade estimator of the group of samples, bench, or an entire deposit.

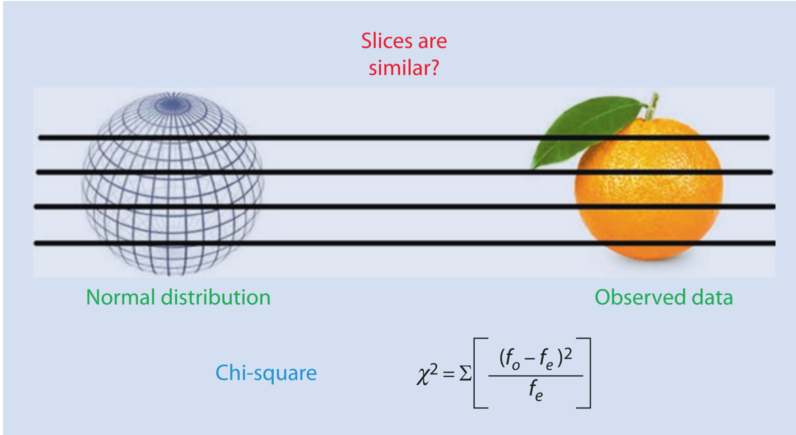


Fig. 4.23 Chi-square test

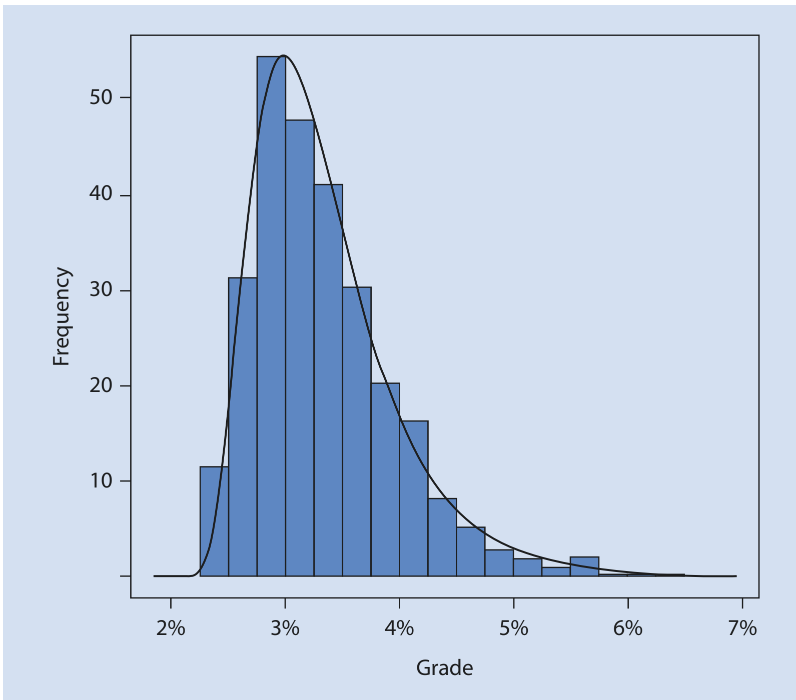


Fig. 4.24 Lognormal population (right skewed)

Lognormal Population

Most of natural distributions encountered in geology are not symmetric, but they are usually more or less skewed to the right, that is, positively skewed (Fig. 4.24). Thus, higher grades occur in addition to the average grade and they extend beyond the range considered as normal distribution. The lognormal distribution, in which logarithms of the individual values can be described by a normal distribution, has become very important for the treatment of skewed distributions in exploration geology. In this sense, experience shows that in the majority of cases, geological assay data do not display a normal distribution but rather that their logarithms trend to be normally distributed (David 1977). The type of logarithm is not important, and either the natural logarithm, which is based on the natural number $e = 2.7183$ (thus x is transformed to $\ln x$) or the decimal logarithm to the base 10 (thus x is transformed to $\log x$) can be used.

Where a population is positively skewed, it is generally advisable to undertake a log transformation of the data and then replot the histogram to see if the population is normalized by this process. If it is, then it is possible to describe the population as being a two-parameter lognormal population (the parameters being log mean and log variance). Again, a chi-square test or by plotting a log-probability diagram can be used to test the approach to normality of the log-transformed data. Logarithmic values are therefore used for the derivation of the mean and calculation of the variance and standard deviation, in the same way as has already been described for normal untransformed values. All values to be considered in logarithmic distribution have to be >0; otherwise statistical parameters like the mean and the variance cannot be calculated.

The parameters normally used to describe a lognormal distribution are the median of the distribution, which is $\gamma = e\alpha$, being α the average of the logarithms and β their standard deviation. This characterization is most used in ore reserve calculations (Sichel 1952; Krige 1951), and the better way to estimate the mean of a lognormal population is to use the following relationship:

$$\bar{X} = e^{\alpha} \cdot e^{\frac{\beta^2}{2}} = e^{\left(\alpha + \frac{\beta^2}{2}\right)}$$

The mean of the lognormal distribution, which is the geometric mean, is commonly less than the arithmetic mean. Sichel (1966) developed a factor, the Sichel's *t* estimator, to solve the problem of obtaining the best estimation of the arithmetic mean for skewed sample sets that have an approximately lognormal distribution (Box 4.2: Sichel's *t* estimator).

Box 4.2

Sichel's *t* Estimator

Sichel (1966) developed a factor, Sichel's *t* estimator, to solve the problem of obtaining the best estimation of the arithmetic mean for skewed sample sets that have an approximately lognormal distribution. Thus, where an assay population is small ($n < 30$), for example, at the early feasibility stage of deposit evaluation, and where the raw data population has a high coefficient of variation and is lognormal, Sichel's *t* estimator can be used to estimate its mean. The *t* estimator is a useful conservative estimator of the arithmetic mean for small data sets where a lognormal distribution can be assumed with confidence. However, it should be realized that if the log-transformed assay population deviates from normality, then Sichel's *t* estimator would also be biased. Thus, the best estimator of a deposit is the one that gives the lowest variance where the variance of the data about the estimator is calculated.

Sichel's *t* estimator can be calculated from

$$t = m \times f(V; n)$$

where $m = \gamma = e^{\alpha}$ and *f* is a value obtained from tables which is a function of *V* and *n*, being $V = \beta^2$ and *n* the number of samples (α is the average of the natural logarithms of the data and β their standard deviation). Tables for rapid determination of the *t* estimator are provided in the literature. Moreover, 95% confidence limits can also be determined using tables provided by Sichel up to samples of size 1,000 and variance up to 6.0. These tables give the values ϕ_{95} (*V*; *n*) and ϕ_5 (*V*; *n*) which, when multiplied by *t*, give the upper and lower confidence limits, respectively.

For instance, the results of five gold grade analyses (g/t) are the following: 3.6, 7.4, 9.5, 8.1, and 14.3. Consequently, the natural logarithms are as follows: 1.28, 2.00, 2.25, 2.09, and 2.66, respectively. Thus, the average of these logarithm data (α) is 2.06 and their standard deviation (β) is 0.45 ($V = \beta^2 = 0.20$). Calculation of $m = \gamma = e\alpha$ gives a result of 7.85. Therefore, the formula to estimate the arithmetic mean using *t* estimator is

$$t = 7.85 \times f(0.2)5$$

In Table 4.7a the value for $n = 5$ and $V = 0.2$ is 1.103. Thus:

$$t = 7.85 \times 1.103 = 8.66 \text{ g/t}$$

If the upper and lower confidence limits must be calculated, then ϕ_{95} (*V*; *n*) = 2.087 (Table 4.7b) and ϕ_5 (*V*; *n*) = 0.713 (Table 4.7c). Therefore:

$$\text{Upper limit} = 8.66 \times 2.087 = 18.07 \text{ g/t}$$
$$\text{Lower limit} = 8.66 \times 0.713 = 6.17 \text{ g/t}$$

Thus, the estimate of the arithmetic mean grade of this data is 8.66 g/t with a 95% probability that this estimate lies between 6.17 g/t and 18.07 g/t.

Table 4.7. Sichel's *t* estimator tables: **a** Sichel's function (*V*; *n*), **b** upper confidence limit factor ϕ_{95} (*V*; *n*), and **c** lower confidence limit ϕ_5 (*V*; *n*)

a

V	*n* = 2	3	4	5
0.00	1.000	1.000	1.000	1.000
0.02	1.010	1.010	1.010	1.010
0.04	1.020	1.020	1.020	1.020
0.06	1.030	1.030	1.030	1.030
0.08	1.040	1.040	1.040	1.040
0.10	1.050	1.051	1.051	1.051
0.12	1.061	1.061	1.061	1.061
0.14	1.071	1.071	1.071	1.072
0.16	1.081	1.082	1.082	1.082
0.18	1.091	1.092	1.092	1.093
0.20	1.102	1.102	1.103	1.103

b

V	*n* = 5
0.00	1.000
0.02	1.241
0.04	1.362
0.06	1.466
0.08	1.561
0.10	1.652
0.12	1.740
0.14	1.827
0.16	1.914
0.18	1.999
0.20	2.087

Table 4.7 (continued)

c

V	*n* = 5
0.00	1.000
0.02	0.8978
0.04	0.8589
0.06	0.8302
0.08	0.8070
0.10	0.7870
0.12	0.7693
0.14	0.7535
0.16	0.7389
0.18	0.7255
0.20	0.7129

4.3.3 Outliers

Outliers are anomalously high values outside the main population which result in grade bias (Annels 1991) or observations that appear to be inconsistent with the vast majority of data values (Sinclair and Blackwell 2002) (Fig. 4.25). How to consider these errant high values is one of the essential problems in ore evaluation. The reason why no rules of thumb can apply to all cases is that, no two orebodies being alike, erratic highs can reflect any one of a number of conditions depending on the manner in which valuable minerals are distributed throughout the ore body. Thus, the problem is fundamentally geological rather than purely mathematical (McKinstry 1948). The populations of outlier are usually geologically distinct and display limited physical continuity relative to lower grade values. Therefore, to establish that high grades can be expanded into neighboring rock could originate a significant overstatement of the resource or reserves.

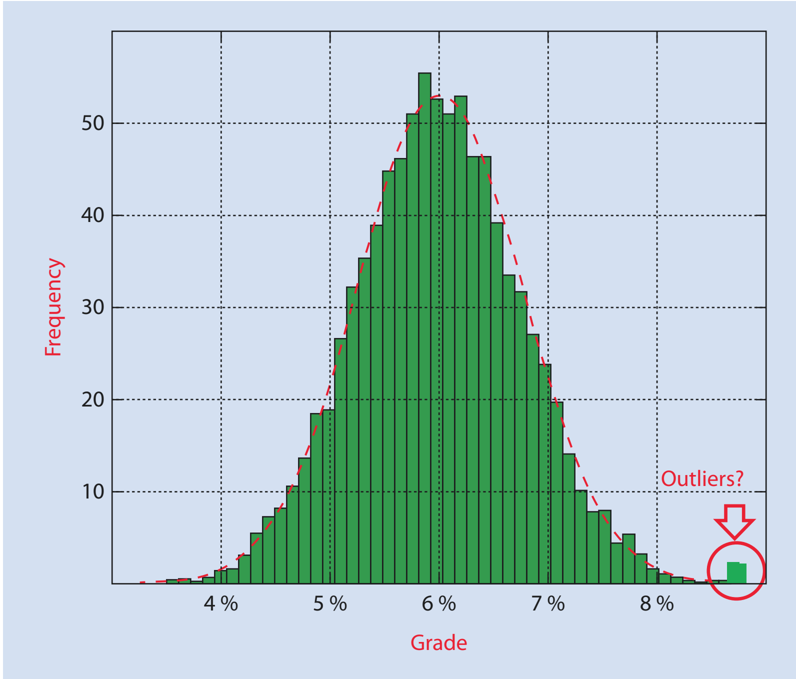


Fig. 4.25 Outliers

These abnormal assays can appear in a sequence of assays that, if not due to contamination, reflect much localized random phenomena such as gash veins, concretions/accretions, or coarsely crystalline aggregates of the valuable mineral (Annels 1991). In other words, sometimes the outliers depict different geologic population in the data that can correspond with an identifiable physical domain and this domain can be accounted separately of the main domain. It is necessary to decide whether to accept them, even though they are much localized and will probably heavily weight or bias the results, or whether to reduce them in some way. In any case, all outlier values must receive special handling, which can involve a number of options: (a) reanalyzing if possible, (b) cutting (also capping) to some predetermined upper limit based on experience, or (c) using an empirical cutting method»(Parrish 1997). The most common method to resolve the problem of outliers is to cut the grade to the average of the adjacent samples, or to the mine average grade, or to an arbitrary percentile value (e.g., 95th percentile of data) based on a cumulative frequency or log-probability plot of mine assays. Alternatively, the mean plus two or three standard deviation value of the mine assay population could be calculated and applied as the level of cut. Table 4.8 shows an example of the result of a capping process for different rock types in a gold mineralization.

Nowak (2015) recommended the following steps in a procedure for treating outliers during resource estimation:

1. determine data validity considering errors in sampling and handling;
2. review geology logs for samples with high grade assays; capping may not be necessary for assays where the logs clearly explain the presence of high grade;
3. capping should not be considered for deleterious substances that have negative impacts on project economics;
4. decide if capping should be considered before or after compositing;
5. keep capping to a necessary minimum;

Table 4.8 Grade capping for different rock types in a gold mineralization

Rock type	Capping grade (g/t Au)	Percentile	No. of samples capped	Metal loss (%)
GWK	25	99.07	127	3.61
SHL/ARG	30	99.58	15	6.97
SLT	20	99.14	17	10.04
MD	30	98.04	26	13.05
RDA	20	99.50	51	1.23
RDF	16	98.91	26	3.35
RDX	30	99.84	34	1.22
RDXB	28	99.71	22	0.99
RDXL	10	98.87	40	2.07

6. restrict influence of very high grade assays; commercial software is well designed for this approach;
7. visually and/or numerically assess the effect of high grade assays to be sure they don't affect estimated block grades; and
8. check the effect of capping on final resource estimates and document the differences.

4.3.4 Coproduct and By-Product

By-product components are both economically and technologically valuable minor elements that are obtained from the ores of the main metals. These components are generally present in ppm ranges, whereas the main metals occur within percent ranges in the mineralization. For instance, germanium occurs in zinc ores, gallium in bauxites, indium in zinc, copper or tin ores, tellurium in copper ores, hafnium in zirconium ores, and tantalum in tin ores. Moreover, many high-technology commodities currently are mostly provided by-product commodities.

Three types of commodities can be defined and classified according to the relative value of each commodity (Jen 1992). Thus, «the principal (metal) product of a mine is the metal with the highest value of output, in refined form, from a particular mine, in a specified period; a coproduct is a metal with a value at least half ... that of the principal product; and a by-product is a metal with a value of less than half ... that of the principal product. By-products are subdivided into significant by-products, which are metals with a value of between 25% and 50% ... that of the principal product and normal by-products, which are metals with a value of less than 25%… that of the principal product». Evaluation of coproducts and by-products usually is carried out by methods similar to that of the main component (e.g., inverse distance weighting or kriging; see the next headings). In these cases, each estimate of the products is calculated regardless of the other, with the tacit assumption that no important correlation is present among the different products, being the estimation procedure time-consuming and costly. In other cases, these estimation processes can be carried out indirectly if a strong correlation among the coproducts and by-products to the principal component is present. Consequently, many multi-mineral deposits are generally valued, planned, and operated on the basis of equivalent grades (Box 4.3: Equivalent Grades).

4

Box 4.3

Equivalent Grades

Multi-mineral deposits are generally valued, planned, and operated on the basis of equivalent grades. The use of equivalent grades for these types of deposits has been a standard practice in the mining industry for many years, especially for base metal deposits. Equivalent grades are used commonly to simplify the problem of mineral inventory calculation by estimating a single variable, rather than the two or more variables from which the single variable (equivalent grade) is derived. In general, the use of equivalent grades should be discouraged (Sinclair and Blackwell 2002). In this approach, each mineral is converted to its equivalent economic value in terms of one of the minerals, which is taken as a standard. For example, in a silver-lead-zinc deposit, a weighted sum of the three metal grades can be used to provide a single zinc-equivalent grade. This is generally done to avoid the complexities of a three-dimensional, or in general n-dimensional, grade analysis (Cetin and Dowd 2013). With this method, the amounts of each mineral extracted in the mining stage and sent to the processing plant and subsequent stages are estimated on the basis of equivalents and not on the basis of the component minerals.

Since equivalent grade values are values in which the grade of one metal is expressed in terms of another, after allowance has been made for the difference in metal prices, an example of determination of Au equivalent grade (Au eq) in an Au deposit containing some Ag should be as follows:

$$\mathrm{Aueq}(\mathrm{g/t}) = \mathrm{Au}(\mathrm{g/t}) + k \cdot \mathrm{Ag}(\mathrm{g/t})$$

where k is a parameter that generally is taken as the ratio of Ag price to Au price (e.g., $k = 1/66$ if Au and Ag values are USD 990/oz and USD 15/oz, respectively). It can be seen that equivalent grades depend on both prices and grade, and thus they are time-dependent based on how prices behave and how grades vary during operation. Metal recoveries can also be included in the calculation.

Considering the cutoff grades, operating cutoff grades for the equivalent grades do not necessarily correspond to achievable, or even meaningful, cutoff grades for the grade-tonnage distributions of the individual minerals. While there is direct relationship between the individual grades and the equivalent grade, there is no unique inverse relationship from the equivalent grade back to the individual grades. The actual amount of each individual mineral above the equivalent cutoff grade therefore will differ from the values calculated from the equivalents. This difference will increase as the correlation among the components decreases. Thus, using equivalent grades can undervalue or overvalue mining projects.

In other cases, for the purpose of assigning a dollar value to mineral blocks, so that a cutoff can be applied to show reasonable prospects of economic extraction, a dollar equivalent can be calculated in a similar way.

4.4 Cutoff Grade and Grade-Tonnage Curves

The so-called cutoff grade is commonly the standard value that discriminates between ore and waste within a given mineral deposit (Fig. 4.26). As economic conditions change continuously, obviously the cutoff grade can increase or decrease. Thus, it is the most important economic feature for estimation of resource and reserve data from prospecting information. It is common to calculate the resources/reserves of a mine for different cutoff grades and plot the results as a series of curves, usually termed grade-tonnage curves, which are widely used in the mining industry. From geology and mining planning to management and investment areas, grade-tonnage curves are used for economic and financial analysis, being probably one of the most important tools for representing variations in the characteristics of a deposit in function of cutoff grades.

4.4.1 Cutoff Grade

Cutoff grade (COG) is generally defined «as the minimum amount of valuable product of metal that one metric ton of material must contain before this material is sent to the processing plant» (Rendu 2014) or it is «an artificial boundary demarcating between low-grade mineralization and techno-economically viable ore that can be exploited at a profit» (Haldar 2013). A similar definition of cutoff grade is as «any grade that, for any specific reason, is used to separate two courses of action, for example to mine or to leave, to mill or to dump» (Taylor 1972). These definitions are utilized to discriminate raw materials that cannot be mined from those which must be processed. Therefore, cutoff grades reused to choose blocks of ore from waste blocks at various stages in the evolution of mineral resources/reserve estimation in a mineral deposit (e.g., during prospection and mining stages). Consequently, if material concen-

Fig. 4.26 Discriminating between ore and waste in underground mining (Image courtesy of North American Palladium Ltd.)

tration in the mineralization is above cutoff grade, it is defined as ore; conversely, if material concentration is below cutoff grade, it is considered as waste. However, blending methods (low-grade and high-grade mineralization) are commonly carried out in the mine for an effective usage of the mineral resources.

Cutoff grade is a geological/technical measure that embodies the important economic aspects of mineral production from a deposit. It is defined not only by the geological characteristics of the deposit and the technological limits of extraction and processing but also by costs and mineral prices. Annels (1991) classified the many factors that influence the cutoff grade in three categories: geological (e.g., mineralogy, grain size, presence of deleterious-penalty elements, shape and size of the deposit, structural complexity, or water problems), economic (e.g., accessibility to markets, labor availability, current metal prices, political and fiscal factors, cost of waste disposal and reclamation, or capital costs and interest rates), and mining methods (open-pit versus underground mining) (Table 4.9). Change of any one criterion or in combination of more gives rise to different cutoff and average grade of the deposit. For instance, if mineral prices rise and all costs stay the same, then the COG will fall because extraction of mineralization with lower grades now will be profitable. COG can vary significantly from deposit to deposit, even in those that are very similar geologically because of differences among deposits in a wide variety of factors, as those cited above.

Table 4.9 Cutoff grades based on underground mining method

Mining method	Gold price (US$/oz)	Total cash cost (US$/t)	Cut-off grade (g/t)
Mechanized cut-and-fill	530	67.54	4.6
Longitudinal longhole	530	60.85	4.2
Transverse longhole	530	59.43	4.0

Data courtesy of Eldorado Gold Corporation

The concept of cutoff grade works well in case of deposits with disseminated grade gradually changing from outer limits to the core of the mineralization; on the contrary, in heterogeneous vein-type deposits with rich mineral at the contacts, the COG indicator has no use in establishing the ore boundaries (Haldar 2013). It is possible also to differentiate between COG and minimum mining grade (MMG), since there is a confusion in the utilization of both terms. Thus, one definition of COG is «the lowest grade material that can be included in a potentially economic intersection without dropping the overall grade below a specified level, referred to as the minimum mining grade» (Annels 1991).

Technical literature includes many publications on estimation and optimization of cutoff grades, being the most comprehensive reference the book entitled *The Economic Definition of Ore: Cut-Off Grades in Theory and Practice* (Lane 1988). This book is considered the standard for mathematical formulation of solutions to COG estimation where the objective is to maximize net present value (see ▶ Sect. 4.5.1.4) because the cutoff grades define the profitability of a mining operation as well as the mine life. There are many approaches for the determination of cutoff grades, but most of the research done in the last four decades shows that determination of cutoff grades with the objective of maximizing NPV is the most acceptable method. A high cutoff grade can be utilized to increment short-term profitability and the net present value of a mineral project, but increasing the cutoff grade is also likely to decrease the life of a mine. This shorter mine life can also produce higher socioeconomic effect with decreased long-term jobs and decreased profits to employees and local communities (2013). It is generally accepted that «the COG policy that generates higher NPVs is a policy that use declining cut-off grades throughout the life of the project» (Ganguli et al. 2011).

Estimation of cutoff grade, although a complex economic problem, is tied to the concept of operating costs per ton and can be viewed simplistically for open-pit mines (John 1985). Although long-range production planning of an open-pit mining operation is dependent upon several factors, cutoff grade is probably the most significant aspect, as it provides a basis for the determination of the quantity of ore and waste in a given period (Asad and Topal 2011). Thus, operating cost per ton milled, OC, is given by (John 1985)

$$\mathrm{OC} = \mathrm{FC} + (\mathrm{SR} + 1) \times \mathrm{MC}$$

where FC are the fixed costs per ton milled, SR is the strip ratio, and MC are the mining costs per ton mined. Cutoff grade, useful at the operational level in distinguishing ore from waste, is expressed in terms of metal grade; for a single metal, cutoff grade can be determined from operating cost as follows:

$$g_{\mathrm{c}} = \frac{\mathrm{OC}}{p}$$

where g_{c} is the operational cutoff grade (e.g., percent metal) and p is the realized metal price per unit of grade (e.g., the realized value from the smelter of 10 kg of metal in dollars where metal grade is in percent).

Another equation to derive the cutoff grade (e.g., in gold) is the following:

$$\begin{aligned}\text{Cutoff grade (gold)} = {} & [\text{mining cost} + \text{process cost} + \text{general and administrative (G \& A) costs}] \\ & /[(\text{payable recovery} \times ((\text{gold price} - \text{refining and sales cost}) \\ & /\text{conversion factor})) \times (1 + \text{royalty})].\end{aligned}$$

An example of estimation of cutoff grade using this equation is as follows:

$$\begin{aligned}& \text{Cutoff grade (4.94 g/t gold for mineral reserves)} \\ & \quad = [\text{mining cost US\$50/t} + \text{process cost US\$38/t} + \text{G \& A costs US\$62/t}] \\ & \quad /[(\text{payable recovery 95\%} \times (\text{gold price US\$1,100/oz.} \\ & \quad - \text{refining and sales cost US\$7/oz})/\text{conversion factor 31.1035g/oz}) \\ & \quad \times (1 + \text{royalty 10\% of sales})];\end{aligned}$$

It is important to note that «sustainable development basis are being increasingly applied by mining companies and there is a balance between the cut-off grade determination and sustainable mining practice» (Franks et al. 2011). In fact, to obtain the optimal cutoff grades and maximum NPV, the environmental issues and social impacts must be included in the mine design (Mansouri et al. 2014). Thus, optimum cutoff grades determination is counted as one of the main challenges in sustainable development principles of mining, including environmental, cultural, and social parameters. Therefore, an optimum cutoff grade model must rely not only on economic and technical considerations but also reclamation, environmental, and social parameters (Rahimi and Ghasemzadeh 2015).

4.4.2 Grade-Tonnage Curves

At the early stages of the planning of a mine, an important decision tool is the grade-tonnage curve. For a given cutoff, a certain tonnage of ore is expected and consequently a certain profit. If the tonnage later proves to be less than expected, the consequences are obvious (David 1972). Thus, it is common practice to calculate the resource tonnage at a series of cutoff grades since the resource potential of a mineral deposit will be determined by the cutoff grades (Fig. 4.27). The action of changing these values usually produces a clear impact on resource/reserve data. The information is plotted on a grade-tonnage graph and the obtained curves are called grade-tonnage curves, which are essential in mine planning. It is clear that compilation of this information will mean knowing the deposit fully. The information can be also showed in table format (Table 4.10). Grade-tonnage curves are used extensively and updated regularly to calculate the impact that different cutoff grade strategies have on the economics of a mining operation. The type of information, for example, sample data or block estimates, used in the construction of a grade-tonnage curve should be documented clearly.

The approximation of the grade-tonnage curve to reality is highly dependent on some natural parameters as the geology and grade distribution of the deposit. In general, the more variable the grades, the more complex is the geometry and the less reliable becomes the curve. All grade-tonnage curves contain several errors, including those based on an abundance of closely information. However, obviously the better is the quality of data, the better are the calculations and the grade-tonnage curves obtained. One error that needs to be mentioned in grade-tonnage curves is analytical and sampling error since the election process is not based on true grades but on estimated grades from samples. With relatively little data at the prospection stage, large sampling and analytical error can generate an important effect on the grade-tonnage patterns, usually originating an overvaluation of high-grade tonnage (Sinclair and Blackwell 2002).

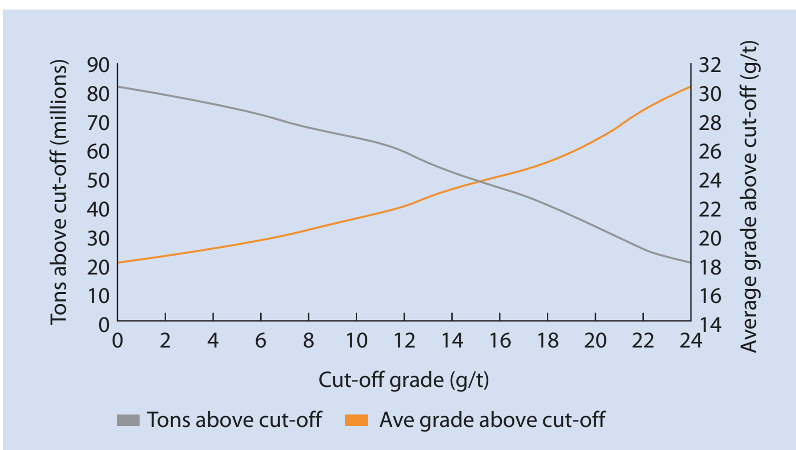


Fig. 4.27 Grade-tonnage curves (Illustration courtesy of AngloGold Ashanti)

Table 4.10 Grade-tonnage table including multi-element information

Cut-off grade (g/t PD)	Tonnage	PD	PT	AU	NI	CU
	$T \times 1000$	g/t	g/t	g/t	%	%
3.0	461	3.65	0.225	0.181	0.084	0.077
2.9	558	3.53	0.219	0.176	0.082	0.075
2.8	687	3.40	0.214	0.170	0.080	0.074
2.7	866	3.27	0.208	0.164	0.079	0.071
2.6	1,088	3.14	0.204	0.158	0.077	0.068
2.5	1,491	2.98	0.212	0.145	0.078	0.064
2.4	1,745	2.90	0.210	0.145	0.077	0.064
2.3	2,078	2.81	0.205	0.143	0.076	0.063
2.25	2,298	2.76	0.200	0.141	0.075	0.062
2.2	2,524	2.71	0.197	0.140	0.075	0.062
2.1	3,019	2.62	0.194	0.138	0.074	0.062
2.0	3,614	2.53	0.191	0.137	0.073	0.061
1.9	4,288.68	2.436	0.187	0.135	0.072	0.061
1.8	5,162.87	2.336	0.181	0.132	0.071	0.060

4.5 Estimation Methods

The prediction of grade and tonnage in a mineral deposit is an essential problem in mineral resource estimation. The classical approximation to this issue is to calculate the mineral grade for quantities significant to the mine planning and base the recoverable resource estimation on those calculations (Rossi and Deutsch 2014). The process of calculating a mineral resource can only be carried out after the estimator is convincing of the robustness of the factors that justify the evaluation process, from choice of method of sampling to sales contract specifications. In this sense, ore estimation is the bridge between exploration, where successful, and mine planning (King et al. 1982). Thus, the geological data must be sufficiently complete to establish a geological model and this itself «must have internal consistency, should explain the observed arrangement of lithological and mineralogical domains, and should represent the estimator's best knowledge of the genesis of the mineral deposit» (Glacken and Snowden 2001). In summary, regardless of the method used, all estimates start with a comprehensive geological database, primarily derived from drilling; without detailed, high-quality geological and geochemical data, a resource estimate cannot be considered valid.

The estimation procedure is not only a mere calculation but also a process that includes assumption of geological, operational, and investigational information. All estimates should have the best possible geological input combined with well thought out statistical or geostatistical treatment; no purely mathematical estimate should be accepted. The calculations therefore form only part, and not necessarily the most important part, of the overall procedure. It is common practice in exploration to begin with economic evaluations as early as possible and to update these evaluations in parallel with the physical exploration work. In an early stage, the geologist has only a tentative idea about expected grades and tonnages based on the initial geological concept and early concrete indications through observations from trenches or a limited number of drillholes. This early idea about grades and tonnages can be called grade potential and tonnage potential (Wellmer et al. 2008). In this sense, the four Cs (character of mineralization, continuity, calculation, and classification) are the basis for the correct estimation of ore resources or reserves (Owens and Armstrong 1994).

4.5.1 Drillhole Information and Geological Data

The essential data needed for resource estimation are derived from drillhole information. It includes detailed logs of the rock types and mineralization and geochemical and assay data for all samples that were collected. It also includes survey data for each drillhole. It is critical that the locations in 3-D space of the mineralized zones are known. Moreover, the shape, form, orientation, and distribution of mineralization in a deposit must be known with sufficient confidence to estimate the grade and tonnage of mineralization between drillholes.

Regarding the geological model, it obviously should support the distribution of mineralization achieved by sampling. A geological model involves examining cross sections, long sections, plan maps, and 3-D computer models of the deposit. The resource estimation process includes definition of ore constraints or geological domains, analysis of the sample data, and application of a suitable interpolation technique. In general, less than one-millionth of the volume of a deposit is sampled, and grades and other attributes must be estimated in the unsampled region, which is a high-risk process. In summary, knowledge of the geology of the mineral deposit is a prerequisite to any reliable computation: an incorrect model for the deposit will lead to incorrect resource estimate (Stevens 2010). This understanding involves space location, size, shape, environment, country rock, overburden, and hydrology; mineral, chemical, and physical characteristics of the raw material; as well as average grade and distribution of valuable and gangue minerals (Popoff 1966).

4.5.2 General Procedure

It is important to note that in ore reserve calculation, it is necessary express the data including a volume, a tonnage, and an average grade. The tonnage is derived from the volume by multiplying by the specific gravity of the ore. The volume is commonly determined by calculating an area in two of the dimensions and then multiplying by the third dimension to determine the final volume. To determine total area, it is usually possible to divide the area under consideration into a number of regular geometric figures such as squares, triangles, etc. (Reedman 1979). Thus, the resource or reserve calculation in a mineral deposit includes one formula, or a variation of it, which is always used:

$$T = A \times \mathrm{Th} \times \mathrm{BD}$$

where T is the tonnage of ore, in tons; A is the area of influence on a plan or section, in m^2 or km^2; Th is the thickness of the deposit within the area of influence, in meters; and BD is the bulk density. Then, tons of valuable component (e.g., copper) are obtained multiplying tonnage of ore by the grade of the ore. In summary, the general procedure is a three-step process: limit and volume determination, grade estimation, and mass determination using the specific gravity of the rocks and ores.

The method used to calculate the ore reserve estimation will change according to the type of commodity, type of mineral deposit, geometry, distribution and homogeneity of the ore, mode of data collection, among others, but conceptually the steps to be taken will be always the same as expressed in the previous formula. It should also be borne in mind that ore reserve statement is an estimate, not a precise calculation. All formulas for computing volumes, tonnage, and average factors are approximate because of the irregular size and shape of the ore body, errors in substituting natural bodies by more simple geometric ones, geologic interpretation, assumptions, and inconsistency in the variables. Accuracy of the results usually depends more on geologic interpretation and assumptions rather than on the method used (Fig. 4.28). Resources or reserves of the same category computed by different methods and based on the same data usually differ slightly. In fact, if sampling spacing could be sufficiently close, estimation would be a matter of simple arithmetic; this is almost the situation, for example, in grade control process (see ▶ Chap. 5) where samples are separated 3 or 5 m each other. In other words, the closer the sample spacing, the less important the

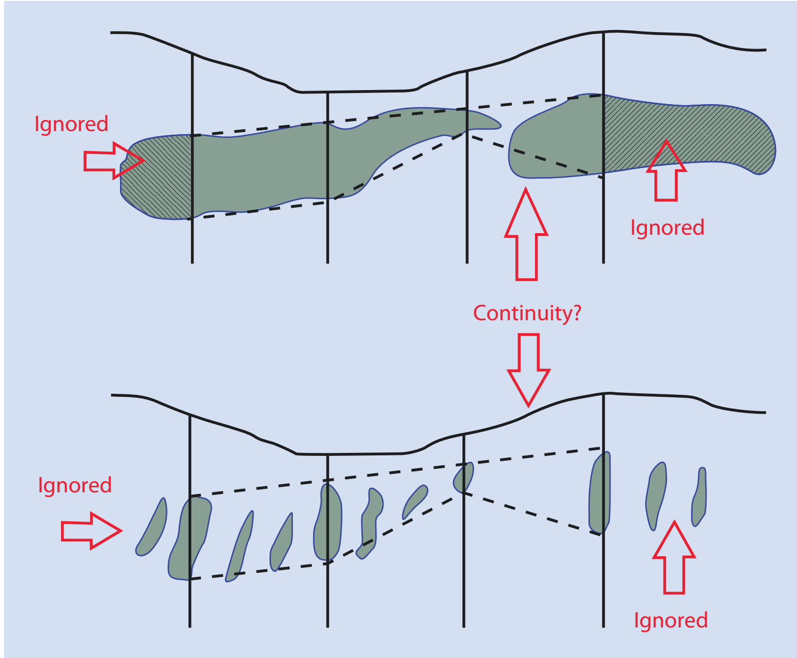


Fig. 4.28 Accuracy of the results depends mainly on geological interpretation

procedure of ore estimation; the sparser the data, the more critical the procedure, not only quantitatively, but also qualitatively, because of the greater dependence on subjective assumptions (King et al. 1982).

4.5.3 Bulk Density

Bulk density or specific gravity, which is a term that is widely used interchangeably with density, is required to convert volumes of ore to tons of ore (tonnage = volume × bulk density). A density that takes voids in account is termed specifically bulk density. Obviously, where porosity is negligible, density and bulk density are equivalent terms. In situ bulk density must be modeled at the time of resource estimation. Although bulk density determinations can seem to be a trivial matter, if the values are incorrect, the accurate amount of mineralization in a deposit cannot be determined: accurate rock bulk density values are required for accurate resource estimates (Stevens 2010). Any error in bulk density determination is directly incorporated into tonnage estimation. Bulk density determination is controlled by many factors such as homogeneity or heterogeneity of the materials to be sampled, the practice of computing dry or wet densities, relationships between ore grade and densities, and many others. If the volume is expressed in cubic feet, it is divided by the tonnage-volume factor, which is the number of cubic feet in a ton of ore. This is the origin of the term «tonnage factor».

The bulk density of a mineralization is obtained by laboratory measurement of field samples (Fig. 4.29) or from the mineralogical composition of the ore. The most common way to determine bulk density of an ore in the laboratory is to weigh a sample in the air and then weighing the same sample suspended in water, and later apply the formula:

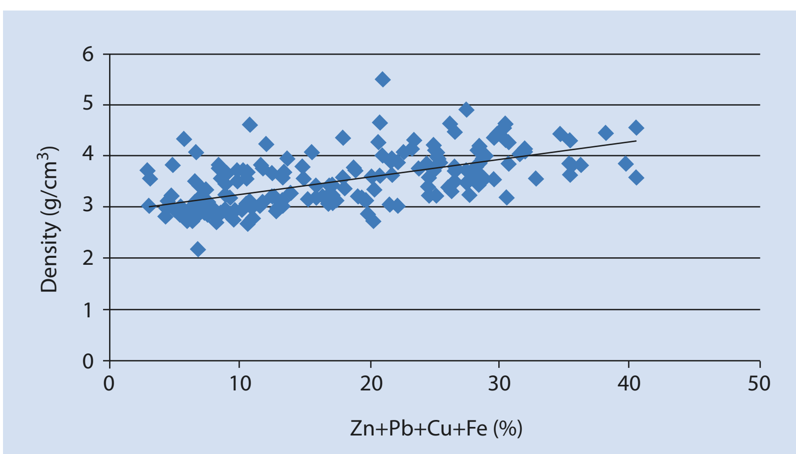
$$\text{Bulk density} = \frac{\text{weight in the air}}{\text{weight in air} - \text{weight in water}}$$

In ore bodies that have more than one contained metal, the method of determining specific gravity based on the mineralogical composition of the mineralization is to compute an average specific gravity utilizing specific gravities of individual minerals and being the percentages of minerals in the ore correctly known. At an early stage of defining the deposit, the bulk density of a suite of representative samples is determined, and these values are applied to the rest of the deposit. Sometimes, a constant value obtained from the average of representative samples is applied for the entire deposit, but this method can lead to considerable errors in the determination of tonnage of ore and contained metal, especially if metal grades are highly variable, if the host-rock lithology changes, if the degree of alteration or depth of weathering is variable, and if the mineralogy of the valuable components change (Annels 1991). Failure to utilize specific gravity in mineral deposits with a high-density contrast between valuable minerals and gangue will result in incorrect determinations of the average grade.

◘ **Fig. 4.29** Station for measuring dry bulk density (Image courtesy of Lydian International Ltd.)

In some cases, different bulk densities are determined and applied to different areas of a mineral deposit and/or different lithologies. Since mineralogical variation is the principal control on bulk density in many deposits, mineralogical zonation is commonly a practical guide to systematic variations in bulk density. For example, in a massive sulfide deposit, samples with between 70 and 90% sulfides will be assigned one bulk density, those with 40–60% sulfides another (lower) bulk density, and so on. In deposits with simple mineralogy, it is often possible to prepare a nomograph relating bulk density to assay data. Thus, the factor used for ore is controlled by changes in the ore content and grade. However, graphs and linear equations of specific gravity against the grade of one metal are not considered accurate enough in a multi-mineral deposit but appear to be very satisfactory in a theoretical one sulfide/gangue mix (Bevan 1993). Alternatively, a relationship between density and combined grades can be established (◘ Fig. 4.30). It is important to bear in mind that a typical massive sulfide deposit contains pyrite and/or pyrrhotite and varying amounts of chalcopyrite, sphalerite, and galena. A more fundamental approach to developing a mathematical model for bulk density is the use of multivariate methods, such as multiple regression. This arises because bulk density is commonly a function of mineralogy and porosity.

◘ **Fig. 4.30** Relationship between density and combined grades



4.5.4 Estimation Procedures

A variety of procedures have been developed to estimate the tonnage and grade of mineralization in a deposit. The methods can be grouped into two categories: classical and geostatistical methods. Classical methods involve commonly the use of section and plan maps, whereas geostatistical methods involve complex, computer-driven 2-D and 3-D statistical techniques to estimate tonnage and grade. The utilization of geostatistical methods involves a further complexity in calculation, all based upon the theory of regionalized variables described by the French mathematician Georges Matheron in the early 1960s. These methods use the spatial relationship between samples, as quantified by the semivariogram, to generate weights for the calculation of the unknown point or block values. The standard technique of geostatistics was called «kriging» by Matheron in honor to the South African mining engineer Danie Krige and the type most frequently utilized is the variants of ordinary kriging, namely, linear kriging techniques.

Classical (also called traditional, geometric, or conventional) estimation methods can be used to assign values to blocks (e.g., polygonal or inverse distance methods), and they are commonly utilized at early stages of a mining project. These techniques are not particularly reliable but can offer an order-of-magnitude resource calculation. They are also utilized to check the results obtained using more complex geostatistical estimation methods. The classical methods have stood the test of time, but, because of the uncertainties and subjectivities involved in assigning areas of influence, they are now largely superseded by geostatistical techniques for the past three decades, which are described in the following section. However, these classical methods are still applicable in many situations and can well produce an end result superior to that possible by a geostatistical method. Critical assessment for the use of geostatistical kriging should always be undertaken before dismissing the classical methods. Too often, attempts to apply kriging are based on the use of mathematical parameters that have not been adequately tested or proven, perhaps due to time or information constraints. Geostatistical methods will only work satisfactorily if sufficient sampling is available to allow the production of a mathematical model adequate to describe the nature of the mineralization in the deposit under evaluation. Otherwise, it is much better to apply one of the classical methods.

Classical and geostatistical methods for reserve estimation in a single deposit are complex to apply with skewed distribution mineralization variables that include grade, ore body thickness, and grade thickness and need sophisticated data processing (Wang et al. 2010). The problem lies in the presence of local outliers or anomalies, which produce great effects on the estimation process, and the need for replace these outliers.

4.5.5 Classical Methods

Classical or traditional methods utilize analytical and geometric procedures and constitute a deterministic approach. The method aims to establish discrete geological boundaries to the mineralization, both in mineral exploration and exploitation, that are directly related to a sampling grid.

For resource/reserve computations, a mineral deposit is converted to an analogous geometric body composed of one, several, or an aggregate of close-order solids that best express size, shape, and distribution of the variables. Construction of these blocks depends on the method selected. Some methods offer two or more manners of block construction, thus introducing subjectivity. In such a case, a certain manner of construction is accepted as appropriate, preferably based on geology, mining, and economics (Popoff 1966). Numerous methods of reserve computations are described in the literature; some are only slight modifications of the most common ones. Depending on the criteria used in substituting the explored ore bodies by auxiliary blocks and on the manner of computing averages for variables, classical methods can be classified into six main types: (1) method of sections, (2) polygonal method, (3) triangular method, (4) block matrices, (5) contour methods, and (6) inverse distance weighting methods (Fig. 4.31). These methods do not consider any correlation of mineralization between sample points nor quantify any error of estimation. All of them are empirical and their use depends mainly of the experience of the user.

Selection of a method depends on the geology of the mineral deposit, the kind of operation, the appraisal of geologic and exploration data, and

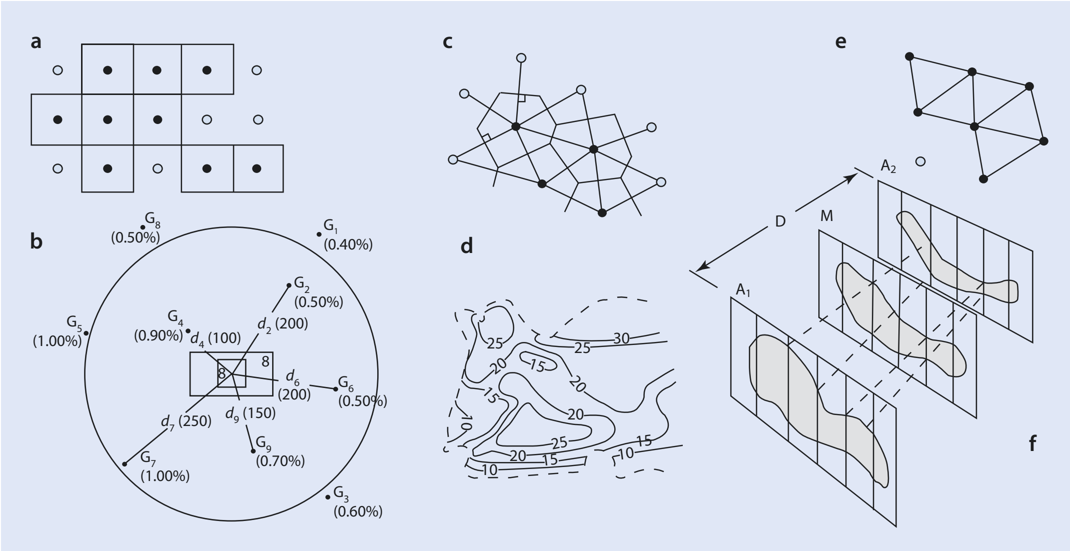


Fig. 4.31 Classical methods for resource/reserve calculations: **a** block matrices, **b** inverse distance weighting, **c** polygons, **d** contour, **e** triangles, and **f** sections

the accuracy required. Time and cost of computations are often important considerations. The purpose of reserve computations is one of the most important considerations in selecting a method. For preliminary exploration, the method should best illustrate the deposit, the operations, and permit sequential computations and appraisal. On the other hand, time-consuming procedures must be avoided if reserves are being computed for prospective planning. The system of mining or the problem of selecting one can influence the preference. A certain method of computation can facilitate more than others the design of development and extraction operations owing to technical and economic factors such as mining by levels, average grade, or different cutoff grades.

A careful analysis of geology and exploration should be made to select the best method of estimation. In general, the method (or combination of methods) selected should suit the purpose of computations and the required accuracy; it should also best reflect the character of the mineral deposit and the performed exploration. In a complex or irregular deposit, it is advisable to use two or more methods for better accuracy and self-confidence. Average of these methods can be accepted as a final result, or the values of one method can be considered as a control of others. Thus, the use of two or more methods to compute reserves for the same deposit is common practice. Various methods can be also applied for different parts of a body depending on the geology, mine design, type and intensity of exploration workings, and category of reserve computations. A second method can often be used for control of the computations made by the principal method, so that no crude errors can occur. A common example of combined methods is where one method is applied to outline and divide the mineral body into blocks and another to determine the parameters of each block.

Cross-Sectional Methods

If a deposit has been systematically drilled on sections according to a regular grid, reserve calculation will be based on cross sections along these lines. The cross-sectional methods are based on a careful consideration of the geology of the mineral deposit and the developing of a correct geological model that is essential for good resource estimates (Stevens 2010). It is possible to distinguish two variables of the standard method: vertical sections or fence used mainly in exploration and horizontal sections or level used in mining. Although there are many geometric possibilities, in the traditional cross-sectional method, the area or ore in a given cross section is calculated (e.g., with a planimeter, counting squares, or through Simpson’s rule), and the volume of the ore body is commonly computed using, as a solid figure,

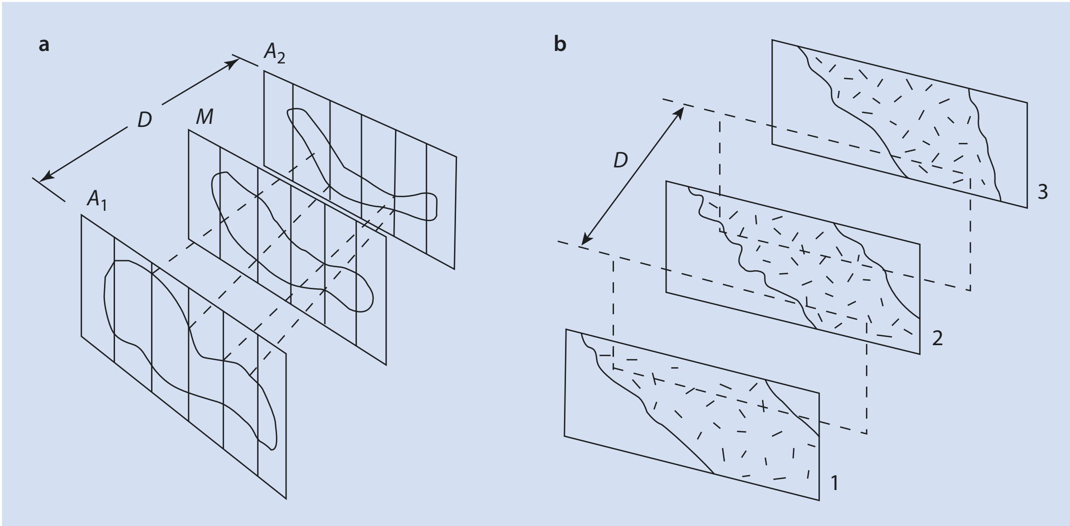


Fig. 4.32 Cross-sectional methods: **a** solid figure formed by two consecutive cross sections; **b** solid figure obtained corresponding to half the distance to the two adjoining sections

two consecutive cross sections and the distance between them (Fig. 4.32a):

$$V = \frac{A_1 \times A_2}{2} \times L$$

where V is the volume, in m^3; A_1 is the area of section A_1, in m^2; A_2 is the area of section A_2, in m^2; and L is the distance between A_1 and A_2, in meters. The interval between sections can be constant, for example, 50 m, or can vary to suit the geology and mining requirements. Another possibility is to compute the volume corresponding to half the distance to the two adjoining sections (Fig. 4.32b). Thus, the limits of the blocks defined lie exactly halfway between the drillholes. Obviously, an end correction is necessary for the volumes at the extremities of the ore body. For these two cases, the volume can be calculated using half the distance between drillholes, seldom more than 50 m. To increase the accuracy of computations, the number of blocks should be as large as possible. Care should be exercised to avoid arbitrary locations and construction of sections. In exploration, distance between sections is usually governed by the character of the mineral body and the distribution of mineral values. Selection of sections unjustified by exploration data can influence the size of the areas and, in turn, computation. Most of the disadvantages in the use of this method can be avoided by properly planned exploration.

The volume of each block multiplied by the bulk density of the mineralization, calculated, for example, in the laboratory with samples including valuable mineral, waste, pores, etc., gives the tonnage of ore in tons. The reserves in tons of the valuable component in each block (e.g., copper in sulfide mineralization) are subsequently estimated multiplying the ore tonnage by the average grade. As explained before, a range of methods is available to determine average grade: statistical, metal accumulation, area of influence, etc. The sum of the tonnages of ore or valuable component in each block generates the total tonnage ore resources/reserves for the entire mineral deposit.

The cross-sectional methods are simple and rapid, but they are not accurate because normally intercross-sectional distance varies between 50 m and 100 m. These methods however are the most convenient ways for computing reserves of uniform mineral deposits. Thus, well-defined and large bodies that are uniform in thickness and grade or show gradually changing values can generally be computed accurately by cross-sectional methods. The method should be used with caution where the bodies are irregular or where values tend to concentrate in some ore zones. Where computations of several valuable components are required and the mineral body shows grade variations for each component, it is difficult and often impossible to apply cross-sectional methods.

In underground mining, horizontal cross sections constructed along the proposed mining levels are often preferred in mine design. Two sets of vertical sections at right angles to each other would better illustrate ore bodies than any other method. The method is applied most successfully in the case of a deposit that has sharp, relatively smooth contacts, as with many tabular (vein and bedded) deposits. Assay information, for instance, from drillholes, commonly is concentrated along equispaced cross sections to produce a systematic data array; in some underground situations, more irregular data arrays can result, for example, from fans of drillholes. The great strengths of the procedure based on sections are the hard geologic control that can be imposed (Sinclair and Blackwell 2002). Moreover, cross-sectional methods are easily adaptable for use simultaneously with other classical methods. In fact, these methods have an advantage over the polygonal methods (see next section) in that it is easy to observe variations in the shape and grade of mineralization.

Method of Polygons

Where drillholes are randomly distributed (e.g., in an irregular grid), the grade and thickness of each hole can be assigned to an irregular polygon, and it is assumed that both variables remain constant throughout the area of the polygon. The polygonal estimate is based on assigning areas of influence around drillhole intercepts. Thus, this method shows the intuitive idea that the amount of data generated by each sample is proportional to its area or volume of influence. The most common drawing of polygons around the drillholes is using a series of perpendicular bisectors of lines joining sample locations (Fig. 4.33a). The perpendicular bisector of a line segment is a line for which points are at the same distance from either side of the line segment. This procedure is equivalent to a process known as Voronoi tessellation. Therefore, in this method each polygon incorporates a unique sample location and all the points included in the polygon are nearer to the contained datum than to any external datum. The Russian scientist B.T. Boldyrev gave the first description of the method applied to geology as early as 1909. Another possibility to define the polygons is to use the angular bisectors (Fig. 4.33b). Here, each polygon is established by linking drillholes with tie lines and then constructing angular bisectors between these lines to define a central polygon (Annels 1991).

There are arbitrary decisions that must be made as to how marginal prisms are bounded at their outer edge. There are different possibilities to resolve this problem, including the utilization of geologic information at the boundary, if possible, or more usually to fix a maximum distance from the sample. A combination of indicated, probable, and possible resources constructing outer fringes and assigning resources categories to each fringe from distances to drillholes can be also used to

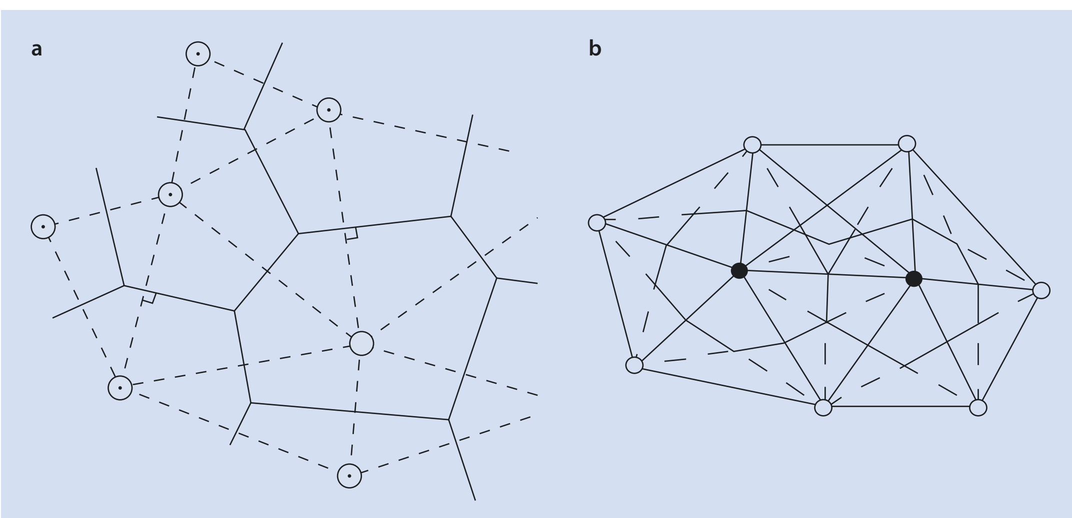


Fig. 4.33 Method of polygons: **a** perpendicular bisectors; **b** angular bisectors

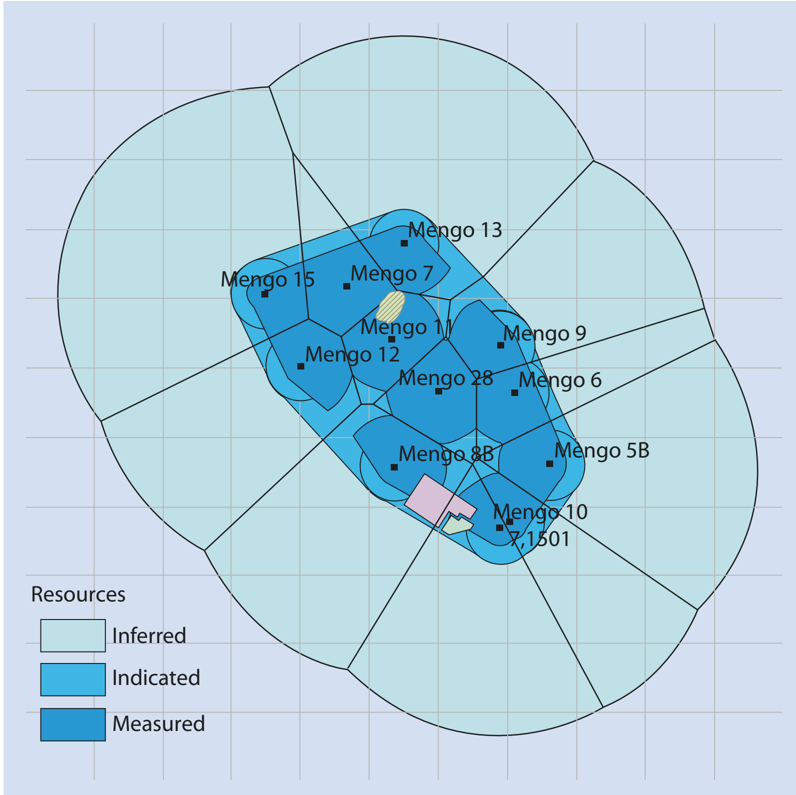


Fig. 4.34 Polygons based on resources categories

solve the problem (Annels 1991) (Fig. 4.34). In any case, the final drawing to close the polygons is almost always arbitrary, which produces an important impact on the results.

The third dimension, that is, the height of the polygonal prism, is defined by the thickness of the deposit or bench and is perpendicular to the projection plane. This process originates a general pattern of polygonal prisms that are assigned the grade of the contained datum. Regarding the grade procedure, the average grade of ore found in the sample point (e.g., drillhole) within the polygon is considered to accurately represent the grade of the entire volume of material within the polygon. In this sense, the use of raw sample grades for mean grades of large volumes overestimates the grade of high-grade blocks and, correspondingly, underestimates the grade of low-grade blocks (e.g., a conditional bias, in which the bias is dependent on the grade estimated).

The polygonal method is deficient in exposing the morphology of the mineral body and the fluctuations of variables within the individual blocks; although average thickness and grade are computed, the pattern of their space distribution is not revealed. An alternative to single grade weighting by polygon can be drawn (Camisani-Calzolari 1983). The method involves allocating 50% of the weight to central drillhole and the remaining 50% to surrounding drillholes, in equal proportions. These weighting coefficients are entirely arbitrary and no allowance is made for thickness. However, it is an attempt to overcome one of the main criticisms of the method: that polygons, sometimes very large in areas of sparse drilling, are evaluated by only one drillhole, totally ignoring adjacent drillholes (Annels 1991). Another possibility is to weigh the grades of the adjacent drillholes according to their distance away the center of the polygon, with the inverse squared of the distance being the most common weighting factor.

With regard to the mathematical procedure of estimation, it is somewhat similar to that used in the cross-sectional method. After the polygons had been drawn, the area of each polygon is computed by using a planimeter or counting squares. Then, a polygonal prism is constructed using the thickness of the mineralization as the height of the prism. The volume computed in the prism is later multiplied by the bulk density of the mineralization to obtain the tonnage of ore in tons and the average grade of the drillhole is then used to calculate the reserves in tons of the valuable component. The sum of ore or valuable component in each polygonal prism produces the resources and/or reserves for the studied mineral deposit.

Favorable criteria for the use of the method of polygons are the proven continuity of a mineral body between drillholes and the gradual changes of all variables. The polygonal method is successfully used in computing reserves of tabular deposits such as sedimentary beds of coal, phosphate rock, or oil shales as well as large lenses and thick vein bodies. The greater the number of polygonal prisms and the more regular the grid, the more accurate are the computations. Polygons must be used with caution in cases of no uniform and irregularly shaped mineral bodies. They are incorrect where the bodies cannot be correlated satisfactorily between drillholes, where they are small and distributed erratically, or where intercalations of waste are present. In mineral deposits composed of several bodies overlying each other, separate groups of polygons can be delineated for each one (Popoff 1966).

Method of Triangles

The method of triangles represents a modification of the polygon method. In this method, a series of triangles is constructed with the drillholes at the apices (Fig. 4.35). This method has the advantage in that the three drillholes are considered in the calculation of the thickness and grade parameters for each triangular reserve block. Obviously, the triangles method is more conservative than the assignment of single values to large blocks, just as in the polygonal method. The construction of the triangles can use Delaunay triangulation, the precursor to Voronoi tessellation. The triangles must have angles as close to 60° as possible, but certainly avoiding acute-angled triangles (Annels 1991). In this way, triangular prisms are defined on a two-dimensional projection (e.g., bench plan) by linking three sample sites so that the resulting triangle contains no internal sample sites. Each triangle on the plant represents the horizontal projection or the base area of an imaginary prism with edges equal to vertical thicknesses of the mineral body in the drillholes. Thus, the average of the three values of the variables, grade and thickness, at the apices of a triangle is assigned to the triangular prism.

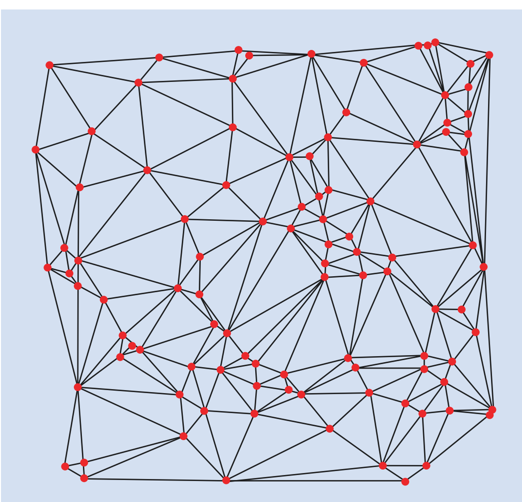


Fig. 4.35 Method of triangles

Calculating ore reserves by this method involves the determination of the area of each triangle using the procedures described above for polygonal or cross-sectional methods, the calculation of the volume of each triangular prism multiplying the area by its weighted thickness, and obtaining the tons of mineralization and valuable component using bulk density and grade, respectively. Where the support of the grades is a constant, as in the bench of an open-pit, there are two main methods of estimating the grade: arithmetic mean and included angle weighting. Discrepancy between the two values obtained increases as the corner angle deviate from 60° (Annels 1991). Where the thickness at each intersection is variable, again two methods can be used to determine grade: thickness weighting and thickness and included angle weighting. Even side lengths of each triangle, distances of each hole from the center of gravity, and/or areas of influence of each hole, constructed by rule of nearest point, can be used for weighting (Popoff 1966).

The principal advantage of this method is that it produces some smoothing in the calculations of individual prisms. As a result, estimation of the tail of the grade density distribution is more conservative than is the case with the traditional polygonal approach. That the samples can be used an unequal number of times is part of the fringe problem: how far should ore be assumed to extend beyond an outside hole in ore, although this problem is common to most procedures. Regarding the disadvantages of this method, Sinclair and Blackwell (2002) suggest the following: (1) the smoothing is entirely empirical; (2) the weighting (equal weighting of three samples) is arbitrary and thus is not optimal other than coincidentally; (3) anisotropies are not considered; and (4) the units estimated do not form a regular block array.

For many decades, the triangular method was considered standard, although errors in results due to the manner of dividing the area into triangles were recognized. The procedure for reserve computations by the method is relatively simple, although modifications of the method, such as included angle weighting or distances of each hole from the center of gravity to calculate grade, required more elaborate computations. The relative error depends on the manner in which the area is divided into triangles, their form, and the total number of triangles. Thus, errors in computing reserves can be substantial, particularly where fluctuation of variables is large and the number of triangles is small. In comparison with other methods, the triangle method requires construction of a greater number of blocks ultimately resulting in labor and time-consuming computations. Where an ore body contains several valuable components, computations can also be cumbersome. Moreover, the method is not exact where variables decrease from the center to the outside boundaries, such as the thickness of lens-like bodies. In these cases, the volume reserves computed will be underestimated. In general, the uniform and gradual changes of variables, which are positive to use the triangular method, are characteristic features of only a few mineral deposits, predominantly sedimentary.

Block Matrices

Where the data are on lines or on rectangular or regular offset grids, regular blocks of square or rectangular shape can be fitted to the drillholes (Fig. 4.36). The method is basically similar to the use in the polygon method and is particularly suited to the exploration phase of drilling of a prospect where rapid updating of the reserve can be undertaken as each new hole is drilled and where precision of the estimates is not as crucial as at a later feasibility or mining stage. According to the method used to construct the blocks, some methods allow extrapolation of mineralization beyond drilling but only use one hole to evaluate each block; other methods give conservative reserves using four holes to evaluate both grade and thickness and they are thus somewhat reliable. Generally, the thickness apply in the latter cases is the arithmetic mean, while the grade is thickness weighted, plus bulk density if required, among the four holes (Annels 1991).

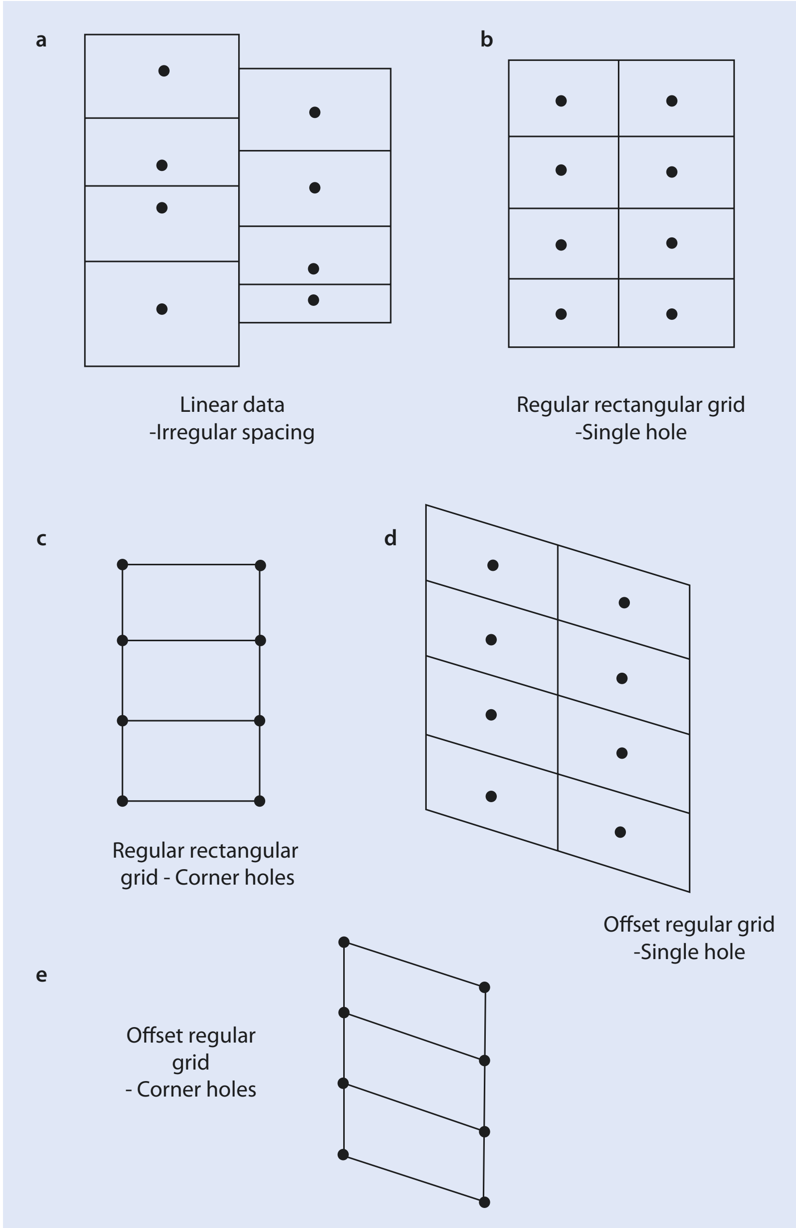
Contour Methods

Contour methods are very simple to use and produce good results, especially for mineral bodies where there are certain natural regularities of the variations in thickness and grade. The methods are based on the assumption that unit values, from one point to another, undergo continues and uninterrupted changes according to the rule of gradual changes. To construct isolines, intermediate values are determined by interpolation between points of known values. As a result, certain properties of mineral bodies can be presented graphically on a plan or section by a system of isolines. Common cases are computation of average thickness (Fig. 4.37), average grade, and average value of a mineral deposit from appropriate isoline maps. The methods require sufficient number, appropriate density, and distribution of observations for accurate plotting of isolines. A major advantage of the methods is their descriptiveness; the isopach map gives an idealized likeness of the mineral body, whereas the isograde map shows the distribution of rich and poor ore. Thus, the boundaries of cutoff ore are easily constructed and changed; likewise, volume can be computed by measuring areas of respective isolines without additional drawing. Moreover, if the requirements for minimum grade, thickness, or value of ore are changed, the isomaps remain the same. The methods of isolines are applicable to deposits of gradual physical and chemical changes such as sedimentary deposits, for instance, large placer gold deposits explored with hundreds of drillholes.

Contouring is normally invoked to avoid the irregular and commonly artificial ore/waste boundary that arises in estimating blocks. In cases in which data are abundant, they commonly are contoured directly without the intermediate stage of grid interpolation. As an estimation procedure, contouring of grades is typically applied to grade control in open-pit mines where the controlling data are blasthole assays.

Up to four methods of contouring can be distinguished (Annels 1991), being the main three described below: (1) the grid superimposition method, (2) the moving window method, and (3) the graticule method. In the grid superimposition method, drillhole intersection points are plotted

Fig. 4.36 a–e Block matrices (Annels 1991)



on plan along with the relevant component of thickness and grade. Contour plans are then produced and a matrix of ore blocks is superimposed, whose dimensions allow them to fit exactly within mining blocks. For all blocks within the ore limits, values are assigned to the midpoint of each block by interpolation between contours, first for thickness and second for grade. Where blocks overlap the boundary, an estimate of the proportion of ore in the block is made together with an estimate of grade and thickness as the center of gravity of this section of ore. To calculate the reserves of the deposit, the area of each block, obviously the same for all blocks, is multiplied by the interpolated thickness, and the volume obtained is multiplied by the bulk density, in a similar way than proceeding methods, to obtain the tonnage of mineralization. The tonnage will be later multiplied by the interpolated grade to compute the valuable component reserves of the block. The sum of reserves of each block will give the reserves for the entire deposit.

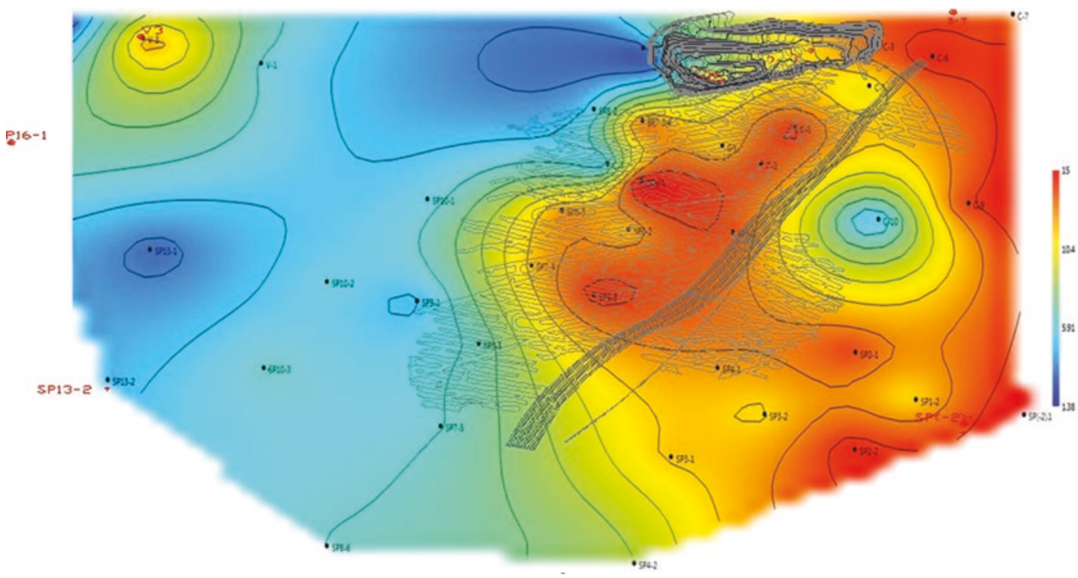


Fig. 4.37 Contour of magnesite average thickness in a magnesite deposit (Illustration courtesy of Pedro Rodriguez)

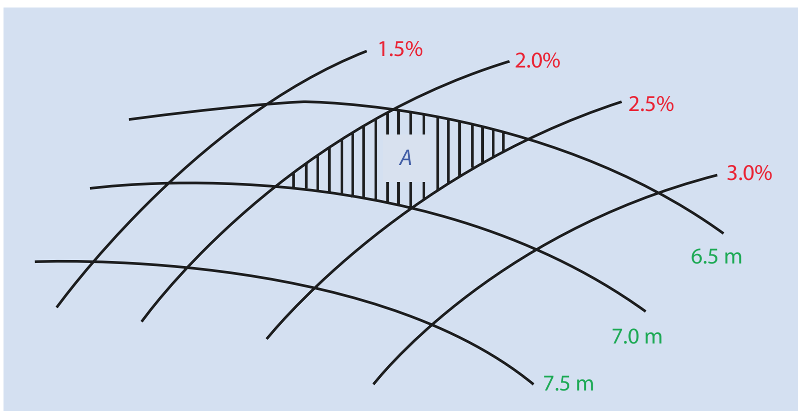


Fig. 4.38 Graticule method (Annels 1991)

The moving window method is a smoothing technique, particularly suited to the calculation of reserves of an open-pit bench that has been intersected by a series of irregularly spaced drillholes, or blast holes, which have revealed a highly erratic fluctuation in bench composite grades. For this reason, contouring of the data is not possible, and as a result, grid superimposition and the grade interpolation method could not be applied. The moving window method involves fitting of a grid of ore blocks to the outline of the deposit in the bench under evaluation. A search window is then drawn whose dimensions are twice as those of each ore block. Ideally, at least 15 drillholes should fall in the search area, so the search window dimensions can be modified to achieve this number if required. As the dimensions of the search windows increase, greater degree of smoothing of the data will be achieved. The window is positioned so that its center falls over the first block to be evaluated and the arithmetic mean of all the raw data values falling in the window, or their log-transformed equivalents, is calculated and the result assigned to this block. The window is then moved laterally to the next block and the above calculation is repeated. The rest of the procedure is equal to that shown in the grid superimposition method.

Where no correlation exists between thickness and grade, the graticule method can be used (Annels 1991) (Fig. 4.38). Contour maps of the variables are superimposed and the area of each graticulate is determined using the methods described above. The thickness and grade assigned to each graticulate within the ore body limits are the mean of the bounding contours. The global procedure is similar to that shown in previous methods: determination of volumes, tonnage of mineralization, and tonnage of the valuable component.

Inverse Distance Weighting Methods

Inverse distance methods are a family of weighted average methods, being one of the most characteristic features the clear smoother process generated in the estimations. Thus, these methods provide for a gradual change in values between multiple sample points rather than an abrupt and unnatural change at the boundary between adjacent polygonal blocks. The technique applies a weighting factor to each sample surrounding the central point of an ore block. This weighting factor is the inverse of the distance between each sample, and the block center is raised to the power «*n*,» where «*n*» usually varies between 1 and 3. Only samples falling within a specified search area (2-D) or volume (3-D) are weighted in this way. Because the method is laborious and repetitive, it is necessary to use a geological modeling software package.

The inverse distance is taken into consideration by assuming that the influence of a borehole over a point varies inversely as the distance. The method begins to take the spatial distribution of data points into account in the calculations, a characteristic that will be repeated with geostatistical methods. Although subjective, inverse distance weighting estimation procedures remain popular. They have been found commonly often to generate results that are somewhat similar to geostatistical estimates produced using ordinary kriging methods. However, the application of ID methods has been steadily decreasing through the years in favor of geostatistical methods.

The procedure comprises the division of the deposit into a group of regular blocks within the geologically defined boundary. The available data are then used to calculate the variable value, thickness or grade of the mineralization, for the center of each block. According the name of the method, obviously near points are given greater weighting than points far away. The weighted average value for each block is calculated using the following general formula:

$$Z_{\mathrm{B}} = \frac{\sum_{i=1}^{n}\left(Z_i / d_i^n\right)}{\sum_{i=1}^{n}\left(1 / d_i^n\right)}$$

where Z_{B} is the estimate of block grade or thickness based on the values of each of these (Z_i) at each sample location in the search area; ($1/d$) is the weighting function, being d the distance of each sample from the block center; and «*n*» is the power to which the distance is raised. It is necessary to define the data utilized in the process, being this selection based on distance factor for the search area, the power factor used, and how many points should be utilized to estimate the center point for each block. Inverse distance methods must be done so that weights sum to one; otherwise, the method is biased and therefore unacceptable.

The most common exponents used are n = 2 (inverse distance squared, IDS) and n = 3 (inverse distance cubed, IDC). Figure 4.39 shows an example of the application of inverse distance method and the influence of the power «*n*» in the estimation final result (Annels 1991). Three-drillholes fall in the search area (circular) and their grades (%) are given in the diagram, together with distance values. As can be seen in

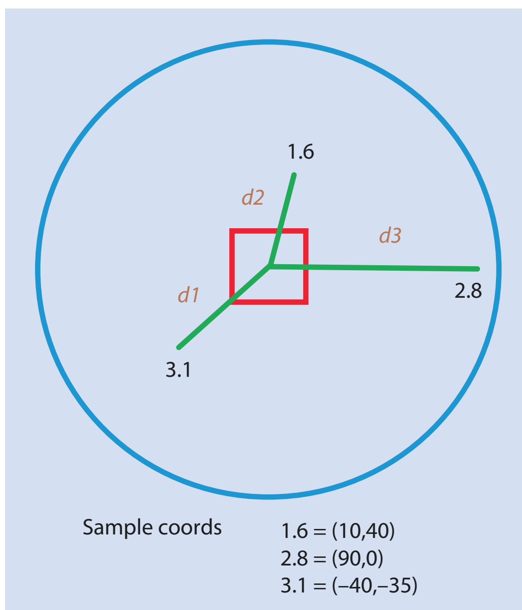


Fig. 4.39 Inverse distance weighting with circular search area (Annels 1991)

Table 4.11 Results of the example considered in Fig. 4.39 for different values of the power «*n*» (Annels 1991)

% Weighting to each grade				Weighted block grade
n	1.6%	3.1%	2.8%	
1	44.8	34.7	20.5	2.37%
2	55.2	33.2	11.6	2.34%
3	64.0	29.9	6.1	2.12%

Table 4.11, the weighting given to the nearest sample (1.6%) increases with «*n*», while that given to the others decreases.

Larger exponents (IDC) are applied where large weights are decided for the closest samples. The extreme case is to increase the value of the exponent so that only the closest sample receives any weight at all, but this selection is a nonsense because then the procedure is equivalent to polygonal method. The opposite extreme occurs where the exponent is zero, which amounts to an equally weighted moving average as described in the previous methods (moving window method).

Techniques such as search using a quadrant or an octant can also optimize the spatial distribution of data utilized to produce a block or point estimate. Where the deposit is considered to be isotropic, in that grade or thickness variations are constant in all directions or the drilling grid is square, a circle (2-D) or a sphere (3-D) is used as search area. But if the deposit is anisotropic, an ellipse (2-D) or an ellipsoid (3-D) is preferred as search area. Another possibility is to divide the search area around the block center to be evaluated into four or more commonly eight sectors and then proceed to search for the nearest specified number of samples in each sector in turn; usually, an eight-point sector search is used, but this can be varied by the user. This means that a maximum of 64 samples would be utilized in an eight-point sector search, although some sectors can reach the set distance limit before eight points are located and thus a smaller data set would be used. This method reduces the bias incurred where denser sampling exits to one side of the block under evaluation. Problems still exist however for blocks at the ore body fringes where some sectors will be totally empty.

As aforementioned, inverse distance weighting is a smoothing technique and as such is unsuited to deposits that have sharply defined boundaries and very sudden drop in grade. In these situations, the methods tend to produce larger tonnages at lower grade than actually exist, which can thus seriously affect the results of any economic feasibility study. Therefore, it is evident that inverse distance weighting works best for mineralization that displays gradual decline in grade across its economic fringes. It is ideal for porphyry deposits, some alluvial or eluvial deposits, and for limestones (Annels 1991).

4.5.6 Geostatistical Methods

Introduction

The classical methods described so far are based on the assumption that the individual samples, such as sample values from drillholes, are statistically independent of each other. In the context of an ore body, this implies that the position from which any sample was taken is not relevant. Theoretically, using classical statistics, taking samples in opposite sides of an ore body would be as good as taking them a short distance apart. This kind of independence is rarely found in mineral deposit data, but there is frequently certain spatial interdependence among the samples, which is studied by geostatistics. Geostatistics is therefore statistics by which the spatial association is taken into consideration and where the variables are known as regionalized variables. Matheron published in 1963 that «Geostatistics, in their most general acceptation, are concerned with the study of the distribution in space of useful values for mining engineers and geologists, such as grade or thickness, including a most important practical application to the problems arising in ore-deposit evaluation… Any ore deposit evaluation as well as proper decision of starting mining operations should be preceded by a geostatistical investigation which may avoid economic failures.» Moreover, classical methods do not include any estimation of the errors involved in the evaluation and this general, being this concept fundamental in any estimation method of mineral resources and reserves. In this sense, geostatistics estimates the error involved in estimation. In Fig. 4.40, two blocks are going to be estimated, one of which by relatively few data (left) and the other by more

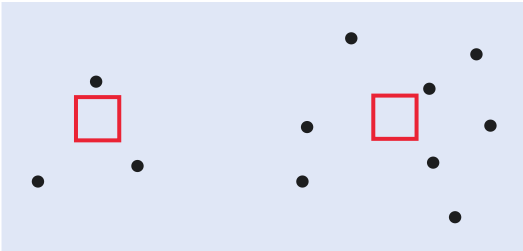


Fig. 4.40 Geostatistics estimates the error of the estimation

abundant data (right). In addition to generating block estimates, in the same way as classical methods, geostatistics computes the error of estimation.

Numerous books are available on the subject, including those by Matheron (1971), David (1977), Journel and Huijbregts (1978), Clark (1979), Isaaks and Srivastava (1989), and Goovaerts (1997). Geostatistics is also applied to other topics in mineral resource exploration/evaluation such as classification of ore reserves based on geoestatistical and economic parameters (Wober and Morgan 1993).

There are two areas where geostatistical calculations can be important, even in the early phases of evaluating a mineral deposit: (a) the calculation of errors or uncertainties in reserve estimates («knowledge of ore grades and ore reserves as well as error estimation of these values is fundamental for mining engineers and mining geologists») (Matheron 1963) and (2) the determination of grades, for instance, in single mining blocks. As a consequence, a geostatistical reserve study with careful attention to geologic controls on mineralization will provide not only a good total reserve estimate but also a more reliable block-by-block reserve inventory with an indication of relative confidence in the block grades estimated. Obviously, geostatistical methods, like any others, cannot increase the quantity of basic sample information available nor they can improve the quality or accuracy of the basic assays. Geostatistical techniques should be regarded as a comprehensive suite of ore reserve estimation tools, which, if they are correctly understood and utilized, should commonly lead to few astonishments where the mine come into production (Readdy et al. 1998). Other advantages of geostatistical methods include determination of the best possible unbiased estimate of grade and tonnage, which is important where an operation is working close to its economic breakeven point, or the assignment of confidence limits and precision to estimates of tonnage and grade.

In general, the geological contest defines the grade and thickness in a deposit. Thus, changing geological and structural conditions produce variations in grade or quality and thickness between deposits, even within one deposit. However, it can be logically considered that samples taken close together tend to reflect probably the same geological conditions. And, as the sample distance increases, the similarity decreases until at some distance there will be no correlation. In this way, geostatistical methods quantify this concept of spatial variability within a deposit and display it in the form of a semivariogram. Once that correlation between samples is established, it can be utilized to estimate values between existing data points. The estimation of the correlation is referred to variogram modeling. Thus, geostatistical methods use the spatial relationship between samples as quantified by the semivariogram to generate weights for the estimation of the unknown point or block.

Matheron developed the basis for geostatistics in the mineral industry during the 1950s and 1960s. As aforementioned, geostatistics is defined as applications of the theory of regionalized variables. They are associated with both a volume and shape, called a «support» in geostatistics (Fig. 4.41), and a position in space. The term regionalized variable also emphasizes the two aspects of the variables: a random aspect which accounts for local variations and a structured aspect which reflects large-scale tendencies

Fig. 4.41 The volume, size, and position in space of a sample is the «support»

of a phenomenon. Geostatistics also assume the stationarity into the mineral deposit. It means, simply, that the statistical distribution of the difference in, for example, grade between pairs of point samples is similar throughout the entire deposit or within separate subareas of the deposit. The concept of stationarity can be difficult to understand but can be associated to the term homogeneity utilized by geologists to characterize domains of similar geologic features such as types of mineralization.

Classical statistic considers only the magnitude of the data and geostatistics takes into account not only the value at a point but also the position of that point within the ore body and in relation to other samples. Of course, geostatistical estimation does not mean necessarily better estimates than those obtained by other methods. In fact, any estimation procedure can produce incorrect results because the procedure has not been applied correctly, inappropriateness of the procedure, or changings in the geologic model as a consequence of new information further obtained (Wellmer 1998). Geostatistics has a clear potential if it is reconciled with the geology of the mineral deposit (King et al. 1982). Thus, it is important to note that geostatistical methods cannot replace meticulous geological data acquisition and interpretation. They are computational tools that rely on good geology and extend its reach. For instance, erroneous application of geostatistics is to calculate a semivariogram with data that comprise distinct domains. For this reason, geostatistical results (kriging) should always be checked with other method such as classical ones. Geostatistical calculations also require suitable computer programs and a considerable mathematical background. However, geologic cross sections, bench plans, and most importantly, the acquired understanding of the ore body in terms of the lithologic, structural, or other controls on the mineralization is of paramount importance in any geostatistical study.

A geostatistical ore reserve study will usually include the following main steps: (1) study of the geologic controls on the grade, thickness, or other variables of the mineralization, (2) computation of experimental semivariograms, (3) selection of suitable semivariogram models to adjust to the experimental semivariograms, and (4) estimation of the variable value and the estimation error from the surrounding sample values using kriging. Geostatistical methods are optimal where data are normally distributed. Therefore, the first step in geostatistical studies is to check the normality of the data distribution. It can be carried out using the methods described in ▶ Sect. 4.2.2. The four numbered steps will be presented here in such a way to minimize the use of mathematical expressions and notation (e.g., triple integrals).

Spatial Correlation: Semivariogram

The amount of spatial correlation or continuity is determined by the primary geostatistical tool: the variogram or semivariogram; there is a clear difference between both terms, but here they will be used indistinctly. The semivariograms, which represent the characteristics of the mineralization, are a prerequisite to any geostatistical ore reserve estimation, and they are used in all subsequent phases. As explained in a previous section, semivariogram defines the concept of «area of influence» and can be used in determining the optimum drillhole spacing and optimum sample size. The semivariogram serves to measure and express the correlation of the variable under consideration in a specific space and at a given orientation. In this method, it is always assumed that variability between two samples depends upon the distance between them and their relative orientation. By definition, this variability (semivariance or $\gamma(h)$) is represented calculating the variance between pair of samples separated by a distance «h» (lag distance), following the formula:

$$\gamma(h) = \frac{1}{2n_{(h)}} \cdot \sum_{i=1}^{n} \left(x_i - x_{i+h}\right)^2$$

where $\gamma(h)$ is the semivariance, x_i are the data values of the regionalized variable (e.g., ore grades), x_{i+h} is the data value at a distance «h» from x_i, and «n» is the total number of value pairs that are included in the comparison; lag (h) is purely the spacing at which the squared differences of sample values are obtained (lag 1 is thus the minimum sampling interval). For n samples regularly distributed along a line, at intervals of «h» meters, we will have $(n - 1)$ pairs to compute $\gamma(h)$, $n - 2$ to compute $\gamma(2h)$, and so on.

The sample pairs are each oriented in the same direction, are each separated by the same distance

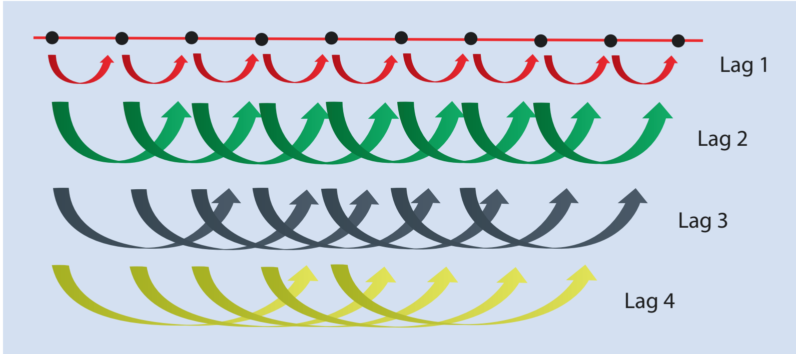


Fig. 4.42 Calculation of $\gamma(h)$ at different lags

(h) in meters, and are equivolume, the support concept commented previously. On the semivariogram, $\gamma(h)$ is plotted as a function of the spacing or lag h, and the result is the so-called experimental or empirical semivariogram, because it is based only of samples. Alternatively, semivariograms can be computed on the logarithms of grade if this variable is logarithmically distributed. A semivariogram is therefore ideally suited for clarifying the problem of whether the sample values are statistically independent of each other or if they are spatially interdependent. Commonly, values of $\gamma(h)$ increase steadily with increasing distance and reach a limiting or plateau level. Figure 4.42 shows how the semivariance or $\gamma(h)$ is calculated.

If sampling density is too low for any underlying correlation to be detected, if the ore body is extremely homogeneous, or if poor sample collecting, preparation, and assaying procedures were used, then no structure or continuity will be visible in the semivariogram. From an operational viewpoint, geostatistical calculation requires a large sample size. With a small number of exploration works, for example 20 drillholes, the calculation of variograms becomes increasingly uncertain, even impossible. At least 30 pairs are necessary for each lag of the experimental semivariogram and that no lag greater than $L/2$ should be accepted, where L is the average width of the data array in the direction for which the semivariogram is being estimated (Journel and Huijbregts 1978).

The examination of the variogram can be used also to determine the nature of mineralization. The uniformity of the ore, the degree to which it has been concentrated by various processes during precipitation of the ore minerals, or remobilized in later metamorphism or secondary enrichment, can be deduced from the study of the semivariogram. An insight is gained into the relative importance of spatial controls (e.g., distance from an igneous contact, presence of faults, or palaeo-shoreline) and random influences (e.g., fracture infillings or metamorphic lateral secretion veins) operating during the mineralization process (Annels 1991).

Semivariogram Models and Fitting

Once an experimental semivariogram has been calculated, it must be interpreted by fitting a model to it (Fig. 4.43). Any function that depends on distance and direction is not necessarily a valid semivariogram. The experimental semivariogram cannot be utilized directly to generate kriging estimates since it is established only for a finite number of lag distances, those used in its construction. After joining its values at such lag distances, the resulting function can not necessarily fulfill the conditions that every semivariogram must meet. In kriging estimation process, a continuous function must be included in the calculations, and since experimental semivariogram is not a function of this type, it is necessary to fit a theoretical model to the experimental semivariogram obtained. In other words, kriging estimation will need access to semivariogram values for lag distances other than those used in the empirical semivariogram. Another reason is to ensure that kriging equations are solvable and kriging estimates have positive kriging variances.

There are several possibilities to select a model to fit but not infinite because strong mathematical constraints exist (concept of a mathematical property called positive definiteness). Fitting a semivariogram model can be done by manual or automatic statistical fitting, usually being a combination of

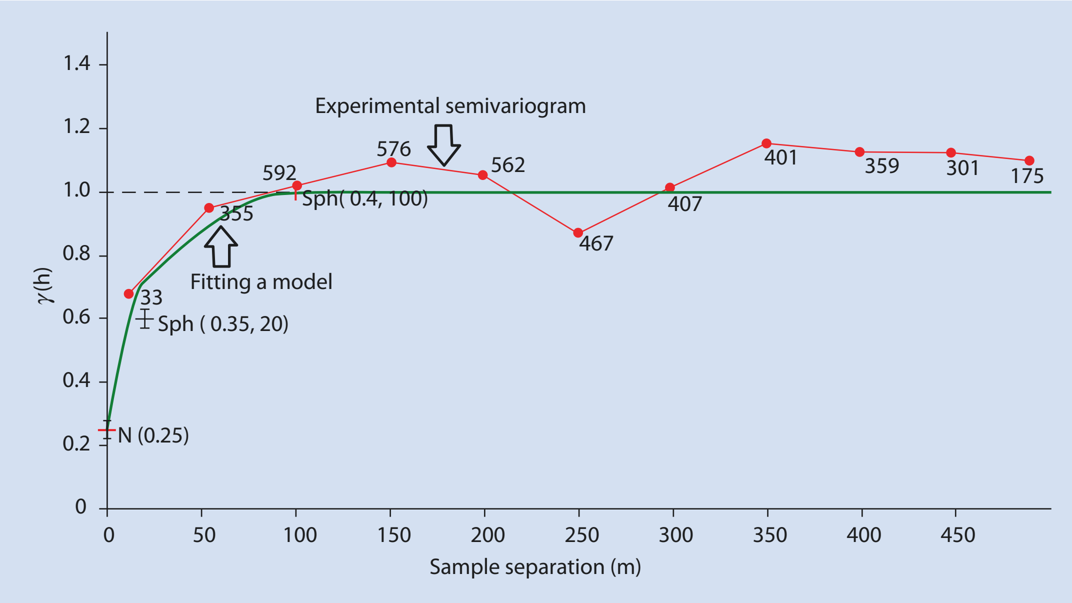


Fig. 4.43 Experimental semivariogram and fitting a model

both the best option. Cross validation is then performed to compare alternative variogram models to fit. Fitting models is not easy for different reasons: (a) the accuracy of the observed semivariances is not constant; (b) the spatial correlation structure is not the same in all directions, that is, anisotropy is commonly present; and (c) the experimental semivariogram can contain much point-to-point fluctuation, among others.

The spherical or Matheron model is the most common type of model used in mining variables, for instance, grade or thickness of the mineralization, although other types do exist such as circular, exponential, linear, Gaussian, or de Wijsian, among others (e.g., Journel and Huijbregts 1978; Annels 1991) (Box 4.4: Spherical Model). From a mathematical point of view, it is possible to combine two or more simple models to fit. In many cases, it is not possible to make an adequate approximation of an experimental semivariogram by a single model. In other words, regionalization can be present at several scales. The use of nested structures or combined models provides enough flexibility to model most combinations of geologic controls. It is important to note that all semivariograms fitted in various directions of a mineral deposit are part of the same model, and all should have the same components, except in the case of a zonal anisotropy.

Kriging

Georges Matheron selects this name for the estimation process because he wanted to recognize the work of D. G. Krige. This author proposed the use of regression after concluding that the polygons method originated overestimation or underestimation of the grades in the estimation results. The kriging method is a geostatistical technique or a group of techniques for determining the best linear unbiased estimator (BLUE) with minimal estimation variance. It is best because it keeps the errors as low as possible; if Z and Z^* are the true and estimated values, respectively, the variance of differences ($Z - Z^*$) for all estimates must be minimized. Linear because kriging calculates the variable as a linear combination of the values of the nearest samples. And unbiased because the estimation process is globally unbiased, but unbiased on average, that is, over the entire data range. Therefore, kriging is the operation of weighting samples in such a way as to minimize errors in the estimation of grades of the deposit.

Kriging generates the estimate at each point or block employing the semivariogram model fitted to the experimental semivariogram. The main problem to be resolved by kriging is to generate the best possible estimate of an unknown point or block from a group of samples. The general term kriging covers several specific methods are such as

Box 4.4

Spherical Model

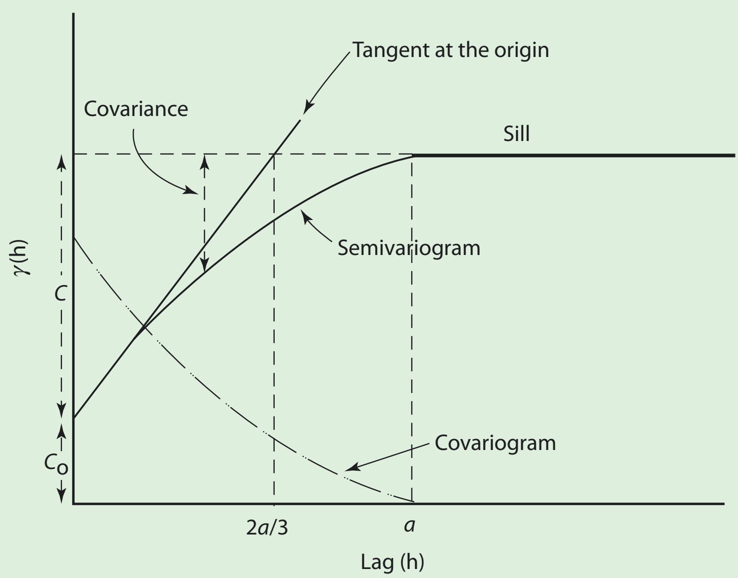
The spherical or Matheron model, and many others, can be described quantitatively by three parameters: (1) range, (2) sill, and (3) nugget effect (Fig. 4.44). Range (a) is the distance at which the semivariogram levels off at its plateau value. This reflects the classical geological concept of an area of influences. Beyond the distance of separation, sample pairs no longer correlate with one another and become independent. Regarding the sill, it is the value at which the variogram function plateaus. For all practical purposes, the sill is equal to the variance of all samples used to compute the semivariogram. As a general rule, the semivariogram model starts at zero on both axes; at zero separation ($h = 0$) there should be no variance. Even at relatively close spacings, there are small differences and variability increases with separation distance. This is seen on the semivariogram model where a rapid rate of change in variability is marked by a steep gradient until a point where the rate of change decreases and the gradient is zero. Beyond this point, sample values are independent and show variability equal to the theoretical variance of sample values. This variability is termed the sill (C) of the semivariogram. The sum of the nugget effect plus the sill is known as the total sill value ($C + C_0$).

The third characteristic considered is the nugget effect. The semivariogram value at zero separation must be zero, but there is often a discontinuity near the origin, which is called the nugget effect (C_0). It expresses the local homogeneity or lack thereof of the deposit. This is generally attributable to differences in sample values over very small distances and can include inaccuracies in sampling and assaying (this item sometimes is called «human» nugget effect) as well as associated random errors. If the semivariogram shows random fluctuation about a horizontal line, a so-called pure nugget effect is present in the ore body. In that case, the best option is to evaluate the deposit using classical methods since errors estimating the reserves of an ore deposit with pure nugget effect in the semivariogram can be huge.

The three parameters mentioned (range, nugget effect, and sill) characterize each type of mineral deposit. Very irregular deposits, such as gold or pegmatite deposits, will show high nugget effect and/or small range; relatively uniform deposits such as stratiform, sedimentary Pb–Zn occurrences show low, even zero, nugget effect and large range. Information from other deposits of the same type, preferably neighboring deposits or deposits in the same geological region, can help as a priori information, for example, for the estimation of the range or the relative nugget effect, if in the early stage of the exploration only limited data were available to calculate the variogram (Wellmer 1998). Semivariograms in different orientations can also identify the presence of anisotropic features in mineral deposits. Anisotropic features are reflected by the range and sill, which are dependent on the orientation; the nugget effect is generally an isotropic quantity.

The spherical model has the mathematical form shown in the two equations below:

Fig. 4.44 Spherical model



$$\gamma(h) = C_0 + C\left(\frac{3h}{2a} - \frac{1(h)^3}{2(a)^3}\right)(for\, h < a)$$

$$\gamma(h) = C_0 + C \quad (for\, h > a)$$

Some semivariogram phenomena in the spherical model can appear, such as proportionality effect, drift, directional anisotropism, or hole effect. If a deposit is very large, then it is perhaps unrealistic to assume constant spatial variation so that it is necessary to divide the deposit up into subareas or levels provided that there are still enough samples in each. Each subarea or level will get a semivariogram with a different sill: this is the proportionality effect. Regarding the drift, an assumption made in geostatistics is that no significant statistical trends occur within the deposit, which would cause a breakdown in stationary, but sometimes such a statistical trend can be present and the sill value increases over a specific distance (drift); since the drift used to be at distances beyond the range, it will not interfere with local estimations of the deposit. In respect of the cited anisotropy, this occurs where different semivariograms are obtained for different directions in an ore body, and this means that an elliptical zone of influence exists. Anisotropism is especially marked, for example, in alluvial deposits where the range across the deposit is short compared to that parallel to its length. Finally, hole effect can be recognized when areas of high-grade mineralization alternate with areas containing low values. The result is a pseudo-periodicity which is reflected by an oscillation of the semivariogram about the apparent sill level. This effect can be easily confused with the usual erratic oscillation of the semivariogram about the sill value for lag values greater than the range.

simple kriging (SK), ordinary kriging (OK), indicator kriging (IK), universal kriging (UK), and probability kriging (PK), among others. In kriging, the coefficients of such a linear combination are obtained indirectly from the semivariogram, hence the importance of trying to fit correctly the model of semivariogram. Unlike other estimation methods (e.g., inverse distance weighing or nearest point), kriging also gives a confidence level of each estimate.

The kriging estimator has the following general form, for instance, to estimate a grade value in a point:

$$Z^* = \sum_{i=1}^{n} \lambda_i x_i = \lambda_1 x_1 + \lambda_2 x_2 + \lambda_3 x_3 + \ldots + \lambda_n x_n$$

where Z^* is the estimated grade, X_i is the sample grade, λ_i is the weighting coefficient assigned to each respective Xi, and «n» is the selected number of nearest neighbor samples that will be used to estimate the grade. The suitable weights λ_i assigned to each sample are determined by two conditions. The first one expresses that z^* and z must have the same average value within the whole large field and is written as $\Sigma\lambda_i = 1$. The second condition expresses that the λ_i have such values that estimation variance of z by z^*, in other words, the kriging variance should take the smallest possible value (Matheron 1963). In minimizing this estimation error or variance, kriging results in a series of simultaneous equations, which can be solved for each weighting factor, given the position of the sample and a model of the semivariogram representative of the mineralization being studied. The estimation errors in the process will be higher in regions of low drilling density and obviously lower where the deposit has been extensively drilled with closer spaced holes.

The system of linear equations (system of ordinary kriging equations) is set up as follows:

$$\sum_{j=1}^{m} \lambda_j \gamma_{ij} + \mu = \gamma_{i0} \qquad i = 1, 2, \ldots m$$

where «I» and «j» are data locations and «m» is the number of data used in the estimation. The solution of the $m + 1$ linear equations, including $\Sigma\lambda_i = 1$, minimizes the variance of the estimation error. Thus, the essence of ordinary kriging is that the estimation variance is minimized under the condition that the sum of the weights is 1. In the kriging system, «μ» is a Lagrange multiplier needed for the final solution, γ_{ij} are known semivariogram values from the semivariogram function estimated between data points «I» and «j», and γ_{i0} are known semivariogram values between data points «I» and the estimated location x_0 and y_0, if 2-D.

For instance, in a four-sample kriging estimation, the full set of kriging equations is the following, being K equal to λ in the previous formula:

$$\begin{aligned} K_1\gamma_{1,1} + K_2\gamma_{1,2} + K_3\gamma_{1,3} + K_4\gamma_{1,4} + \mu &= \gamma_{0,1} \\ K_1\gamma_{2,1} + K_2\gamma_{2,2} + K_3\gamma_{2,3} + K_4\gamma_{2,4} + \mu &= \gamma_{0,2} \\ K_1\gamma_{3,1} + K_2\gamma_{3,2} + K_3\gamma_{3,3} + K_4\gamma_{3,4} + \mu &= \gamma_{0,3} \\ K_1\gamma_{4,1} + K_2\gamma_{4,2} + K_3\gamma_{4,3} + K_4\gamma_{4,4} + \mu &= \gamma_{0,4} \\ K_1 + K_2 + K_3 + K_4 &= 1 \end{aligned}$$

In addition to the estimate, the kriging variance $\sigma^2{}_{\mathrm{E}}$ or $\sigma^2{}_{\mathrm{K}}$ is found from

$$\sigma_{\mathrm{E}}^2 = \sum_{i=1}^{m} \lambda_i \gamma_{i0} + \mu$$

Kriging variance depends on the distance of samples used to estimate the point or block value. Thus, a lower kriging variance means a point or block that is estimated from a near set of samples and a higher kriging variance represents a point or block that is calculated using samples some distance away. Having computed a reliable group of regular data values using kriging, these values can be contoured and showed graphically as well as the corresponding estimation variances (Fig. 4.45). Thus, areas with comparatively high estimation variances can be analyzed to see whether there are data errors or if further drilling is needed to diminish the value of the estimation variance. This is one of the most important applications of point kriging.

Once kriging variance is determined, it is possible to calculate the precision with which it is possible to know the various properties of the deposit investigated by obtaining confidence limits (σ_{E}) for critical parameters. According the features of geostatistics commented previously, the errors show a normal distribution, which allows the 95% confidence limit ($\pm 2\sigma_{\mathrm{E}}$) to be calculated. Another application of kriging variance can be the classification of reserves according their levels of uncertainty and precision, the latter based on the relative kriging standard deviation (Diehl and David 1982).

The general procedure of kriging contains a number of important implications that are not particularly obvious to those with limited mathematical background. Some of them are:

1. Kriging is correct on average although any single comparison of a kriging estimate with a true value might show a large difference; however, on average such differences generally are less for kriging than for other interpolation techniques.
2. Kriging of a location (point) for which information is included in the kriging equations results in a kriging estimate equivalent to the known data value; in other words, kriging reproduces existing data exactly.

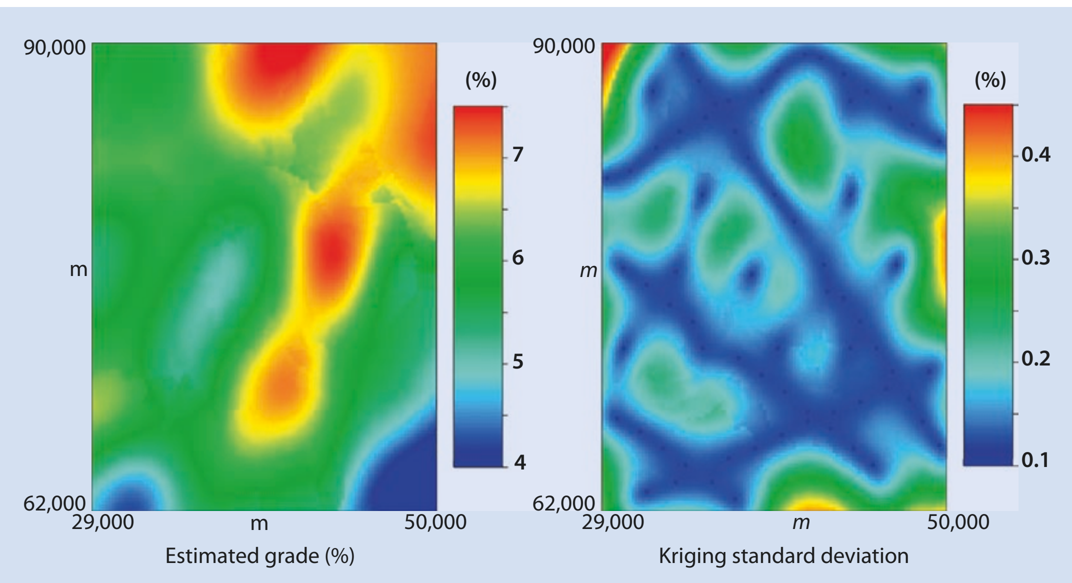


Fig. 4.45 Contoured kriging and kriging standard deviation estimates

3. Kriging takes into account data redundancy; in the extreme, a very tight cluster of several analyses carries almost the same weight as a single datum at the centroid of the cluster.
4. Kriging can be carried out as described but on transformed data; if the transform function is not linear, the back transform will not produce an optimum estimator (Sinclair and Blackwell 2002).

In lognormal distributions, kriging is carried out using log-transformed data. These lognormal distributions are very common when using geochemical variables, for instance, gold values. Thus, the value estimated is the mean log-transformed value, the back transform of which is the geometric mean. But in lognormal distributions, the geometric mean is commonly lower than the arithmetic mean. It should be therefore borne in mind that the arithmetic mean and associated error dispersion must be calculated from the estimates of log parameters.

Semivariogram modeling process and the need of a high-quality model fitted to the experimental semivariogram are of paramount importance. Thus, an almost perfect semivariogram model must be integrated with the geologic model of the mineral deposit. Once the semivariogram model is determined, the subsequent processes are (1) cross validation of the semivariogram model, (2) criteria to select data for individual point or block estimates, and (3) definition of minimum and maximum numbers of data for kriging each point or block. Finally, a systematic kriging of each point or block is carried out.

Regarding the selection of data, generally all data within a special search radius of the point or block being evaluated are selected. The search volume can be spherical or ellipsoidal, if anisotropy is present. A maximum number of data is imposed on each point or block estimate so that the set of kriging equations is relatively small and its solution is efficient. In addition, a minimum number of data is usually established with the objective to prevent large errors in which only local stationarity is ensured and guarantee interpolation as opposed to extrapolation. It is appropriate to need that data be fairly well spread spatially, not all clustered.

For a particular data density, a search radius too small results in too few data being selected. On the contrary, a search radius too large originates huge amount of data being elected, with the result that computation time is clearly increased. Sometimes, just less than the range of the semivariogram can be a good option to select the search radius, since beyond this distance sample pairs no longer correlate with one another and become independent.

Point Kriging and Block Kriging

Point kriging takes into account only relationships between individual sample points, which were drillhole sites in the previous example, but does not take the size of the blocks into consideration. This technique is then best suited to contour isolines of equal grades or thicknesses of the ore body. With regard to block kriging, it estimates the value of a block from surrounding data. Block kriging can therefore replace such techniques as inverse distance weighing or cross sections to evaluate the reserves of a mineral deposit. The estimation block selected initially should have dimensions consistent with the anisotropy of the deposit, the geological model, the grid size, and the area of influence.

To determine the covariance between a sample and a block, the block is considered to be represented by a grid of «*n*» points. Thus, covariance between each of these points and the sample is determined and the average computed. The grid size could be 10 × 10 and the estimation would be the mean of 100 values. Block kriging amounts to estimating the individual discretization points (e.g., 10 × 10) and then average them to obtain the block value. This formulation was originally the most widely used form of kriging in mining applications.

As a general rule, it is not prudent to compute blocks whose dimensions are less than half the sample spacing. As they diminish in size, such blocks become too numerous and the estimation grades quickly become meaningless as they become less and less related to the sample information. Moreover, the variance of such blocks will be excessive, in inverse ratio to size. Regarding the shape of the block, an appropriate shape may be cubic blocks for an isotropic mass and parallelepipeds with proportions related to the dimensions of the zone of influence.

Indicator Kriging

Indicator kriging (IK) was introduced in the early 1980s as a technique in mineral resource estimation (Journel 1983). It is the prime nonlinear geostatistical technique used today in the mineral industry. The original appeal of IK was that it was nonparametric, in the sense they do not make any prior assumption about the distribution being estimated. IK involves transformation of data to zeros or ones based on the situation of a value relative to an assigned threshold. The binomial coding of data into either zero or one, depending upon its relationship to a cutoff value, Z_k, is given, for a value $Z(x)$:

$$i(x;z_k) = \begin{cases} 1 & \text{if} \quad z(x) \geq z_k \\ 0 & \text{if} \quad z(x) < z_k \end{cases}$$

IK has the potential to generate recoverable resources where carried out over a larger area for a series of blocks. In other words, the proportion of a block theoretically is available above a given cutoff grade (an arbitrary threshold called the indicator threshold or indicator cutoff). Thus, if the observed grade is less than the cutoff grade, indicator will be 1. Otherwise, it will be 0. Therefore, indicator kriging is simply the use of kriging to estimate a variable that has been transformed into an indicator variable. Obviously, the indicator variable changes as the variable (e.g., cutoff grade) changes. IK is really a procedure that avoids the influence of the high samples over the whole of the deposit rather than applying it only to the estimation blocks close to these very high grades. The technique is particularly applicable where strict ore/waste boundaries exist within giving blocks, for example, large copper porphyries where grade zoning is the major control as well as in low-grade deposits where the cutoff value is of major concern.

Such applications of indicator kriging have found an extensive utilization in mineral deposit estimation due to their simpleness. This indicator kriging method has been utilized for estimating relative proportions of mineralized versus unmineralized ground, the proportion of barren dykes within a mineralized zone (Sinclair et al. 1993), to delineate different lithological units of an ore deposit (Rao and Narayana 2015), and so on. Repeated indicator kriging for different thresholds, a process known as multiple indicator kriging (MIK), allows the local cumulative curve to be estimated. Thus, the local mean can be determined, a block distribution can be estimated, and the proportion of blocks above cutoff grade, and their average grade, can be calculated. MIK is broadly used to apparently erratic values, such as those usual in most gold and uranium deposits (Sinclair and Blackwell 2002). New applications of MIK include to infer the variogram for the median of the input data and to use this for all cutoffs. This so-called median IK approach is very quick because the kriging weights do not rely on the cutoff being considered (Ali Akbar 2015). Median indicator kriging is achieving growing acceptance as practical and cost effective method for resource estimation and grade control.

Cokriging

Cokriging is a method of estimation that obtains the value of a variable evaluated in a point in space based on the neighboring values of one or several other variables. For example, gold grades can be estimated from a combination of gold and copper samples values. The equations used in cokriging are basically the same as for simple kriging, but considering the direct and cross covariances. The utilization of a secondary variable that is commonly more regular is clearly an interesting advantage over ordinary kriging. It allows estimation of unknown points using both variables globally for the mean of all estimates but also conditionally for the estimates within individually specified grade categories. This can aid in minimizing the error variance of the estimation. To perform cokriging, it is necessary to model not only the variograms of the primary and secondary data but also the cross-variogram between the primary and secondary data. If the secondary variables are present or available, then the use of these secondary variables via the cokriging technique could be advantageous in estimating values of the primary variable, although sometimes the improvement of cokriging is very little or none (Genton and Kleiber 2015).

Cross Validation

Different models can fit the same experimental data, so it is natural to control which is the best model. The best method to select the adequate semivariogram model for kriging is the so-called cross-validation process. It estimates the value at each drillhole or sample location after removing the observed value by kriging all those adjacent values that fall in the search area around this point. Therefore, not only the known value (Z) but also the estimated value (Z^*) are computed, and the experimental error ($Z - Z^*$) can be estimated as well as the theoretical estimation variance.

The best variogram model would be the one that yields lower average error. In summary, cross validation is to krige known values to obtain the best semivariogram to krige in unknown points or blocks. If the number of samples is great, cross-validation techniques can be used to see if the method applied or the model fitted to the experimental variogram is acceptable or can be improved. However, in the early exploration stages there are rarely enough data to do a meaningful cross-validation computation.

Cross-validation process can be performed in two distinct ways: (1) a spatial leave-one-out reestimation whereby one sample at a time is removed from the data set and reestimated from the remaining data and (2) a subset of the data (e.g., 20 or 30% of the total) being separated completely from the data set and reestimated utilizing the rest of the data. The first method is the most commonly used, but there have been a number of objections to this option: (a) the method is generally not sensitive enough to detect minor differences from one variogram model to the next; (b) the analysis is performed on samples or composites but not in a different volume support; (c) the sill of the semivariogram cannot be cross-validated from the reestimation; (d) semivariogram values smaller than the minimum lag between samples cannot be cross-validated (Isaaks and Srivastava 1989). Therefore, it is difficult to define a useful goodness of fit test for a semivariogram model. Many times the most important factor to select the best semivariogram model is the user's experience and the goals of the project.

As previously commented, in cross validation each drillhole or sample has both an observed value and a kriged estimate for the regionalized variable at this point. Thus, final outputs in the cross-validation process are the following (Annels 1991):

1. Mean algebraic error:

$$\frac{\sum_{i=1}^{N}\left(Z_i - Z_i^*\right)}{N}$$

where Z_i is the actual value at each point and N is the number of points. This calculation takes into account the sign of $(Z - Z^*)$.

2. Mean absolute error:

$$\frac{\sum_{i=1}^{N}\left|Z_i - Z_i^*\right|}{N}$$

This is the mean of the differences, but this time the sign is ignored.

3. Mean kriging variance:

$$\frac{\sum_{i=1}^{N}\sigma_K^2}{N}$$

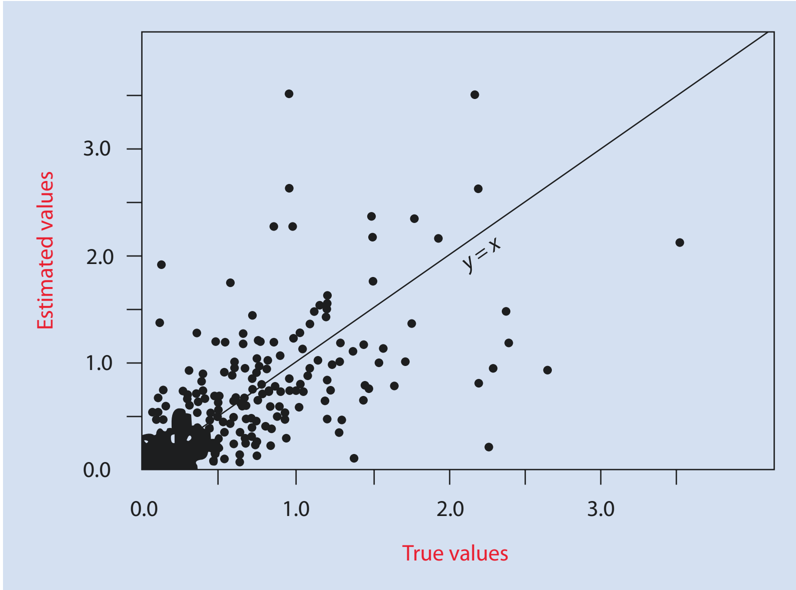
4. Mean square error of estimation:

$$\frac{\sum_{i=1}^{N}\left(Z_i - Z_i^*\right)^2}{N}$$

5. Number of points valued by point kriging.

If the model allows accurate estimation of the data population, the value of (1) approaches zero and is not more than 1% of Z (the mean of all the Z_i values), (4) should be almost equal to (3), and (5) should be as large as possible. A significant difference between (3) and (4) can be due to outliers, for example, abnormally high or low values in the data set, which greatly increase the $(\text{difference})^2$ values between these and adjacent points. Removal of these outliers can allow the mean squared differences value to approach the mean point kriging variance. Another way to test the semivariogram model is plotting Z against Z^* and if the values are uniformly distributed about a best fit regression line whose slope is 45° (the values show a high correlation coefficient, near 1), then the conditional unbiasedness has been achieved (Fig. 4.46). The following Box is an example of using geostatistical methods in mineral deposit evaluation (Box 4.5: Amulsar Deposit Evaluation).

Fig. 4.46 Testing the semivariogram model selected using cross validation (plotting Z – true value against Z^* – estimated value)



Box 4.5

Amulsar Deposit Evaluation: Courtesy of Lydian International Ltd.

The Amulsar Gold Project is located in south-central Armenia approximately 170 km southeast of the capital Yerevan and covers an area of approximately 56 km^2. The Amulsar gold deposit is situated on a ridge in south-central Armenia and is hosted in an Upper Eocene to Lower Oligocene calc-alkaline magmatic-arc system that extends northwest through southern Georgia, into Turkey, and southeast into the Alborz-Arc of Iran. Volcanic and volcano-sedimentary rocks of this system comprise a mixed marine and terrigenous sequence that developed as a nearshore continental arc between the southern margin of the Eurasian Plate and the northern limit of the Neo-Tethyan Ocean.

The geology of the Amulsar deposit area consists of mainly porphyritic andesites with strong argillic alteration forming strata-parallel panels with typical thicknesses of 20–100 m. Interleaved with these rocks are silicified volcano-sedimentary rocks that host gold and silver mineralization. The strong stratiform control on the location of the base of the silicified volcano-sedimentary rocks has given rise to the mapping definition of upper volcanics and lower volcanics representing silicified volcano-sedimentary and altered andesites rock units, respectively. The division into upper volcanics and lower volcanics is also based on alteration and structural position. The Amulsar project is a high-sulfidation epithermal deposit, but its close association with syndepositional deformation adds a signature characteristic of orogenic gold systems. The deposit also has some characteristics of low-temperature variants of IOGC deposits.

The resource database used to evaluate the mineral resources for the Amulsar project comprise an Excel spreadsheet database updated with drilling completed after the previous resource estimate. These spreadsheets contained all information for diamond core and reverse circulation drillholes and channel samples for the project. The database consists of 1,298 drillholes and channel samples collected in exploration work undertaken between 2007 and 2013. The data is comprised of 315 diamond drillholes (41,819 m), 512 reverse circulation drillholes (73,543 m), and 358 channel samples (1,337 m). The Amulsar deposit has a complex history of structural events, including east- and west-directed thrusting and related complex deformation, and two episodes of extensional faulting within large northeast-trending grabens. This has resulted in a complex of structurally positioned blocks of upper volcanic (UV) and lower volcanic (LV) rocks. Mineralization is predominantly confined to rocks of the UV zone. The LV zone is generally not mineralized, except near contacts with mineralized UV rocks or related mineralized structures. Based on a major structural break, UV rocks have been subdivided into a northern Erato zone and a southern Tigranes-Artavasdes-Arshak (TAA) zone (Fig. 4.47).

The drillhole and chip sample database used for estimation of resources consists of 106,038

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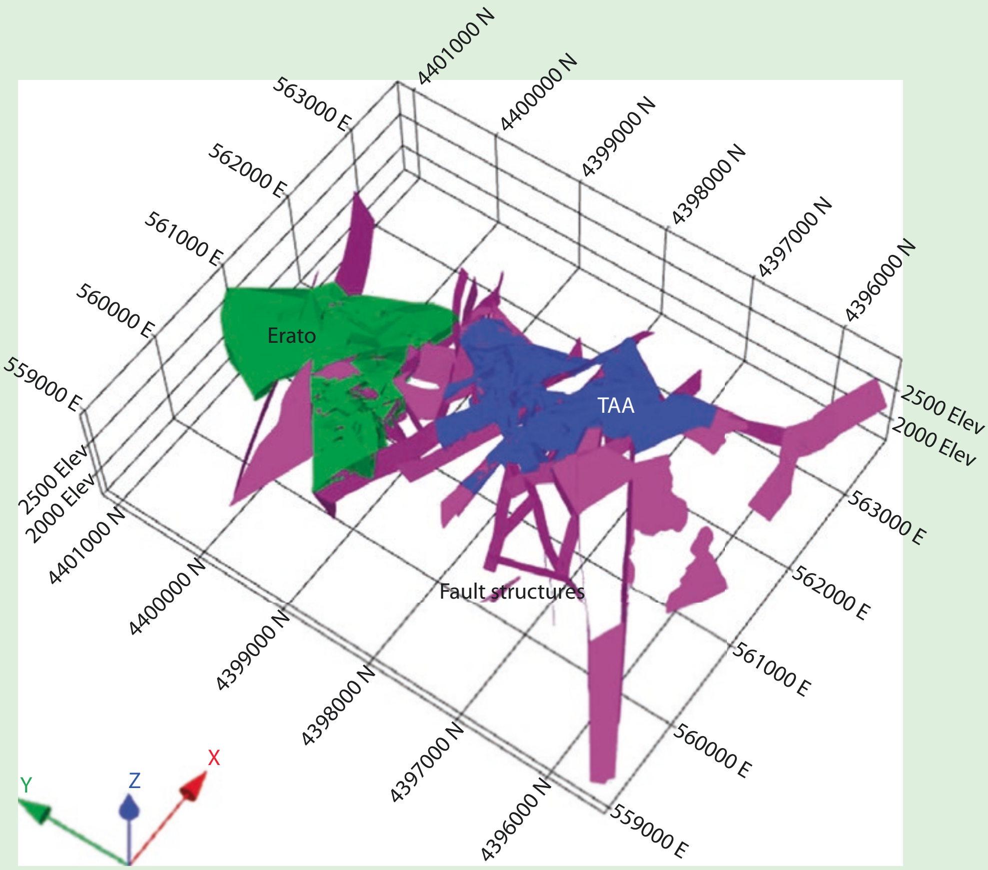

■ **Fig. 4.47** Wireframe models for Amulsar deposit and interpreted faults (Illustration courtesy of Lydian International Ltd.)

gold assays and 101,038 silver assays, and 1,198 dry bulk density measurements. The drillhole database excludes 92 geotechnical, metallurgical, and condemnation drillholes which were not assayed for gold and silver or were not assayed using the same techniques used for all other samples (i.e., metallurgical boreholes). In addition, eight drillholes within the mineralization areas were excluded or partially excluded as all or part of the drillholes were not sampled or drillholes were abandoned due to drilling problems. Drillholes for each of the two UV zones comprising Erato and TAA, and a single LV zone covering the rest of deposit volume, were composited at 2 m intervals to provide common support for statistical analysis and estimation for gold and silver grades. Approximately 99 percent of assay samples were sampled at 2 m intervals or less. Capping of high gold and silver grades for the Erato, TAA, and LV zones is not required.

Conditional statistics were generated for the Erato and TAA zones using gold composites and were used to determine intraclass mean grades to be used for post-processing of model panel grade estimates. Seventeen and sixteen indicator thresholds were selected for Erato and TAA zones, respectively. These indicators were considered sufficient to discretize both the composite and metal values. The selected thresholds represent the entire grade range and therefore represent the spatial variability of the mineralization. A suite of gold variograms were generated and modeled for the Erato-LV and TAA-LV declustered composites; variograms were generated for gold and indicator thresholds. Traditional semivariograms were used as the spatial model for Erato and TAA zones. Gold indicator variograms were used to estimate gold grades. Gaussian-transformed gold variograms were developed for variogram analysis and were back transformed to gold values to derive change-of-support correction factors and for the selective mining unit (SMU) localization of the MIK estimates. Gaussian-transformed omnidirectional variogram models were generated for LV zone gold composites and silver composites for Erato, TAA, and LV zones (■ Fig. 4.48). Some examples of the

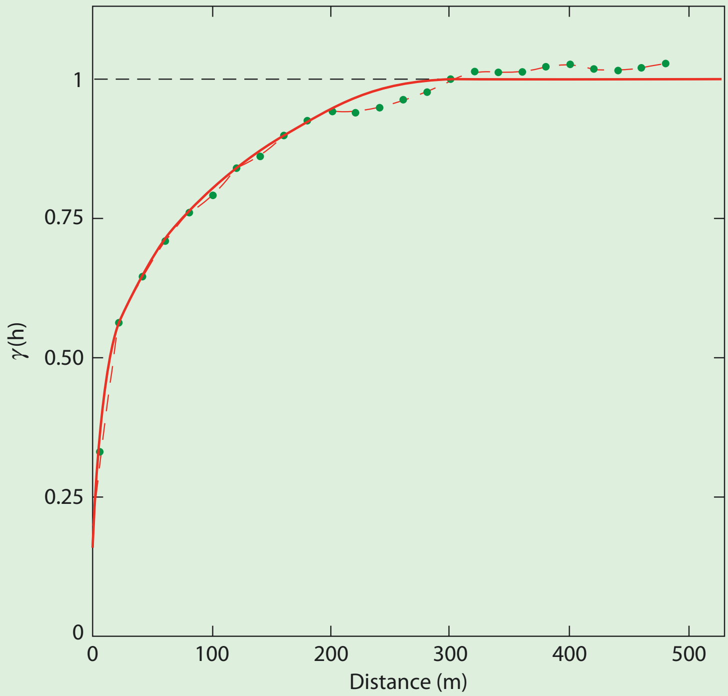

Fig. 4.48 Silver variogram model for LV zone (Illustration courtesy of Lydian International Ltd.)

Table 4.12 Some variogram models used in the project

Variable	Zone	C_0	CC	Structure model	*Rx* (m)	*Ry* (m)	*Rz* (m)
Au	Erato	0.0886	0.0783	Spherical	15	15	15
			0.0880	Spherical	60	60	60
			0.0740	Spherical	160	160	160
Au	TAA	0.3715	0.2961	Exponential	15	15	15
			0.2334	Spherical	57	57	57
			0.0990	Spherical	205	205	205
Au	LV	0.1800	0.4700	Exponential	30	30	30
			0.3500	Spherical	265	265	265

Data courtesy of Lydian International Ltd.

variogram models for the project are provided in Table 4.12.

Erato and TAA zone composites were used to estimate gold into each of the Erato and TAA models using hard boundaries. A panel model with the dimensions of 20 mE × 20 mN × 10 m elevation was used for the MIK estimates. In preparation for ranking of localized estimates, gold grades were estimated by ordinary kriging (OK) into a target SMU model with the dimensions 10 mN × 10 m E × 5 m elevation. Hard boundaries were used for each respective zone to estimate gold grades into the Erato and TAA block models (Table 4.13). A change-of-support adjustment was applied in order to produce resource estimates that reflect the anticipated level of mining selectivity. When estimating local recoverable resources, the objective is to obtain the proportion of mineralization above a particular cutoff grade (pseudo tonnage), within panels that are large enough to achieve a robust estimation.

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Table 4.13 Block model definition

Model	Coordinate	Origin (m)	Block size (m)	No. of blocks
SMU	Northing	559,600	10	312
	Easting	4,396,300	10	492
	Elevation	2,270	5	146
Panel	Northing	559,600	20	156
	Easting	4,396,300	20	246
	Elevation	2,270	10	73

Data courtesy of Lydian International Ltd.

A localized MIK (LMIK) SMU model was generated using the MIK SMU-corrected histogram and partitioning of the estimated tonnage and metal from the MIK panel model evenly into SMU blocks within the panel. In this manner, grades are mapped into each of the SMU-sized blocks, thereby replicating the targeted mining selectivity. Gold grades were estimated by ordinary kriging for the LV unit using hard boundaries. No distinction was made between Erato and TAA areas for these estimates. Silver grades were estimated using OK for the Erato, TAA, and LV zones using silver composites with hard boundaries for each zone. Uncapped composites are used for estimation of silver grades in the Erato and TAA models. Silver grades were estimated using an OK estimator. Dry bulk density values were assigned to each estimated model on the basis of the average dry bulk density measurements in each of the estimated zones.

Indicated resources were classified on the basis of a volume that enclosed relatively closely spaced drilling (approximately 45 m intervals) and included holes drilled vertically and at inclined angles, demonstrating vertical and horizontal continuity. The outline was drawn to enclose a continuous zone of mineralization and areas where a high number of composites are used to make each block estimate. These outlines were designed around areas that showed lateral continuity exceeding 150 meters. Indicated classification was extended to include overlying or underlying blocks of the lower volcanic unit. Resources classified as measured were contained within the indicated wireframe, but block grades are estimated by 40 composites and 60 composites for the Erato and TAA zones, respectively. The measured classification encompassed only blocks in the Erato or TAA zones. The likelihood of the resource being potentially economic was determined by generating a conceptual optimized pit shell using the following assumptions: (a) metal prices of USD 1,500 per ounce gold and USD 25 per ounce silver, (b) average pit slope of 32 degrees, (c) average mining cost of USD 2.00 per ton and processing and administration costs USD 4.60 per ton, and (d) gold cutoff grade of 0.20 g/t. Mineral resources are reported on the basis of all estimated blocks that are contained within this pit shell.

At a cutoff grade of 0.20 g/t gold, the mineral resources are estimated at 77.2 Mt at 0.78 g/t Au and 3.6 g/t Ag (1.9 million ounces gold and 8.8 million ounces silver) of measured category, 45.1 Mt at 0.76 g/t Au 3.5 g/t Ag (1.1 million ounces gold and 5.1 million ounces of silver) of indicated category, and 106.2 Mt at 0.59 g/t Au and 2.6 g/t Ag (2.0 million ounces of gold and 8.9 million ounces of silver) of inferred category resources (Table 4.14).

Regarding the mineral reserves of the project, the pit designs and the estimate of mineral reserves were based on a number of pit optimization runs carried out utilizing the Lerchs and Grossmann algorithm. These optimization runs examined the effect of:

1. Cutoff grade ranging from 0.1 Au g/t to 0.3 Au g/t, in increments of 0.05
2. A 6.5 percent ramp gradient
3. The inclusion of Inferred material
4. Waste haulage options, exploring the effect of a reduction in mining cost due to utilizing a combination of waste dump and in-pit waste backfill
5. Optimizing each deposit separately
6. The effect of sterilization due to a zone containing an endangered flora
7. the effect of applying dilution by regularizing the resource model
8. The sensitivity of the resource block model considering only gold compared to including the contribution of silver

Figure 4.49 shows the optimization results by pit shell for all deposits and Table 4.15 tabulates the mineral reserves for the project.

Table 4.14 Mineral resource statement

Classification	Quantity tonnes	Gold grade, g/t	Silver grade, g/t	Contained gold, toz	Contained silver, toz
Measured	77,200,000	0.78	3.6	1,940,000	8,810,000
Indicated	45,100,000	0.76	3.5	1,100,000	5,120,000
Inferred	106,200,000	0.59	2.6	2,010,000	8,980,000
Total measured and indicated	122,400,000	0.77	3.5	3,030,000	13,930,000
Total inferred	106,200,000	0.59	2.6	2,010,000	8,980,000

Data courtesy of Lydian International Ltd.

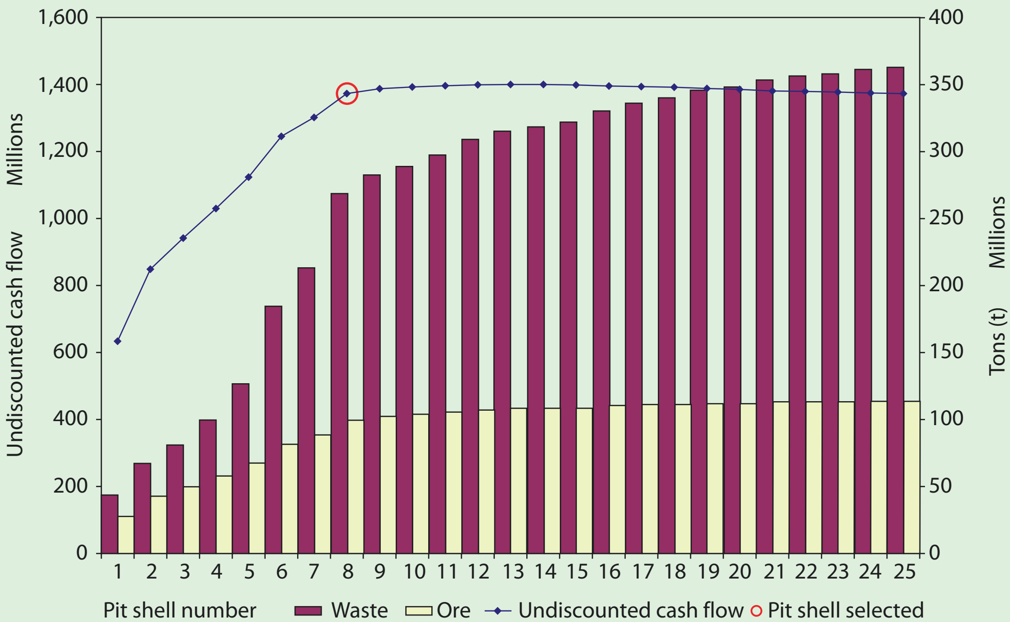

Fig. 4.49 Optimization results by pit shell for all deposits (Data courtesy of Lydian International Ltd.)

Table 4.15 Mineral reserves for the project

	Ore (Mt)	Au grade (g/t)	Ag grade (g/t)	Au metal (koz)	Ag metal (koz)	Waste (Mt)	Stripping ratio (W:O)
Proven	72.9	0.77	3.6	1,816	8,436		
Probable	28.9	0.77	3.7	712	3,481		
Proven + probable	101.8	0.77	3.6	2,529	11,917	284.8	2.80

Lydian International Ltd.

4.6 Mining Project Evaluation

Project evaluation is the process of identifying the economic feasibility of a project that requires a capital investment and making the investment decision (Torries 1998). Much care and perhaps multiple evaluation methods are required to obtain results on which to base mineral investment decisions. Mineral investments show certain characteristics that differentiate them from other types of investment opportunities such as the depletable nature of the ore reserves, the unique location of the deposit, the existence of many geologic uncertainties, the significant time needed to place a mineral deposit into production, the commonly long-lived nature of the operation itself, and the strong cyclical nature of mineral prices. This decrease in flexibility obviously increment the risk of mining projects compared to other types of investment opportunities. The term risk has many meanings in the mining world, but a broad definition of risk is «the effect of uncertainty on objectives» (ISO 31000: 2009. Risk management – Principles and guidelines). It can be used by any organization regardless of its size, activity, or sector.

Deeper in the subject, Rudenno (2012) selects up to seven differences between resource and industrial companies:

1. Volatility of share prices: share price volatility for resource stocks has historically been greater than for industrials.
2. Exploration: a unique feature of the mining industry is the need to explore in order to find and define an economic resource on which a mining project can be built.
3. Finite reserves: any mineral resource has a finite volume and therefore will have a finite life; industrial companies are in theory able to operate for an indefinite length of time, once they have a raw material supply and a market for their product.
4. Commodity price volatility: resource stocks are exposed to greater external commodity price volatility than most industrial stocks, since most of the world's major exporters of raw mineral commodities are price takers rather than price makers.
5. Capital intensity: the mining industry, by its very nature, is capital intensive, being the high level of expenditure due to exploration, economies of scale, isolation, and power and water factors.
6. Environmental: protection of the environment is important for both industrial and resource companies, but mining cycle environmental impacts (see ► Chap. 7) are clearly more intensive and harmful in mining projects.
7. Land rights: although industrial-based companies can be faced with problems related to land rights, they are not as exposed as mining companies, which are often involved in exploration on land not covered under freehold title.

Moreover, the effects of time greatly influence the value of a mineral project as they do any other long-lived investment because many mineral procedures are cyclical and the issues to forecast prices and expenditures poses special problems in calculating and planning mineral projects (Labys 1992). Time also affects mineral projects in several ways that are not always present in other investment opportunities. For example, the first higher-grade ore mined increases early profits but diminishes the average grade of the rest of the ore, thus reducing the global life of the mine. Moreover, it is impossible to establish the right amount or grade of material to be mined until the deposit is depleted. This is related to geologic certainty (only statistical estimates of the reserves) and economic certainty (it is almost impossible to determine reserves since future prices cannot be forecast accurately) (Torries 1998).

In summary, the use of adequate project evaluation techniques is more important in the mining industry than in many other industries. This is because the mining projects are extremely capital intensive and require many years of production before a positive cash flow commences and their life is much longer compared to other industries.

It is important to keep in mind the dynamic nature of project evaluation. Numerous projects compete for the same scarce resources at any given time. Changes in the budget, evaluation criteria, or costs or benefits of any of the competing projects can change the evaluation results and ranking for any single project under consideration.

4.6.1 Types of Studies

Three levels of geological/engineering/economic studies are commonly applied by the mining industry: the scoping study, the preliminary feasibility (pre-feasibility) study, and the feasibility study. Depending on the context, each of these types of study is sometimes generally referenced as a «feasibility study.» The two important requirements for these types of studies, especially feasibility reports, are as follows: (1) reports must be easy to read and their information must be easily accessible; and (2) parts of the reports need to be read and understood by nontechnical people (Hustrulid et al. 2013). Once a resource estimate has been completed, a decision will be made to either to shelve the project, to continue drilling on the project with the hope of increasing the resource, or to proceed with a preliminary economic assessment or pre-feasibility study. These studies build upon the resource estimate by designing a mine around the deposit and undertaking economic analysis of the viability of a mining operation. Each study builds upon the earlier study by increasing the detail and level of rigor.

The primary goal for determining the feasibility of a mineral property is to prove that the mining project is economically feasible if it is designed and operated properly. The terminology for each stage of feasibility study is very varied, and there is no agreed standard for quality or accuracy. Thus, it is very common to refer it as scoping studies, pre-feasibility studies, and feasibility studies. It is convenient to use this terminology although the study process is iterative and several increasingly detailed pre-feasibility studies can be undertaken before committing to the final feasibility study. Some of these steps are usually overlapped, but this is improbable to reduce the time involved. In this sense, it is not rare to spend about 15 years between the beginning of the prospection program and the start mine production (Moon and Evans 2006).

The studies range from the lowest level of certainty (scoping) to the highest level of certainty (feasibility) and show increasing levels of detail and expense associated with their completion. Only the final feasibility study is considered to have sufficient detail to allow a definitive positive or negative decision for corporate and financial purposes. However, it is important to note that production of a final feasibility study report does not in itself mean that a project is viable or that the project will be one that will attract project finance. Often these project stages are required to be undertaken in line with international codes such as JORC or NI 43-101 (see ► Chap. 1) to determine what is required and includes their associated confidence levels. Regarding the cost of these studies, they vary substantially depending on the size and nature of the project, the type of study being undertaken, the number of alternatives to be investigated, and numerous other factors. For this reason, some estimated data are offered in each type of study.

Pre-feasibility and feasibility studies involve establishing several key components of a mining operation, including mine design, processing methods, reclamation and closure plans, and cash flow analysis. These are referred to as the «modifying factors» under the International Reporting Standards. Mine design involves determining the mining methods, annual and life-of-mine production, equipment needs, and personnel requirements. Processing methods are the methods and equipment needed to concentrate mineral or recover metal from ore, commonly presented in a flow sheet diagram that outlines the steps the ore will go through from the time it leaves the mine until the final product is produced. Reclamation and closure plans are part of the overall mining operation and must be factored into the mine and mill design as well as the cash flow analysis.

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Table 4.16 Example of capital costs in a feasibility study (sustaining cost covers the entire mine site operation from year 1 to the end of production)

Description	Initial cost ($)	Sustaining cost ($)
Mining	15,632,000	3,910,000
Infrastructure	9,343,000	0
Processing	60,130,000	0
Tailings and water management	13,159,000	8,275,000
Construction indirects	18,155,000	0
Owner's costs/ land acquisition	5,644,000	0
Rehabilitation and closure	0	9,824,000
Contingency	22,389,000	4,202,000
Total capital	**144,452,000**	**26,211,000**

Data courtesy of Nouveau Monde Mining Enterprises Inc.

Table 4.17 Example of operating costs

Activity	Annual cost ($/y)	Cost per tonne milled ($/t)	Cost per tonne of conc. ($/t)
Mining *(5.31 $/ tonne mined)*	12,041,000	9.93	241.20
Processing & tailings	16,543,000	13.64	331.38
General and administration	4,366,000	3.60	87.46
Total opex	**32,950,000**	**27.17**	**660.04**

Data courtesy of Nouveau Monde Mining Enterprises Inc.

It represents the detailed economic assessment of the proposed mine and will be taken in detail in next section. Cash flow analysis may be very complex and generally include the capital costs (Table 4.16), the operating costs (Table 4.17), taxes and royalties, and the revenues generated by the sale of products.

Scoping Studies

NI 43-101 Canadian code defines a preliminary assessment, or scoping study or order-of-magnitude study, as «a study that includes an economic analysis of the potential viability of mineral resources taken at an early stage of the project prior to the completion of a preliminary feasibility study.» Thus, this study is the first level of geological/engineering and economic analysis that can be performed, usually at an early stage in the project. At this phase, it is obviously undesirable to expend further funds on something that has no chance of being economic. The bases for these studies are the geology plans from the exploration phase (Fig. 4.50), limited drilling, and other sample collections. This allows to carry out rational estimates using known costs and likely outcomes. The results define the presence of sufficient inferred resources to warrant further work. Where a resource is classified as indicated, a coping study will provide financial assessment of the resource.

This type of study provides a first-pass examination of the potential economics of developing a mine on a mineral deposit. Though a scoping study is useful as a tool, it is neither valid for economic decision-making nor sufficient for reserve reporting. The evaluation is conducted by using mine layouts and factoring known costs and capacities of similar projects completed elsewhere. The study is directed at the potential of the property rather than a conservative view based on limited information, and it is commonly performed to determine whether the expense of a pre-feasibility study and later feasibility study is warranted. At this stage, mineralogical studies will identify undesirable elements and other possible metallurgical issues. It is also common to explore different options for mining and processing the deposit in order to choose the most promising methods for further study.

Scoping study usually takes a few weeks to a few months to complete and cost USD 20,000 to USD 200,000 (Stevens 2010) or 0.1–0.3%, expressed as a percentage of the capital cost of the project

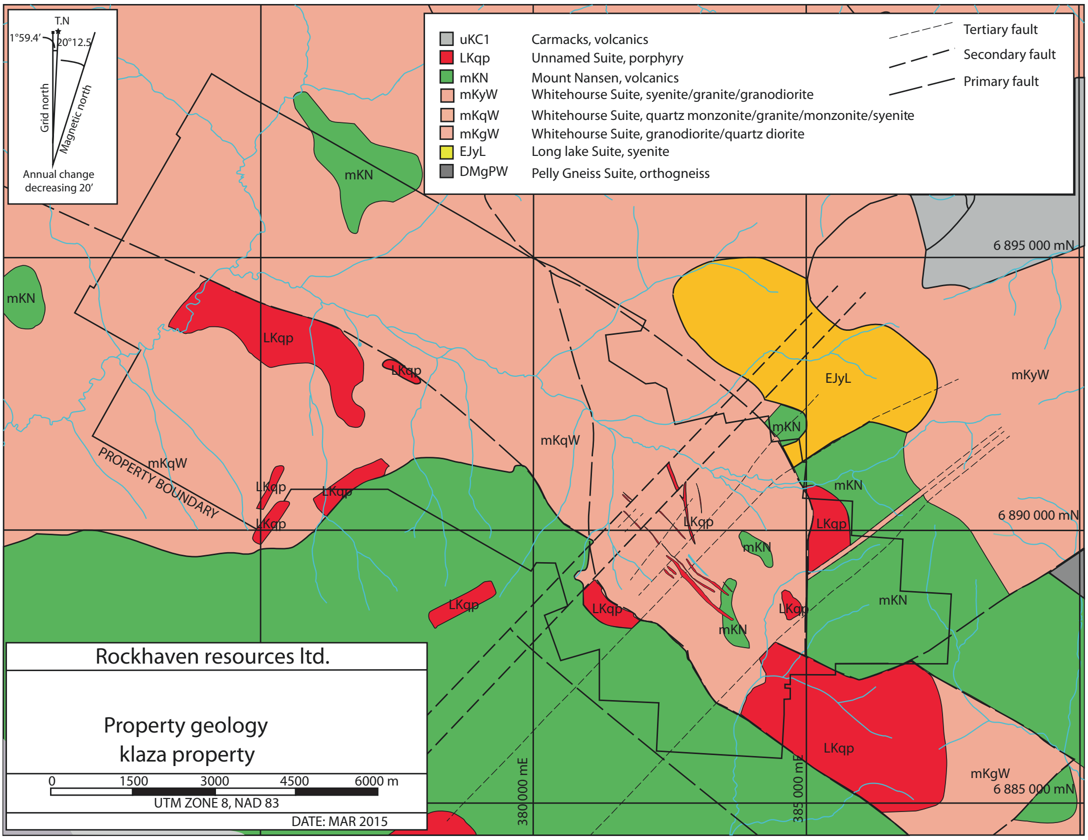

Fig. 4.50 Property geology of Klaza project used in exploration (Illustration courtesy of Rockhaven Resources Ltd.)

(Rupprecht 2004). The major risk at this stage is that a viable mining project is abandoned due to an inadequate assessment. For this reason, it is paramount that expert people are implicated in the study. The intended estimation accuracy is usually 30–35%, though some companies accept ±50%.

Pre-feasibility Studies

NI 43-101 defines a pre-feasibility study as «a comprehensive study of the viability of a mineral project that has advanced to a stage where the mining method… has been established and an effective method of mineral processing has been determined, and includes a financial analysis based on reasonable assumptions of technical, engineering, legal, operation, economic, social, and environmental factors and the evaluation of other relevant factors which are sufficient for a qualified person, acting reasonably, to determine if all or part of the mineral resource can be classified as a mineral reserve.» One of the most important aspects of a pre-feasibility study is that a mineral resource cannot be converted to a mineral reserve unless it is supported by at least a pre-feasibility study. It is common that the results of the pre-feasibility study can be the first hard project information which is seen by corporate decision-makers and investors. The aim of the pre-feasibility study is «to evaluate the various options and possible combinations of technical and business issues to assess the sensitivity of the project to changes in the individual parameters, and to rank various scenarios prior to selecting the most likely for further, more detailed study» (Moon and Evans 2006).

There are many reasons for carrying out a pre-feasibility study, being the most important as follows: (a) as a basis for further development of a major exploration program following a successful preliminary program, (b) to attract a buyer to the project or to attract a joint venture partner, (c)

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Table 4.18 Test results of a selective flotation process carried out in a pre-feasibility study

Product description	Weight %	% Cu	ppm Au	Cu Rec %	Au Rec %
Copper cleaner concentrate	0.9	24.03	26.82	87.0	64.9
Copper rougher concentrate	1.3	17.07	19.17	87.2	65.5
Copper cleaner tailings	0.4	0.17	0.36	0.3	0.6
Copper rougher tailings	98.7	0.03	0.13	12.8	34.5
Total copper tailings	99.1	0.03	0.13	13.0	35.1
Pyrite cleaner concentrate	1.3	0.31	2.96	1.6	10.6
Pyrite cleaner tailings	0.3	0.24	0.63	0.3	0.5
Final tailings	97.2	0.03	0.09	10.8	23.4
Feed	100.0	0.24	0.37	100.0	100.0

Data courtesy of Euromax Resources

to provide a justification for proceeding to a final feasibility study, and (d) as a means to determine issues requiring further attention (Rupprecht 2004). For these reasons, especially the second one, the pre-feasibility study must be carefully prepared by a small multidisciplined group of experienced technical people, and its conclusions should be heavily qualified wherever necessary, being the assumptions realistic rather than optimistic. Thus, the pre-feasibility study represents an intermediate step between the scoping study and the final feasibility study, requiring a high level of test work and engineering design. At the end of a pre-feasibility study, geological confidence is such that it is suitable to publicly disclose ore reserves from measured and indicated resources and any other mineral resources that can become mineable in the future with further study. These studies tend to achieve an accuracy within 20–30%.

In a pre-feasibility study, economic evaluation (see the following headings) is utilized to assess various development options and overall project viability. The results of the study are used to justify expenditure on gathering this additional information and the considerable expenditure needed to carry out the final feasibility study on a substantial project. In a pre-feasibility study, the details of the processing methods will be based on initial metallurgical studies of the mineralization of the deposit (Table 4.18) rather than solely on standard industry methods. Accordingly, pre-feasibility studies can include washing, milling, and numerous other techniques designed to prepare the material for sale and distribution to customers.

Environmental protection, permits including legal and social, and the eventual closure of the mine must all be considered during this phase. The option that demonstrates the highest value with acceptable (lowest) risk will be selected as demonstrably viable. The cost of a pre-feasibility study can be as little as USD 50,000 for a simple project to more than USD 1,000,000 for larger, more complicated projects or 0.2–0.8% of the capital cost of the project (Rupprecht 2004). It commonly takes from 6 months to 1 year to complete (Stevens 2010).

Social and environmental baseline studies must be carried out showing conformance to the Equator Principles. Most importantly, the performance standards along with the World Bank Group's Environmental, Health, and Safety Guidelines form the basis of the Equator Principles. The Equator Principles (EPs) are a voluntary set of standards adopted by financial institutions for determining, assessing, and managing environmental and social risk in project finance activities. According to this, Equator Principles financial institutions (EPFIs) commit to implementing the EP in their internal environmental and social policies, procedures and standards for financing projects and will not provide project finance or project-related corporate loans to projects where the client will not, or is

EUROPEAN UNION

Unique Number: EU

KIMBERLEY PROCESS CERTIFICATE

The rough diamonds in this shipment have been handled in accordance with the provisions of the Kimberley Process Certification Scheme for rough diamonds.

Country of Origin: .... Number of Parcels: ....

Country of Provenance: ....

Name and address of exporter: .... Name and address of importer: ....

HS classification	Carat	Value (US$)
7102.10		
7102.21		
7102.31		

THIS CERTIFICATE

Issued on: .... Expires on: ....

Signature of Authorised Officer / Official Stamp

Fig. 4.51 Kimberley Process certificate in the European Union

unable to, comply with the EP. Obviously, Equator Principles have greatly increased the attention and focus on social/community standards and responsibility since 2010. They include robust standards for indigenous peoples, labor standards, and consultation with locally affected communities within the project finance mining market. The most important lending institutions worldwide, many of whom provide financing for mining activities, have adopted Equator Principles.

A similar initiative is the Kimberley Process (KP). It was founded in 2000 in Kimberley, South Africa, by the governments of South Africa, Botswana, and Namibia. There are actually 54 participants in the KP, including the 28 EU member states, representing 81 countries. The KP tries to join the diamond-producing countries and diamond importers together to remove trade in conflict diamonds and stop them from being used to finance rebel movements. The main KP document applying to rough diamonds is the KP Certification Scheme (KPCS), adopted in November 2002. Today, the KP covers no less than 99.8% of the world diamond trade. Figure 4.51 shows the Kimberley Process certificate in the European Union.

Feasibility Studies

NI 43-101 defines a feasibility study as «a comprehensive study of a mineral deposit in which all geological, engineering, legal, operation, economic, social, environmental and other relevant factors are considered in sufficient detail that it could be reasonable serve as the basis for a final decision by a financial institution to finance the development of the deposit for mineral production.» The term «bankable» is often utilized in connection with feasibility studies. It only means that the study acquires a quality that is acceptable for submission to bankers or other institutions that can finance the project. In fact, it does not really reflect a different type of economic analysis. A better term for a bankable feasibility study would be a «bank-approved» or «bank-vetted study» (Stevens 2010). The reality is that banks or major investment firms will undertake their own internal analysis of a feasibility study to determine if the project meets their investment objectives. If it does meet those objectives, it could be considered bankable.

The feasibility study is the last stage needed to establish if a mine is economically viable. For this reason, it is much more detailed and costly than the previous two study types. The objective is to remove all significant doubt and to present relevant information about referenced material as well as to verify and maximize the value of the preferred technical and business options identified in the previous pre-feasibility study. For this reason, a full feasibility study must prove within a reasonable confidence that the mining project can be operated in a technically sound and economically viable manner. Capital and operating costs are evaluated to an accuracy between 10% and 15%, covering realistic eventualities based on the level of engineering completed. In these studies, the product price is the most important single variable and yet the most difficult to predict. The feasibility study should determine ore reserves as per standard definition (e.g., NI 43-101, SAMREC, or JORC), scale of the project, construction budget and schedule for the project, cost estimate for operating and capital, contingencies (Table 4.19), market estimates (Table 4.20), cash flow studies, and risk analysis (Rupprecht 2004).

Sensitivity analyses are carried out to establish the major factors that can impact upon the reserve estimate (Table 4.21). This will help quantify the risk associated with the reserves, which at this stage will fall within the company's acceptable risk category. Often, financial institutions utilize independent consultants to audit the resource/reserve calculations (Moon and Evans 2006). The defined mine plans in a feasibility study is based on measured and indicated geologic resource, which would become proven and probable reserves. At this stage, consultation and negotiation with local community groups, landowners, and other interested parties will proceed to the point of basic agreement. Full feasibility studies cost in the neighborhood of one to a few million dollars (Stevens 2010) or 0.5–1.5% of the capital cost of the project (Rupprecht 2004) and can take 1–2 years to complete. This type of study is usually undertaken by engineering consulting firms with expertise in various aspects of mine design and development.

Table 4.19 Contingency costs for indirect costs (capital costs) in a feasibility study

Indirect	Cost ($)
910-Construction indirect cost	18,155,000
945-Construction contingency	21,260,000
950-Owner's cost	5,644,000
995-Owner's cost contingency	1,129,000
Total indirect, owner's cost and contingency	**46,188,000**

Data courtesy of Nouveau Monde Mining Enterprises Inc.

Table 4.20 Example of a commodity prices market study included in a feasibility study

Mineral	Units	Spot price (30/09/15)	3-year moving average	Analyst consensus long term	SEDAR (last 12 months)	AmecFW cash row guidelines	AmecFW resources guidelines
Gold	US$ /oz	1,116	1,334	1,227	1,250	**1,250**	1,440
Silver	US$ /oz	14.51	15.11	18.15	18.25	**18.25**	21.00
Copper	US$ /lb	2.35	3.11	3.00	3.00	**3.00**	3.50

Data courtesy of Euromax Resources

Table 4.21 Cutoff sensitivity in the indicated category for the mineral resource estimates

Indicated category			
Cut-off grade (% Cg)	Tonnage (t)	Grade (% Cg)	Graphite content (t)
5.00	1,250,000	5.3	66,250
3.00	21,710,000	3.93	853,200
2.50	26,275,000	3.73	980,060
2.25	27,675,000	3.66	1,012,900
2.00	28,520,000	3.62	1,032,420
1.75	29,150,000	3.58	1,043,570
1.50	29,490,000	3.56	1,049,840
1.25	29,660,000	3.54	1,049,960
1.00	29,775,000	3.53	1,051,060

Data courtesy of Nouveau Monde Mining Enterprises Inc.

4.6.2 Economic Analysis

Cash Flow Analysis

Ancient methods, prior to the early 1960s, in mineral project economic evaluation process include Hoskold method and Morkill method, with the Hoskold method probably the most popular. The Hoskold method was based on the financial policy of British coal mining companies of nearly a century ago. This procedure was no longer in use since the advent of methods based on cash flow analysis such as net present value, internal rate of return, or payback period.

The value of a mineral project can be determined using a variety of valuation techniques and associated methodologies. Although valuation of the mineral project could be required at any stage in its life, and not all of the valuation techniques are applicable to all stages of such a development, some methods are often used to analyze the economic viability of the mining project as a whole. The predominant economic evaluation technique, from pre-feasibility study to operating mine, is the discounted cash flow (DCF) method. The cash flow model must recognize the time value of money discounting at an appropriate discount rate to obtain their present value. DCF criteria values are gross profit; earnings before interest, depreciation, and amortization (EBITDA); net present value (NPV); internal rate of return (IRR); and payback period (PP). NPV, IRR, and PP methodologies are the most accepted by the industry, the financial community, and regulatory bodies. In summary, the general procedure for evaluating investment opportunities is to carry out a comparison between the benefits of any particular opportunity and the associated costs, investing later in those projects that are worth more than they cost.

The change in the amount of money over a given time period is called the time value of money. This concept is based on the principle that, disregarding inflation, money is worth more today than it will be at some future date because it can be put to work over that period. In other words, since investors would rather receive benefits sooner than later, the value of each yearly cash flow generated over the life of a project can be adjusted for the time value of money. Thus, the value of money today is not the same as money received at some future date. The effect of inflation on project value, however, is important and must be considered.

From the concept of time value of money, two important characteristics in valuing mineralization and mineral projects can be outlined. Firstly, as discount factors are highest in the early years, the discounted value of any project is enhanced by generating high cash flows at the beginning of the project. Secondly, discount factors decrease with time and, by convention and convenience, cash flows are not estimated beyond usually a 10-year interval as their contribution to the value becomes minimal. Moreover, it is quite difficult to predict with some degree of accuracy what is to happen after 10 years.

The time value of money is computed using the compound interest formula. For example, if an investment of I = USD 1,000 is made today at an interest rate of 10%, the future value is:

- After 1 year: $I \times (1 + i) = 1000 \times (1 + 0.1)$ = USD 1,100.
- After 2 years: $I \times (1 + i) \times (1 + i) = 1{,}000 \times (1+ 0.1)^2$ = USD 1210.
- After 10 years: $I \times (1 + i)\ 10 = 1{,}000 \times (1 + 0.1)^{10} = 1{,}000 \times 2.594$ = USD 2,594.
- Generally after n years: $I \times (1 + i)n$.

Thus, the general formula will be

$$\mathrm{FV}_n = \mathrm{PV} \times (1+i)^n$$

where FV_n is the future value at year «*n*», PV is the present value, and «*I*» the interest rate. This expression can be rewritten to show the relationship between the future yearly cash flows (CF_t) and the discounted values of the yearly cash flows (DCF*t*) at time period «*t*»:

$$\mathrm{DCF}_t = \frac{\mathrm{CF}_t}{(1+i)^t}$$

Cash flow analysis can be very complex and generally include three main components: (a) capital costs associated with building the mine; (b) operating costs, taxes, and royalties generated to produce the products at a mine; and (c) revenues obtained by the products. Cash flow can be defined as cash into the project (revenue) minus cash leaving the project (cost) or, more in detail, as revenue minus mining, ore beneficiation, transport, sales, capital, interest payments, and taxes costs. From a geological viewpoint, cash flow analysis requires to translate the geologic characteristics of the project into costs of development and extraction and to convert preliminary estimates of reserves into potential revenues from mining, making assumptions about future mineral prices. Regarding the taxation, it is not unexpected in a viable project that taxes and royalties will account for a significant portion of the cash flow.

All texts about mineral project evaluation conclude that the preferred methods of evaluation, where sufficient data is available, are those that include annual cash flow projections and that recognize the time value of money. These are the so-called dynamic methods. They include particularly the net present value and the internal rate of return, as opposed to those employing simple cost and revenue ratios or payback periods and not considering the time value of money (named static methods). On an international level, economic assessment of mining projects are done using basically NPV and IRR and sometimes PP. NPV is a measure of value of a stock of wealth, whereas IRR is a measure of the efficiency of capital use or the rate of accumulation of wealth.

Net Present Value (NPV)

The net present value of a mining project is merely the difference between cash outflows and cash inflows on a present value basis, being the backbone of a project evaluation process. The formula to calculate NPV is as follows:

$$\mathrm{NPV} = (R_0 - C_0) + \frac{R_1 - C_1}{(1+i)} + \frac{R_2 - C_2}{(1+i)^2} + \ldots + \frac{R_n - C_n}{(1+i)^n}$$

where «*R*» is the expected revenues each year, «*C*» is the expected costs each year, and «*I*» is the discount rate for the project. Only cash revenues and costs are incorporated in the net present value calculation, that is, only those revenues actually received or costs currently produced are included in the cash flow for a certain time period. Examples of noncash costs are depreciation and depletion.

In this context, the discount rate equals the minimum rate of return for the project and reflects the opportunity costs of capital, sometimes adjusted for the risk of the project. The opportunity cost of capital is the benefit that would be received by the next investment opportunity. The NPV for different investment projects should be compared using the same discount rate. A positive NPV indicates that expected income is higher than projected expenses and a negative NPV indicates a nonprofit or loss situation so that the project should be abandoned. Obviously, NPV must be positive and usually must be above a certain minimum value determined by the company based on internal standards. The larger the NPV, the richer the investors become by undertaking the project. On the other hand, the higher the discount rate, the lower the NPV of the project.

Selection of a suitable interest rate is essential in the application of NPV because interest rate discounts gradually the cash flow values and establishes finally the net present value of the project. The interest rate for discounting commonly ranges between 5% and 15% over the interest rate of the needed initial capital investment, and in times of high interest rates, this discount rate is particularly onerous. ◘ Table 4.22 can serve as a guideline for discount rate factors at each study level. Often, the different parts involved in the

Table 4.22 Guideline for discount rate factors at each study level

Risk	Study level	Discount rate (%)
Low	Feasibility	8
Medium	Prefeasibility	10
High	Preliminary economic assessment	12
Extremely high	Scoping study	15

mining project had agreed on all aspects of the evaluation and, by combining these components, even on the final cash flow values. The only factor for discussion tends to be related to the discount rate to be used in the calculation of the net present value. Such differences can cause a variation of more than 50% in the value placed on a project. The discount rate is used not only as the discount rate in the NPV method but also as the minimum rate for the IRR.

It is not easy to deal specifically with the selection of discount rates for mineral project evaluations although economic and finance theory proposes the use of the corporate cost of capital as a discount rate. In general, mining companies, for cash flow evaluations at the feasibility study level of projects in low risk countries, commonly select a discount rate of 10%. In fact, the companies actually use to determine the discounting rate to utilize in their financial evaluations applying the weighted average cost of capital (WACC) method. It is the weighted average of the costs that a company has to pay for the capital it uses to make investments. In general, the higher the risk in the project, the higher the discounting rate applied to it. For this reason, sometimes a company will apply a modifying factor to the WACC to account for increased risk in certain projects (e.g., projects with high risk can use a discounting rate equal to the WACC plus 2–3%).

Internal Rate of Return (IRR)

Internal rate of return (IRR) method is one of the most used investment analysis methods and, besides NPV, is probably the most common evaluation technique in the minerals industry. In the IRR method, the objective is to find the interest rate at which the present sum and future sum are equivalent. In other words, the present or future sum of the all cash flows is equal to zero if IRR value is used as interest rate. It is clear that at the discount rate increases for a specific cash flow, the NPV of the cash flow necessarily decreases. The relationship between IRR and NPV can be written as

$$\mathrm{NPV} = 0 = \left[\sum_{t=1}^{n} \frac{\mathrm{CF}_t}{(1+\mathrm{IRR})^t}\right] - I_0$$

where CF_t is the cash flow in year «t,» I_0 is the initial investment (CF_0), IRR is the discount rate that makes NPV = 0, and «n» is the total number of years for the project. In general, calculations of IRR and NPV commonly give the same accept or reject recommendation, but the IRR method is more complicated than relying on NPV estimations.

It can be understood that IRR value is the interest rate at which the investor recovers the investment. The higher a project's internal rate of return, the more desirable it is to undertake the project because the better the return on capital. If all projects require the same amount of initial investment, the project with the highest IRR would be considered the best and undertaken first. Investment banks and other groups that fund the capital costs for mining operation like to see IRR values exceeding 10% and with values of 20% or better being ideal (Stevens 2010).

There are several reasons to explain the widespread popularity of IRR as an evaluation criterion, being probably the most interesting that IRR is expressed as a percentage value and many managers and engineers prefer to think in terms of percentages. Thus, the acceptance or rejection of a project based on the IRR criterion is carried out by comparing the estimated rate with the necessary rate of return: if the IRR exceeds the required rate, the project should be accepted, but if not, it should be rejected. The difference between the discount rate and IRR is that the investor chooses the discount rate whereas the characteristics of the cash flow determine the IRR. Consequently, IRR is determined internally (hence its designation as the «internal» rate of return), as compared to the discount rate for NPV, which is determined externally (Torries 1998).

4

Payback Period (PP)

The payback (or payout) period falls under the heading static methods but it is one of the most simple and common evaluation criteria used by engineering and resource companies. It is the number of years required for a project to generate cash flow or profits equal to the initial capital investment. It is important to note that cash flow in the first year or even more years of a mining operation will be negative since these years are used to pay the previous investments (e.g., exploration). The PP method is a helpful evaluation index since it generates an indication of how long the company has to wait to get its return on investment although it is an inappropriate evaluation technique if used alone because it does not take into account the total cash flow or distribution of cash flows over the life of the project. The rationale of this method is that a shorter time required to get back the investment is better (Torries 1998).

The method does not provide a guidance for the selection of an acceptable payback period, that is, one company may select 3 years, while another can choose 6 years under the exact set of circumstances. However, for most normal mining projects, payback periods lie between 3 and 5 years, and as a rule, shorter payback periods are required in high-risk countries than in stable countries. The method serves as a preliminary screening process, but it is inadequate as it does not take into account the time value of money. Payback period is very helpful in countries of political instability where the retrieval of the initial investment within a short period is clearly essential. For example, consider the use of payback in assessing the feasibility of developing a rich deposit in a remote and politically unstable area. The project can have a very attractive rate of return, but management will probably not give approval until it is shown that payback can be achieved in less than 2 years (Torries 1998). Table 4.23 incorporates a simple calculation of NPV, IRR, and PP values, whereas Table 4.24 is a real case of NPV, IRR, and PP estimations (NPV is estimated for different discount rates).

Table 4.23 Simple calculation of NPV, IRR, and PP values; money is expressed in monetary units (MU)

Year	Revenues	Costs	Cash-flow
0	0	5	−5
1	0	10	−10
2	0	20	−20
3	10	5	+5
4	20	10	+10
5	40	20	+20
6	40	20	+20

VAN = 3.68; IRR = 13.59%; PP = 5 years

Table 4.24 NPV, IRR, and PP estimations in a real economic analysis

Cost category	Unit	Low case	Base case	High case
NPV @ 0%	M$	1,065	1,299	1,532
NPV @ 8%	M$	255	320	385
NPV @ 10%	M$	186	236	286
NPV @ 12%	M$	137	176	215
IRR	%	38.4	45.7	52.8
Payback period	Years	6.0	2.0	1.7

Inflation

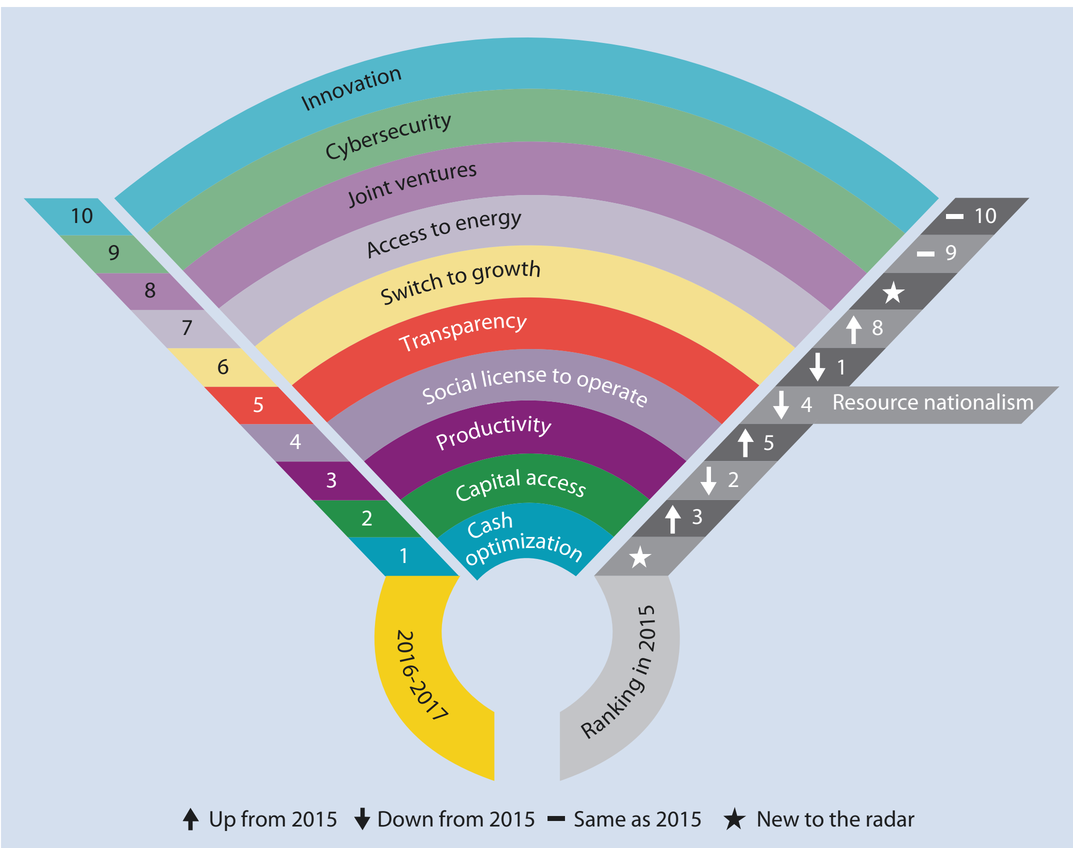
In a project evaluation, anything that changes or impacts costs and revenues is worthy of review. Inflation is such a factor and because it increases at a compounding rate over time, it must be considered carefully before it is applied to a project. Thus, inflation, the sustained increase in the general price level of goods and services in an economy over a period, cannot be overlooked in an evaluation process. If management selects to preclude inflation from the estimation, it should be aware of the results of this decision (Smith 1987). In general, a mining project should be evaluated using several rates of inflation. In the absence of a strong personal or corporate policy on inflation, the consumer price index is often used.

A common error in auditing cash flows is the use of real or constant and nominal or current monetary units (e.g., dollars). Often, mining companies will analyze projects based in real dollars and financial institutions commonly use nominal dollars. The process of converting from nominal to real terms is known as inflation adjustment.

Cash flows can be calculated either on a constant or current (inflated) dollar basis but regardless of which based is used, all prices, costs, and rates must be expressed in the same terms. Thus, it is incorrect to mix current dollars values with constant dollar values in a single cash flow. Most company financial statements and reports are in nominal dollars and can serve as a basis for risk evaluation. If inflation can be forecast, current dollar analysis gives results that are more reliable.

Risk Analysis

The previous methods to evaluate investment alternatives assume that future benefits and costs are known with certainty at the time of investment, which is clearly a questionable assumption especially in many types of mining project investments. Thus, risk is thought of as a measure of the degree of variability of possible future revenues and costs. Mining involves large risks and the magnitude of uncertainties in mine development projects is generally larger than in most other industries used for comparison. A project in which future prices and costs are known with certainty can and should be evaluated in a different manner from one in which these factors are not known. All input values in DCF analysis must be known with certainty, so that there must be no uncertainty or risk. A numerical value of NPV can be correctly determined using any set of numbers, but the true value of an investment can be determined only if all input values are known with certainty. This certainty is seldom possible since future prices or costs are not exactly known. Moreover, the determination of risk is actually complex in the minerals industries because of the need to include environmental risks and costs in project evaluation. Some companies or institutions tend to issue annually a detailed report on the main risks in mining sector, the so-called by EY's Global Mining & Metals Center top 10 business risks facing mining and metals. As an example, the top 10 risks for 2016–2017 are shown in ◘ Fig. 4.52. The report also includes the main



◘ **Fig. 4.52** The top business risks form mining and metals three top risks for each commodity (aluminum, coal, copper, gold, iron ore, lead/zinc, nickel, PGM, potash, silver, steel, and uranium). In this sense, «price and currency volatility» risk is the first one in six commodities and the second in three more.

Where uncertainty and risk are moderately absent, project evaluation is an easy exercise. However, mineral projects commonly involve raw materials for which prices or operating processes are difficult to forecast. In general, the higher the risk experienced by an investor, the higher the expected returns. Without the promise of higher returns, an investor would have no reason to consider projects with higher risks. Consequently, the inclusion or exclusion of risk in economic evaluation is of huge importance. Although the usage of the term risk as a synonym for uncertainty is not right because their definitions are not exactly the same, it is worthy to remember that they are utilized indistinctly in this section. Risk can be denoted by single probability estimation, whereas uncertainty can be denoted by a range of estimates.

There are three categories of mineral-development risk according to cause of the risk: technical, economic, and political risks (Park and Matunhire 2011). The technical risks, which are at least partly under the control of the organizations active in mineral development, are divided into reserve risk, completion risk, and production risk. Reserve risk, determined both by the nature and by the quality of ore-reserve estimates, reflects the possibility that actual reserves will differ from initial estimates. Thus, any resource and reserve estimation is guaranteed to be wrong; some, however, are less wrong than others (Morley et al. 1999). Completion risk reflects the possibility that a mineral-development project will not make it into production as anticipated. Production risk reflects the possibility that production will not proceed as expected because of production fluctuations.

The economic risks are divided into price risk, demand/supply risk, and foreign exchange risk. Price risk is the possible variability of future mineral prices. The most important risk factor is lack of knowledge of the future price of product of mining (Rendu 2002). Demand/supply risk accounts for the difficulty in achieving reliable demand/supply forecasts, and foreign exchange rate risk is the variability of possible foreign exchange rates in the future. Finally, political risks are defined by the political instabilities. In this sense, the general reasoning for a diversification strategy is to reduce fluctuations in earnings produced by mineral price instability and/or unforeseen government actions or other events in a particular country.

To account for risk and uncertainty (the uncertainties begin with exploration and continue up to end of mine life) in economic evaluations, many modifications to NPV analysis are used, including mainly one or more of the following: sensitivity analysis, risk-adjusted discount rates, scenario analysis and Monte Carlo simulation. Other less common techniques include, for example, certainty equivalence or Bayesian analysis.

Sensitivity Analysis

Sensitivity analysis is a form of risk assessment that is applied to financial analysis of any mining project. It is a procedure that analyzes what will happen to the value of the mining project if any of the key inputs were to change. The basic process for conducting sensitivity analysis involves changing each input variable one at a time, leaving all the other variables constant and assessing the effect that has on the total project value. This method of risk analysis is probably the most used in mineral project evaluations. The range of possible outcomes commonly includes best-case and worst-case scenarios, showing the best and worst combinations of other possible values of each variable that influences NPV estimation. Sensitivity analysis can also include testing the extent to which individual variables influence the economic engaging of a mining investment.

In any mining project evaluation, certain components have a greater effect upon the size of the cash flow, and hence value, than others. It is common to look at the effect on the net present value of the project, but it is possible and often necessary equally to look at the effect on the IRR or the payback period. There are three main objectives in the sensitivity analysis process: (a) to determine which variables have the biggest impact on the project value; (b) to reveals the significant variables, which, if varied or misestimated, would significantly change the acceptability of the project; and (c) to determine which variables we need to be estimated more accurately. The results of a

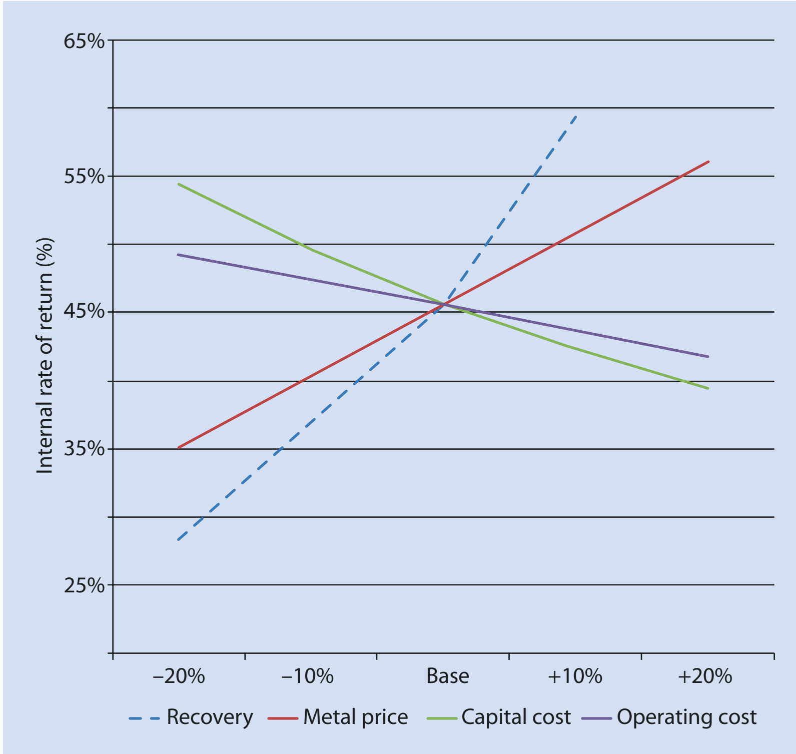


Fig. 4.53 Spider graph of IRR sensitivity (Illustration courtesy of Alabama Graphite Corp.)

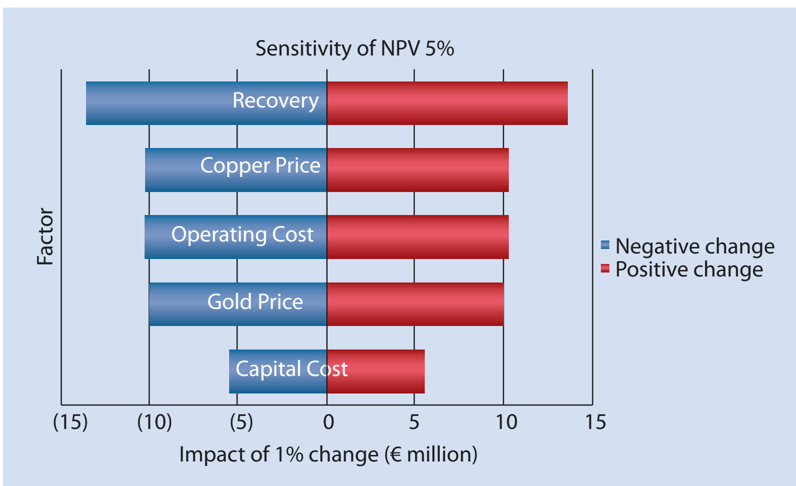


Fig. 4.54 Tornado graph of NPV sensitivity at 5% discount rate (Illustration courtesy of Euromax Resources)

sensitivity analysis are usually presented in two forms, either graphically (e.g., spider and tornado graphs) or in a table. Thus, Fig. 4.53 shows a spider graph of IRR sensitivity and Fig. 4.54 a tornado graph of NPV sensitivity at 5% discount rate. Regarding the presentation of sensitivity analysis data in tables, Table 4.25 shows the NPV and IRR Sensitivity to metal prices.

Scenario Analysis

Multiple combinations of factor values originate uncertainty. As a result, it is necessary to investigate the results of scenarios in which combinations of variables are changed. This type of approach is known as scenario analysis. The problem the decision-maker faces is caused by insufficient information to make an informed

4

Table 4.25 NPV and IRR Sensitivity to metal prices

Gold price (US$/oz)	Copper price (US$ /lb)	NPV at 0% discount (US$ M)		NPV at 5% discount (US$ M)		NPV at 10% discount (US$ M)		IRR	
		Pre-tax	Post-tax	Pre-tax	Post-tax	Pre-tax	Post-tax	Pre-tax	Post-tax
1,100	2.50	$474	$412	$221	$174	$61	$25	12.7%	11.1%
1,220	2.90	$939	$839	$513	$440	$260	$205	19.8%	17.8%
1,400	3.50	$1,637	$1,469	$951	$835	$559	$474	28.6%	25.9%

Data courtesy of Euromax Resources

decision. One way to identify these unknowns is to construct scenarios (e.g., optimistic, base, and pessimistic) involving the expected ranges of input variables. The base case is constructed from the best estimates of the project parameters, and the resulting NPV is often, although incorrectly, called «expected value» of the project (Torries 1998). The pessimistic case shows the results of what happens where everything goes poorly while the optimistic case shows what happens where everything goes well.

Monte Carlo Simulation

A more quantitative approach to risk assessment must also incorporate mathematical and statistical methods to assess the risk associated with a project. In the Monte Carlo method, a simulation modeling technique, random number generator is used to calculate probability for each combination of events. The randomized calculation is repeated for many iterations so that an estimate of the overall probability for each outcome can be estimated. Thus, the method accounts for risk in a continuous manner instead of a discrete way because it takes into account all possible values of the underlying determinants of profitability rather than just different specific values.

In the same way that sensitivity analysis changes one variable at a time, the Monte Carlo simulation changes two or more input variables at the same time. Obviously, the overall impact on the project value will be much greater. There is enormous amount of combinations of different variables and different amounts of variation that we will have to deal with. For this reason, and the huge amount of calculations that result, Monte Carlo analyses are nearly always carried out using specific software packages that can model different combinations very quickly. The number of iteration is determined regarding the project size and the importance of risks (1,000, 2,000, 5,000, and so on). The higher number of runs gives the more accurate results. In most cases, to make the calculation easier, the variables are assumed independent from one another although most of the variables are commonly correlated. For example, ore grades are positively correlated with ore recovery. Regarding the presentation of Monte Carlo simulation results, for every combination of input variables a project value is calculated. After repeating the calculation for every combination (number of iteration), all the project values are plotted on a histogram and statistical parameters, such as median, mean, mode, percentiles, etc., are taken into account. The decision rule is to accept those investments with positive means or expected profits. Figure 4.55 shows the distribution of NPV values in a Monte Carlo simulation (Park 2012).

As a summary of the different steps included in a mining project economic evaluation, the following box shows an example of this type of studies (Box 4.6: Matawinie Project Economic Analysis).

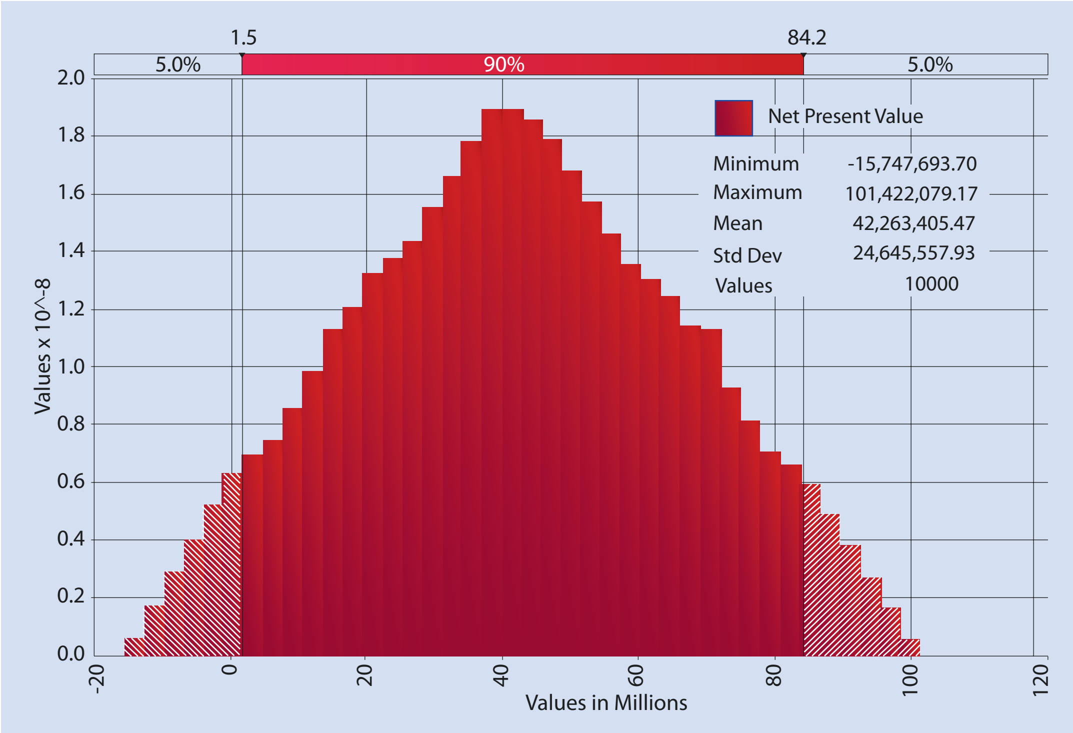


Fig. 4.55 Distribution of NPV values in a Monte Carlo Simulation (Park 2012)

Box 4.6

Matawinie Project Economic Analysis: Courtesy of Nouveau Monde Mining Enterprises Inc.

The economic/financial assessment of the Matawinie Project of Nouveau Monde Mining Enterprises Inc. is based on Q2-2016 price projections in U.S. currency and cost estimates in Canadian currency. An exchange rate of 0.780 USD per CAD was assumed to convert USD market price projections and particular components of the cost estimates into CAD. No provision was made for the effects of inflation. The financial indicators under base case conditions are shown in Table 4.26. A sensitivity analysis reveals that the project's viability will not be significantly vulnerable to variations in capital and operating costs within the margins of error associated with preliminary economic assessment (PEA) estimates. However, the project's viability remains more vulnerable to the USD/CAD exchange rate and the larger uncertainty in future market prices.

The main macroeconomic assumptions used in the base case are given in Table 4.27. The price forecast for graphite concentrate is based on 60-month size-purity-dependent averages calculated from the Benchmark Mineral Intelligence Flake Graphite Price Index. The sensitivity analysis examines a range of prices 30% above and below this base case forecast. The sensitivity of base case financial results to variations in the exchange rate was examined. Those cost components which include US content originally converted to Canadian currency using the base case exchange rate were adjusted accordingly.

The federal and provincial corporate tax rates currently applicable over the project's operating life are 15.0% and 11.9% of taxable income, respectively. The marginal tax rates applicable under the recently adopted mining tax regulations are 16%, 22%, and 28%

4

Table 4.26 Financial indicators under base case conditions

Base case financial results	Unit	Value
Pre-tax NPV @ 8%	M CAD	403.7
After-tax NPV @ 8%	M CAD	237.0
Pre-tax IRR	%	31.2
After-tax IRR	%	24.7
Pre-tax payback period	Years	2.9
After-tax payback period	Years	3.5

Data courtesy of Nouveau Monde Mining Enterprises Inc.

Table 4.27 Main macroeconomic assumptions used in the base case

Item	Unit	Base case value
Average graphite concentrate price (FOB mine)	USD/tonne	1,492
Exchange rate	USD/CAD	0.780
Discount rate	% per year	8
Discount rate variants	% per year	6 and 10

Data courtesy of Nouveau Monde Mining Enterprises Inc.

Table 4.28 Main technical assumptions used in the base case

Item	Unit	Base case value
Open pit resource mined	M tonnes	30.8
Average mill head grade	% Cg	4.48
Design milling rate	K tonnes/year	1,212.5
Average stripping ratio[a]	w:o	0.939
Mine life	Years	25.7
Process recovery	%	89.5
Concentrate grade	% Cg	97.3
Average concentrate production[a]	tonnes/year	49,450
Average mining costs[a]	$/tonne milled	10.04
Average processing costs[a]	$/tonne milled	13.68
Average general and administration costs[a]	$/tonne milled	3.64
Average total costs[a]	$/tonne concentrate	664.26

Data courtesy of Nouveau Monde Mining Enterprises Inc.
[a]Average values have been calculated based on the cash flow statement and represent the average over the life of mine

of taxable income and depend on the profit margin. As the mine is to produce a concentrate, a processing allowance rate of 10% is assumed.

The assessment was carried out on a 100% equity basis. Apart from the base case discount rate of 8.0%, two variants of 10.0 and 12.0% were used to determine the net present value of the project. These discount rates represent possible costs of equity capital. The main technical assumptions used in the base case are given in Table 4.28. A reduced production of 909.4 kt milled in the first production year provides for a ramp-up to full capacity.

Figure 4.56 illustrates the after-tax cash flow and cumulative cash flow profiles of the project for base case conditions. The intersection of the after-tax cumulative cash flow curve with the horizontal dashed line represents the payback period. A summary of the evaluation results is given in Table 4.29. The summary and cash flow statement indicate that the total preproduction (initial) capital costs were evaluated at USD 144.5 M. The sustaining capital requirement was evaluated at USD 14.4 M. Mine closure costs in the form of trust fund payments at the start of mine production were estimated at an additional USD 11.8 M. The cash flow statement shows a capital cost breakdown by area and provides an estimated capital spending schedule over the 2-year preproduction period of the project. Working capital requirements were estimated at 3 months

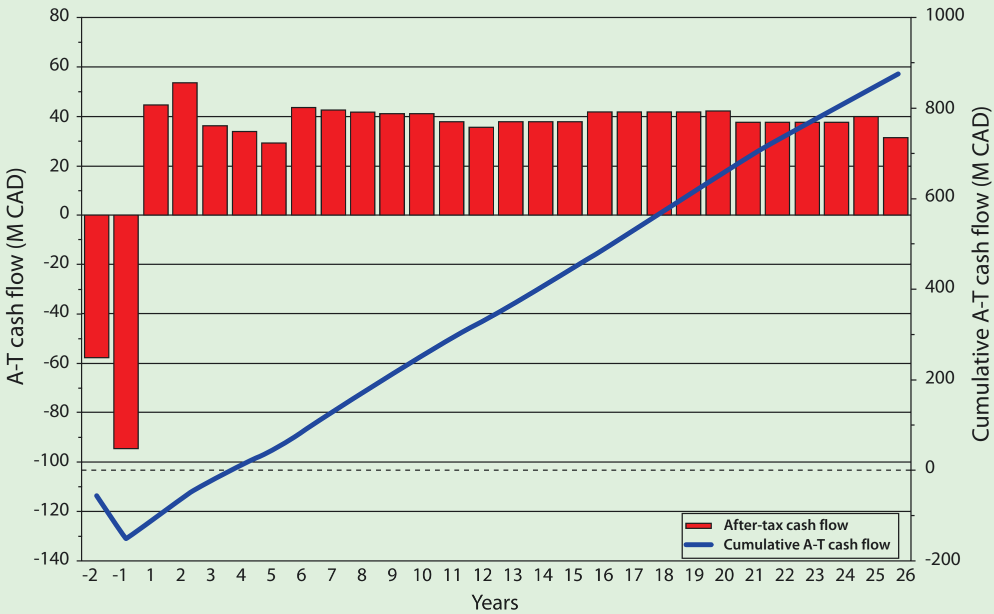


Fig. 4.56 After-tax cash flow and cumulative cash flow profiles (Illustration courtesy of Nouveau Monde Mining Enterprises Inc.)

Table 4.29 Project evaluation summary – base case

Item	Unit	Value
Total revenue	M CAD	2,430.9
Total operating costs	M CAD	884.2
Initial capital costs (excludes working capital)	M CAD	144.5
Royalty buyout	M CAD	2.0
Sustaining capital costs	M CAD	14.4
Mine rehabilitation trust fund payments	M CAD	11.8
Total pre-tax cash flow	M CAD	1,414.1
Pre-tax NPV @ 6%	M CAD	540.2
Pre-tax NPV @ 8%	M CAD	403.7
Pre-tax NPV @ 10%	M CAD	304.7
Pre-tax IRR	%	31.2
Pre-tax payback period	Years	2.9
Total after-tax cash flow	M CAD	872.7
After-tax NPV @ 6%	M CAD	323.4
After-tax NPV @ 8%	M CAD	237.0
After-tax NPV @ 10%	M CAD	174.0
After-tax IRR	%	24.7
After-tax payback period	Years	3.5

Data courtesy of Nouveau Monde Mining Enterprises Inc.

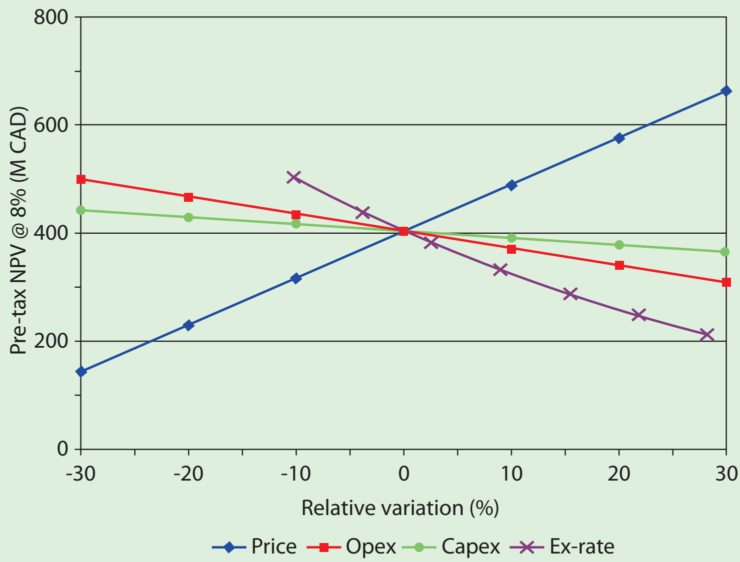
of total annual operating costs. Since operating costs vary annually over the mine life, additional amounts of working capital are injected or withdrawn as required.

The total revenue derived from the sale of the concentrate was estimated at USD 2,430.9 M, or on average, USD 78.79/ton milled. The total operating costs were estimated at USD 844.2 M, or on average, USD 27.36/ton milled. The financial results indicate a pretax net present value («NPV») of USD 403.7 M at a discount rate of 8.0%. The pretax internal rate of return («IRR») is 31.2% and the payback period is 2.9 years. The after-tax NPV is USD 237.0 M at a discount rate of 8.0%. The after-tax IRR is 24.7% and the payback period is 3.5 years.

Regarding the sensitivity analysis, it has been carried out, with the base case described above as a starting point, to assess the impact of changes in total pre-production capital costs («Capex»), operating costs («Opex»), product price («PRICE»), and the USD/CAD exchange rate («EX RATE») on the project's NPV at 8.0% and IRR. Each variable was examined one at a time. An interval of ±30% with increments of 10.0% was used for the first three variables. USD/CAD exchanges rates of 0.70, 0.75, 0.80, 0.85, 0.90, 0.95, and 1.00 (relative variations of −10.3, −3.9, 2.6, 9.0, 15.4, 21.8, and 28.2%, respectively) were used. The US content associated with the capital cost estimate was adjusted accordingly for each exchange rate assumption.

The before-tax results of the sensitivity analysis, as shown in ◘ Fig. 4.57, indicate that, within the limits of accuracy of the cost estimates in this study, the project's before-tax viability does not seem significantly vulnerable to the underestimation of capital and operating costs, taken one at a time. The NPV is more sensitive to variations in Opex than Capex, as shown by the steeper slope of the Opex curve. As expected, the NPV is most sensitive to variations in price and the USD/CAD exchange rate. The NPV remains positive at the lower limit of the price interval and at the upper limit of the exchange rate interval examined.

The same conclusions can be made from the after-tax results of the sensitivity analysis. They indicate that the project's after-tax viability is mostly vulnerable to a price forecast reduction and change in the USD/CAD exchange rate, while being less affected by the underestimation of capital and operating costs. Nevertheless, the NPV remains positive at the lower limit of the price interval and at the upper limit of the exchange rate interval examined.



◘ **Fig. 4.57** Pretax $NPV_{8\%}$: sensitivity to Capex, Opex, price, and USD/CAD exchange rate (Illustration courtesy of Nouveau Monde Mining Enterprises Inc.)

4.7 Questions

Short Questions

- Definition of sampling.
- What QA/QC means?
- Differences between channel sampling and chip sampling.
- What is bulk sampling?
- Explain briefly the relationship between number of holes drilled and the precision of reserve estimates.
- List the factors that influence the appropriate weight of sample.
- Describe the two methods used for reduction of sampling weight.
- What is compositing?
- What outliers means?
- Explain the concept of cutoff grade.
- What is the most common way to determine bulk density?
- Define the concept of geostatistics. What is the most important difference between classical and geostatistical methods regarding error estimation?
- Explain the significance of kriging.
- What is the net present value of a mining project?
- Explain the Monte Carlo simulation in risk analysis.

Long Questions

- Explain the main sampling drillhole procedures.
- Describe the spherical or Matheron model used in geoestatistical studies.

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Mineral Resource Extraction

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Summary

Mineral extraction is the procedure of excavation and recuperation of mineralization and associated waste rock from the crust of the Earth to derive a profit. This chapter reviews the main topics related to mineral resource extraction from surface and underground methods to drilling and blasting. In this process, mineralization is obtained from the ground using surface and/or underground mining methods. These methods are fairly described together with loading and hauling equipment. Transition from surface mining to underground mining is also considered. To illustrate the development of each mining method, several case studies are included in the text. With regard to drilling and blasting, drilling methods, types of industrial explosives, and blast design are discussed. The last heading of the chapter is devoted to grade control, which is essential to the economics of a mine.

5.1 Introduction

Mineral extraction is the procedure of excavation and recuperation of mineralization and associated waste rock from the crust of the Earth to derive a profit. This mineralization generates the essential metal and mineral products used by present society. Mining is where all the hard rock, time and cost of exploration, evaluation, financial analysis, permitting, and construction pay off. Thus, extraction is the culmination of the preceding stages. Although extraction focuses on production, it is accompanied by some exploration and development work, which should continue until the end of the mine's life. The process of planning a mine can be reduced to a network of interrelated systems that are tied together by a common philosophy of mine planning: the resource being mined is to be extracted in a safe (Fig. 5.1), efficient, and profitable manner (Bise 2003). Increasing mining costs, declining average ore grades, environmental considerations, and improved health and safety awareness are some of the main challenges facing the mining industry in the last decades (Wetherelt and Van der Wielen 2011).

In this extraction procedure, mineralization is obtained from the ground using surface and/or underground mining methods. In surface mining, soil and rocks overlying the mineral deposit are removed prior to extract the mineralization, which is exploited from the surface. These soil and rocks are left in place in underground mining, being the mineralization extracted using a network of shafts and adits. Mines range in size from small underground operation producing a few hundred tons of mineralization per day to very large surface mines such as the Escondida mine (Chile) (Fig. 5.2), which produces near 250,000 tons per day of copper, gold, and silver ore and greater amount of waste, being in terms of production, the largest world copper mine.

Whatever the investment activities or metal prices, the amount of metal produced every year in global mining is fairly stable and increasing slowly but steadily. Total volumes of rock and ore handled in the global mining industry amount to approximately 40,000 millions of tons per year. Roughly 50% are metals, coal about 45%, and industrial minerals account for the remainder. The capability of the Earth to meet these rocks and minerals demand is not truly a matter of resources, since they are clearly there, but rather a matter of price and cost (Hustrulid and Fernberg 2012). The answer to this question will be determined by the ability of mining and mineral processing technology to stay ahead of demand growth (Randolph 2011).

During the development and extraction stages of mining, significantly similar unit operations are commonly used leaving aside the mining method selected. These steps contributing directly to mineral extraction are called «production operations» and conform globally to the production cycle. This employs unit operations that are normally grouped into rock breakage and material handling. The basic production cycle in mining consists of drilling + blasting + loading + hauling. In addition to the operations of the production cycle, certain auxiliary operations must be commonly performed. Thus, mines require compressed air, electrical power, mine ventilation, mine dewatering, or pumping and backfill distribution. For instance, in a common day, a typical underground mine uses a greater mass of ventilating air than ore; the deeper the mine, the more air to be moved. For this reason, it is essential to ensure that all the workings in the mine are kept free of blasting fumes and dust. Finally, modern mine designs have to incorporate underground garages, fueling stations, and repair areas.

Fig. 5.1 Lac des Iles mine rescue team (Canada) (Image courtesy of North American Palladium Ltd.)

Fig. 5.2 Escondida mine (Chile) (Image courtesy of Rio Tinto)

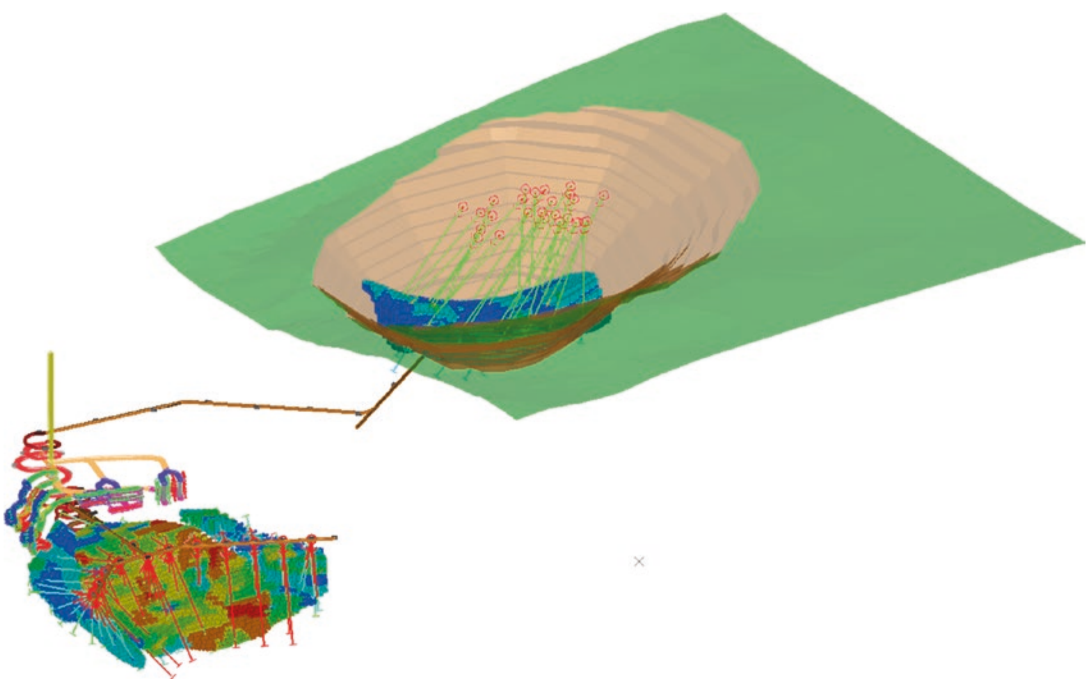
5.2 Surface Mining vs Underground Mining

The method chosen for extraction of the mineral deposit defines the third stage in the life of the mine, being the selection of the method the key decision to be made in mine development. It must take into account many factors and can have to be refined and changed over time. For example, it can be logical for a large copper deposit to be mined first by the open-pit (surface) method, then by block caving (underground) method, and finally by solution mining method. The fundamental rule of extraction is to choose a mining method that combines the singular features of the mineral deposit being extracted and the environment to perform the general lowest cost and return the maximum profit (Hartman and Mutmansky 2002).

Some deposits are mined completely with surface methods, while others can only be worked underground. If an ore body is located very deep, surface mining obviously is not a viable method. These deposits commonly display geological and mineralogical characteristics that require more selective ore extraction. In some cases, especially in areas of high construction density, it is almost impossible to obtain permits for new surface mines. This is the case of quarries for aggregates in large metropolitan areas of many developed countries. To solve this problem, the unique possibility is to develop underground quarries.

The issue arises where the deposit is located at depth that is amenable to either surface or underground mining methods. If an ore body is large and spreads from surface to great depth, mining process starts near the surface, and then extraction continues by underground mining of the deeper parts of the ore body utilizing a ramp from the lower part of the pit (▣ Fig. 5.3). The increasing cost of extract waste at greater depths is one of the major factors in deciding when to transition from surface to underground mining of a given deposit.

Where choosing between surface and underground methods, there are many factors, both quantitative and qualitative, that must be evaluated to select the mining method. Some of the factors that must be considered include (a) size, shape, and depth of the deposit; (b) geological structure and geotechnical conditions; (c) productivities and machinery capacities; (d) availability of experienced work force; (e) capital requirements and operating costs; (f) ore recoveries and revenues; (g) safety and injuries; (h) environmental impacts, during and after the mining; (i) reclamation and restoration requirements and costs; and (j) societal and cultural expectations (Nelson 2011). For example, if the mineral deposit lies horizontal, it is commonly mined through either surface or underground mining methods, but not both; instead, for a steeply dipping vein or massive deposit that outcrops on the surface and extend very deep, the best strategy is often to mine at first using surface methods and then changing to underground mining. In fact, the ore-to-waste

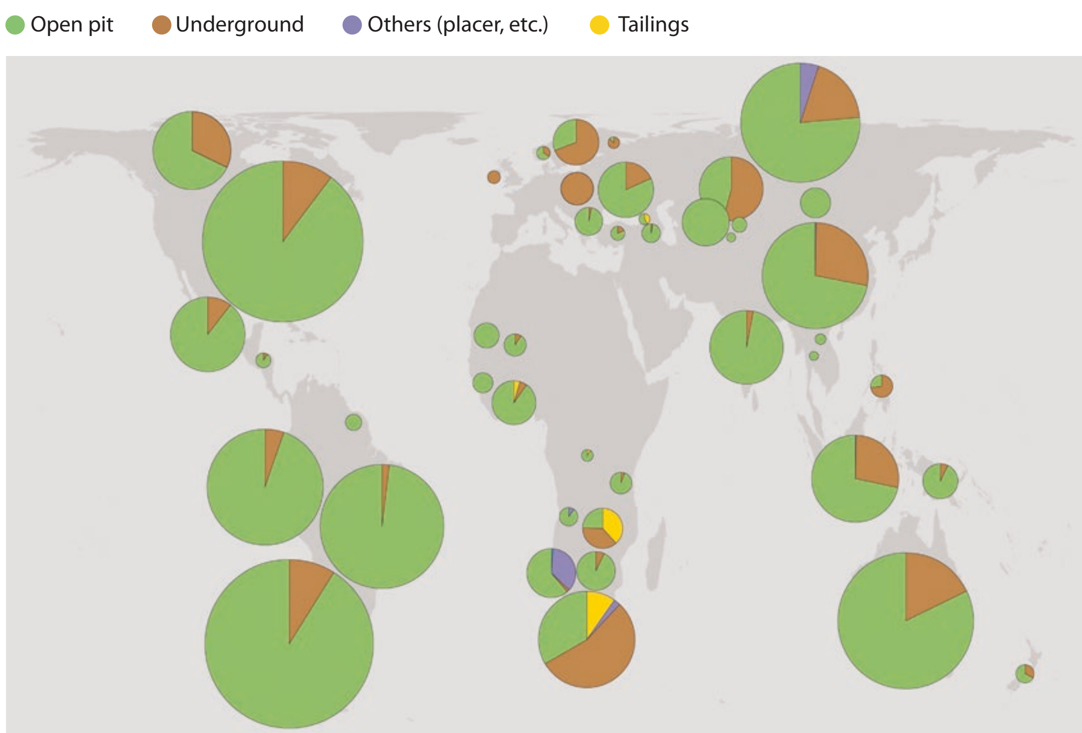


▣ **Fig. 5.3** Software mining design including an open-pit and underground planning with the respective bore holes and block models (Illustration courtesy of Datamine) ratio is the principal feature in the choice between surface and underground mining.

Although both methods have benefits and drawbacks, surface mining is usually a more profitable method than underground mining. In terms of daily production tonnage, surface mines are almost always larger than underground mines producing the same commodity. This is because surface mines must extract much higher waste rock, whereas many of the underground mines extract the same mineral more selectively and with less dilution. With all other conditions equal, surface mining is normally regarded as preferable, because of lower development costs, quicker start-up time, and lower number of accidents; the underground mining environment is recognized as being more hazardous than the surface environment. Underground mining is generally more expensive than surface mining since it is more capital intensive. Surface mining is also preferred because it does not need to extract an excessive amount of waste rock to access the ore.

In an underground mine, a significant amount of infrastructure must be installed before mining begins, in which that large capital investment is often necessary before production can start. The development of a large underground mine can take as many as 5–10 years. Interest costs during this time will therefore be high and can comprise 30–40% of the pre-mining capital requirements before mining can start. However, for large tonnage production, capital and operating costs are commonly higher for surface mines. In these cases, a dual feasibility study must be performed comparing the surface option to the best underground mining option. In all circumstances, capital costs increase and operating costs decrease with increasing production tonnage.

The time between overburden removal and the mining of the product mineral in surface mines should be as short as possible to optimize overall cash flow. Otherwise, high preproduction development costs are produced, and the interest costs during development are high and represent a significant portion of the pre-mining capital requirement before mining can start (Nelson 2011). The dominance of surface operations (ICMM 2012) (◘ Fig. 5.4) is based on the



◘ **Fig. 5.4** Production by mining method 2011 (ICMM 2012) amount of rock handled, many times mainly the removal of overburden, which is often drilled and blasted. Thus, by necessity, the surface operations are larger than the underground ones. Moreover, as a result of economy of scale, mining design and equipment have drastically increased in size in the last decades, although this strategy cannot always be advantageous. Consequently, a number of large mining companies pursued a strategy of owning and operating large-scale world-class mines, typically in the form of large surface mines, although the depth at which surface mines can be developed is limited.

There was a slow trend in the late twentieth century toward surface mining production. Two of the most important reasons for this were the need to mine lower ore grades (ICMM 2012) and the development of new technologies. The former was due to depletion of the richer ore bodies and the higher cost associated with underground extraction methods, which are not economic to produce low ore grades. Regarding the latter, the more efficient extraction of lower-grade deposits using new equipment and new processes, such as the hydrometallurgical methods for copper extraction, has enabled companies to work with lower ore grades than with traditional methods. However, it is important to note that surface mines create much larger environmental footprint than underground mines. For this reason, permit for construction of a new surface mine or expansion of an existing one cannot be obtained easily. In this position, underground mining should be examined.

In the early years of the twenty-first century, new efficient underground methods and equipment have made possible to turn surface mines that had become uneconomical, because of their depth, into profitable underground operations. The ore body in these mines is usually steep dipping and can be mined with the most efficient block caving methods. The competition for land in many densely populated countries has generated the concept that underground mining is the only feasible possibility. However, such developments have halted the growth of surface mining, and it is forecasted that the underground/surface mining ratio 1:6 will maintain in the midterm (Ericsson 2007).

Based on the previous commented factors, the selection of the best mining method for a given deposit, including the choice between surface and underground mining, is a complex process involving the analysis of many interrelated variables. These variables are not only technical, but they include consideration of environmental, social, and political conditions and constraints and of the time and expense required to obtain the government permits. The process is usually iterative in nature, looking at many possible approaches and determining how all the variables interact with each other. Mining companies and consultants commonly use detailed and sophisticated models. These models incorporate all the technical and financial data to provide exhaustive output including items such as mine and mill production, direct and indirect costs, taxes and royalties, and cash flow and risk analysis. Once a mining method has been chosen, the process has not finished because other decisions need to be made such as the specific underground method that will be used or the pit configuration for the surface mining selected, among many others. In this case, the main goal is to maximize ore recovery and minimize removal of waste rock in the most economic, safe, and environmentally sound manner (Stevens 2010).

5.2.1 Stripping Ratio

One of the methods to describe the geometrical efficiency of a surface mining operation is with the term «stripping ratio» (Box 5.1: Stripping Ratio). In-pit design and scheduling is an essential parameter. The stripping ratio commonly increases with the depth of the pit and determines the economic limit of the surface mine. Thus, an increase in the stripping ratio can render the deeper ore uneconomic to mine by surface mining but economic to mine to underground methods. The stripping ratio for metal mines is usually between 1:1 and 3:1 but can exceed 10:1 in mines with high-grade ore. Stripping ratios greater than 20:1 occur in some coal mines (Stevens 2010).

Box 5.1

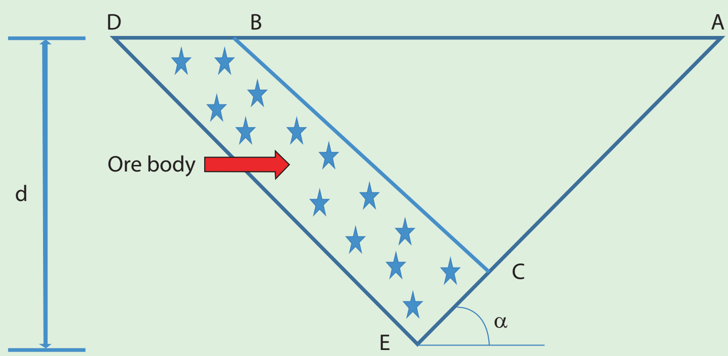
Stripping Ratio

Moving waste material and overburden to recover the ore is known as stripping. Therefore, stripping ratio (a key statistic for mining companies and almost universally used) represents the amount of uneconomic material or waste rock that must be removed to extract one unit of ore or profitable material. The stripping ratio in ◘ Fig. 5.5 (an open-pit with an ore body dipping α) is the ratio between ABC and BDEC. For instance, a stripping ratio of 5:1 or simply 5 means that it must be mined five times more amount of waste than ore. The ratio is commonly expressed as cubic meters/cubic meters, tons/tons, or even in cubic meters/tons for some minerals. If the waste and ore have the same density, it is obviously the same to estimate the stripping ratio in cubic meters/cubic meters or in tons/tons. A wide variety of other units is sometimes used such as overburden thickness/coal thickness or cubic meters/thermal unit in coal mining operations.

The ratio of the total amount of waste to the total amount of ore in an entire mine or from the start of mining up to the moment of the present calculation is defined as the overall stripping ratio. A stripping ratio can also be calculated over a much shorter time span such as 1 year, and this can be referred to as the instantaneous stripping ratio where the instant in this case is 1 year (Hustrulid et al. 2013). Thus, instantaneous stripping ratio is the real relation of the removed waste volumes and the mineral exploited in the pit during a certain and definite period of time. The instant could be defined as a longer or shorter period. For example, the instantaneous stripping ratio for a day in which the mine extracted 5000 tons of waste and 2000 tons of ore will be 2.5. The pit slope angle plays an important role in the estimation of the stripping ratio. Steeper slope angles, common in competent rocks, allow for a lower stripping ratio.

The stripping ratio of a deposit may be used, at least partially, to evaluate how profitable it may be. For instance, a project with a very high strip ratio likely will not be profitable. That is because a high strip ratio means that the unwanted material is much greater than the amount of ore that can potentially be extracted, making it too expensive to mine. Conversely, a project with a low strip ratio will probably have good prospects for profitability. Obviously, since the waste rock must be also drilled, blasted, and hauled out of the pit and this process does not produce any revenue, minimizing the stripping ratio is critical from an economic viewpoint. As a result, mining companies calculate strip ratios for open-pit projects well before they enter development and production and seek out projects with relatively low strip ratios.

◘ **Fig. 5.5** Illustration of stripping ratio concept



5.2.2 Dilution

Moving to a larger scale of operation means less selectivity, hence more dilution. Dilution refers to the waste material that is not separated from the ore, being both mined together. As a rule, dilution varies between 5% and 30%. This waste material is sent with the mineralization to the processing plant. Consequently, dilution increases tonnage of ore while decreasing its grade, increasing operating costs in the mill by incrementing the tonnage of material to be milled. Underestimating dilution can involve a significant risk to a project. For example, a 10% error in copper grade can generate a shift of 60% in the net present value of a project (Parker 2012). Under existing economic conditions, maximum mining efficiency can be defined as 0% dilution at 100% extraction of the mineral being mined. In many projects, it is common to undertake a global dilution such as 5% for massive deposits and 10% for tabular deposits (Ebrahimi 2013).

Dilution can be estimated as the ratio of the tonnage of waste mined and sent to the mill to the total tonnage of ore and waste combined that are milled, being usually expressed in percent format:

$$\text{Dilution} = \frac{\text{Waste (tonnes)}}{\{\text{Ore (tonnes)} + \text{Waste (tonnes)}\}} \times 100$$

Thus, if 100 tons of waste rock are mined with 900 tons of ore and all being sent to mill, dilution is calculated to be 10%. Factors affecting dilution can be divided as deposit related and mine operation related. The first are inherent features of the resource and comprise lithology, structural geology, grade distribution, dip, thickness, and general shape of deposit. Factors related to mine operation include the mining method, mine geometry, mining direction, equipment size, and the skill of operators.

There are two types of dilution: planned or intentional and unplanned or unintentional. In underground mining, planned dilution is usually caused by the design of the stope to improve and stabilize the geometry of the ore due to its irregular shape. Unplanned dilution is usually caused by overbreak of wall rock especially at the hanging wall ore/waste boundary. Generally, it is of the order of 10% but values exceeding 40% have been recorded (Annels 1991). It is almost impossible and costly to eliminate dilution in practice. Some amount of dilution is practically unavoidable in most underground mining operations (Scoble and Moss 1994). However, selective underground mining methods such as sublevel stoping result in a lower rate of dilution than bulk mining methods (Wellmer et al. 2008). In surface mining, dilution can vary in a single mine for different benches and zones.

Dilution can also be classified as internal or external (Fig. 5.6). In the context of using a block model to estimate resources, dilution happens in two different parts. Sometimes within a mining block, there are waste inclusions or low-grade pockets of ore that cannot be separated, and they are inevitably mined with the block. This is called internal dilution and it is difficult if not impossible to avoid.

External dilution, sometimes called contact dilution, refers to the waste outside of the ore body that is mined within the mining block. This type of dilution can be controlled using adequate equipment and mining practices. External dilution is of somewhat less significance in large deposits with gradational boundaries in comparison with small deposits because the diluting material can be a small proportion of the mined tonnage and contains some metal, possibly near the cutoff grade (Sinclair and

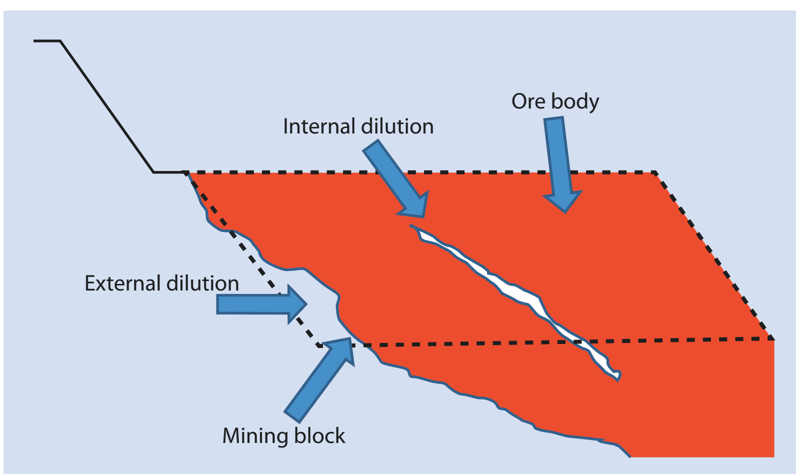


Fig. 5.6 Internal and external dilution (Modified after Ibrahimi 2013)

Blackwell 2002). The local accuracy of external dilution estimate depends on the quality of the geologic model.

5.3 Surface Mining

Surface mining, which is the extraction of mineralization from the ground in mines open to the surface, can be mechanical or aqueous extraction. The former predominates, whereas the latter cannot be employed unless there is sufficient water quantity available. There is a great variation in detail in surface mining, but only some basic techniques are employed, being the terminology more easy to understand than in underground methods. Thus, there are four main mechanical extraction methods to obtain minerals from the ground: (1) open-pit mining, (2) strip (opencast) mining, (3) quarrying mining, and (4) auger mining. In turn, aqueous extraction can be varied: dredging, hydraulic mining, in situ leaching, and evaporite processing. The subdivision of the mechanical extraction methods is clearly related to the commodity mined because open-pit mining is used basically in metals and diamonds (Fig. 5.7), quarrying and mining extract industrial minerals and rocks such as crushed and dimension stone, and strip and auger mining are methods mainly applied to coal deposits.

Open-pit and strip mining are the two most dominant surface mining methods in the world, accounting for approximately 90% of the surface mineral tonnage. The advantages and disadvantages of one type of surface mining versus another are often related to the equipment used and the associated costs and benefits derived from their use (Bohnet 2011). Strip mining has the greatest choice of equipment (e.g., bucket wheel excavators), whereas open-pit loading equipment is usually matched with haul trucks that can be loaded in four passes. In this sense, the life of a mining project is an essential factor to select the most suitable mining method.

Some practical and useful formulas can be provided to estimate the life of a mining project, being the most used those described by Taylor (1977). Considering a wide range of ore body sizes and shapes, the extraction rates seemed proportional to the three-quarter power of the ore

Fig. 5.7 Mirnyy (Yakutia, Russia) open-pit mine to extract diamonds (Image courtesy of Alrosa)

tonnage, and the designed lives were proportional to the fourth root of the tonnage. Thus, Taylor's rule can be formulated as a simple and useful guide that states

$$\text{Life}(\text{years}) \sim 0.2 \times \sqrt[4]{\text{Expected ore tonnage}}$$

In this equation, it is useful to utilize amounts expressed in millions, and except for particular conditions, the common variation is about a factor of 1.2 above and below. The rule can thus be restated as

$$\text{Life}(\text{years}) \sim (1 + 0.2) \times 6.5 \times \sqrt[4]{\text{Ore Tonnage in millions}}$$

The rule offers an adequate output rate for early economic studies. Thus, it will establish a rank of rates for comparison evaluation at the intermediate step after which a preferred single rate can be elected for utilization in the feasibility study (Hustrulid et al. 2013).

5.3.1 Geotechnical Considerations in Surface Mining

The nature of open-pit mining requires the application of sound geotechnical engineering practice to mine design and general operating procedures. Thus, the nature of the geotechnical environment and the resultant geomechanics during excavation is one of the primary influences on mining. It defines where to commence mining in the first place, the choice of mining method, the design of the mine layout, monitoring strategies, and the need for ground control measures during and subsequent to mining.

Understanding the various mining constraints, as a result of the nature of the geotechnical environment, becomes a key mining consideration throughout the entire life of the project (Frith and Colwell 2011). It is clear that the information gained from geotechnical investigations notably provides valuable information for mine design but also assists with the development of mineral resource estimate and ultimately ore reserve estimate. Geotechnical design, monitoring, and stabilization of an open-pit mine are ultimately a matter of economics balancing the benefits and costs of stabilization against the costs and implications of a slope failure (Pine 1992; Wyllie and Mah 2004).

The geotechnical aspects that must be correctly considered during the design, operation, and abandonment of an open-pit excavation are the following:

1. Local geological structure and its influence on wall stability
2. Shear strength of the rock mass and its geological structure
3. A proper analysis of rainwater inflow, surface drainage pattern, groundwater regime, and mine dewatering procedures and their influence on wall stability over time
4. Analysis of open-pit wall stability for the projected geometry of the pit
5. Appropriate drilling and blasting procedures to develop final walls (Fig. 5.8)
6. Appropriate methods of open-pit wall monitoring over a period of time to determine wall stability conditions

In surface mining, slope angles are strongly linked to the geotechnical nature of the overburden or waste. The less competent the overburden, the lower the slope angle must be to maintain an adequate stable and safe pit wall. As the slope angle decreases, the amount of overlying material that needs to be removed to access each ton of mining product increases and, in turn, increases the mining cost. As early as possible in the mine feasibility assessment process, it is crucial to understand and fully consider the interrelationship between the local geotechnical environment and the mining process. Effective ground control is achieved by the successful management of four basic disciplines in an open-pit mine: geology, planning, geotechnical, and production.

Geotechnical Design Process

The geotechnical design process for open-pit slopes, regardless of the size of the pit or materials mined, shall adopt the following strategic approaches: (a) site investigation, (b) formulation of a geotechnical model for the pit area, (c) division of the model into geotechnical domains and design sectors, (d) slope design and stability assessment for the geotechnical domains/design sectors, and (e) design implementation and definition of monitoring requirements (Hoek and Bray 1981).

Fig. 5.8 Appropriate drilling and blasting procedures are fundamental to develop the final walls of the open-pit (Image courtesy of Rio Tinto)

Table 5.1 Classification of some critical geotechnical parameters

Parameter	Very poor	Poor	Fair	Good	Very good
Joint intensity (RQD/Jn)	<4	4–8	8–15	15–25	> 25
Joint shear strength (Jr/Ja)	<0.5	0.5–0.75	0.75–2	2–3	>3
Fracture frequency (FF/m)	>15	3–15	1–3	0.3–1	<0.3
Rock strength (MPa)	<25	25–50	50–100	100–160	>160

Site Investigation

Site investigation is the procedure by which geotechnical and all other relevant information that can influence the design, construction, and performance of the open-pit mine slopes is acquired. Information collected during a site investigation program in the development of a project includes information about the mining history, topography, geomorphology, climate, drainage, physical geology, geological structure, tectonic evolution, lithology, rock mass properties, hydrogeology, and other relevant items to the project. For instance, understanding the cause of the variation in rock mass quality is essential. In addition to the classification ratings, other geotechnical factors can be also defined, and questions about these models commonly help to increase the knowledge of the rock mass variability pattern. Some examples of these critical parameters that are usually interrogated in the modeling process are given in Table 5.1.

Geotechnical Model

The geotechnical model is the keystone in the design of an open-pit mine. The construction of the geotechnical model is an evolving process through the various development levels of an open-pit mine. In many projects, sufficient data to compile a detailed model would only be available at the feasibility or construction stages. At earlier stages such as scoping or pre-feasibility studies, a geotechnical model containing much less detail can only be possible (Box 5.2: Geotechnical Model).

Data compiled in the four models (geological, structural, rock mass, and hydrogeological) are utilized to develop the geotechnical model. This is a stepway procedure of including subsequent layers of individual data sets into a 3-D solid model applying computer-based modeling tools. The geological model, which displays the rock-type limits within the mine, is the beginning point and constitutes the first layer of the geotechnical model. The layers of other information (e.g., rock

Box 5.2

Geotechnical Model

The availability of a comprehensive geotechnical model is the fundamental basis for all slope designs, and it comprises four component models: (1) the geological model, (2) the structural model, (3) the rock mass model, and (4) the hydrogeological model (Guest and Read 2009). Several computer-based modeling tools are available for the development of 3-D geotechnical models. These tools permit visualization and construction of comprehensive models that can include geological and structural information, ore grade distributions, groundwater distributions, and a variety of geotechnical details. Additional information in the geotechnical model includes climate, surface drainage, and regional seismicity. The geotechnical model comprising the four components must be in place before the subsequent steps of setting up the geotechnical domains, allocating design sectors, and preparing the final slope design can start.

The purpose of the geological model (Fig. 5.9) is to permit 3-D visualization of the material types that will be present in the pit slopes. Different material types often have different strength characteristics, which require due attention and consideration in the process of pit slope design. The model should describe the regional and mine site geology and provide clear and unambiguous information on location and extent of different material types. It should represent a broader view of the geology of the deposit, including the surrounding waste rock, focusing on the engineering aspects. This model differs somewhat from that required by mine geologists, whose focus is primarily on mineralization (Read and Keeney 2009).

The aim of the structural model is to describe the orientation and spatial distribution of the discontinuities that are likely to influence the stability of pit slopes. These discontinuities can be divided into two groups: (a) large structural features such as folds and faults that are widely spaced and continuous along strike and dip across the entire mine site (major structures) and (b) closely spaced joints, cleavage and faults, etc., that typically do not extend for more than two or three mining benches (minor structures).

The rock mass model represents the engineering properties of the rock mass. It comprises the various material types and structural defects in which the open-pit slope will be excavated, the rock mass properties including the properties of the intact pieces of rock, the structures that cut through the rock, and the rock mass itself. These properties govern the performance of the slope and therefore the design approach. In a slope constructed in hard rocks, failure could occur along geological structures, which are considered as pre-existing planes of weakness. In relatively weak materials such as weathered or soft rock, failure can propagate through the intact material and/or along geological structures. Therefore, it is essential to determine the engineering properties in the various geological units present in a pit slope.

Regarding the presence of groundwater in a pit slope, it can have significant negative effects on its stability. In the case of open-pit mines excavated within weak materials such as clay or completely weathered rock, pore pressures play a significant role on the stability of pit slopes. High pore pressures reduce the effective stresses with concomitant reduction in shear strength of both soil/rock material and rock mass. This could lead to instability in the pit slope. High water pressures also reduce shear strength of structural defects in unweathered strong rock, leading to structurally controlled instability. Groundwater can also create saturated conditions and lead to water ponding inside the pit, which in turn can lead to unsafe working conditions. Other problems that could result from saturated conditions or standing water in the pit include loss of access to all or part of the pit, difficulties in the use of explosives for rock blasting, and reduced efficiency in the mining equipment. Thus, it is essential to develop a good groundwater model at early stages of any open-pit mining project so that effective control measures can be designed and implemented to minimize the adverse effects of the groundwater regime. In open-pit mines excavated below the groundwater table, dewatering or depressurization can be necessary for the abovementioned reasons (Kroeger 2000; Beale 2009).

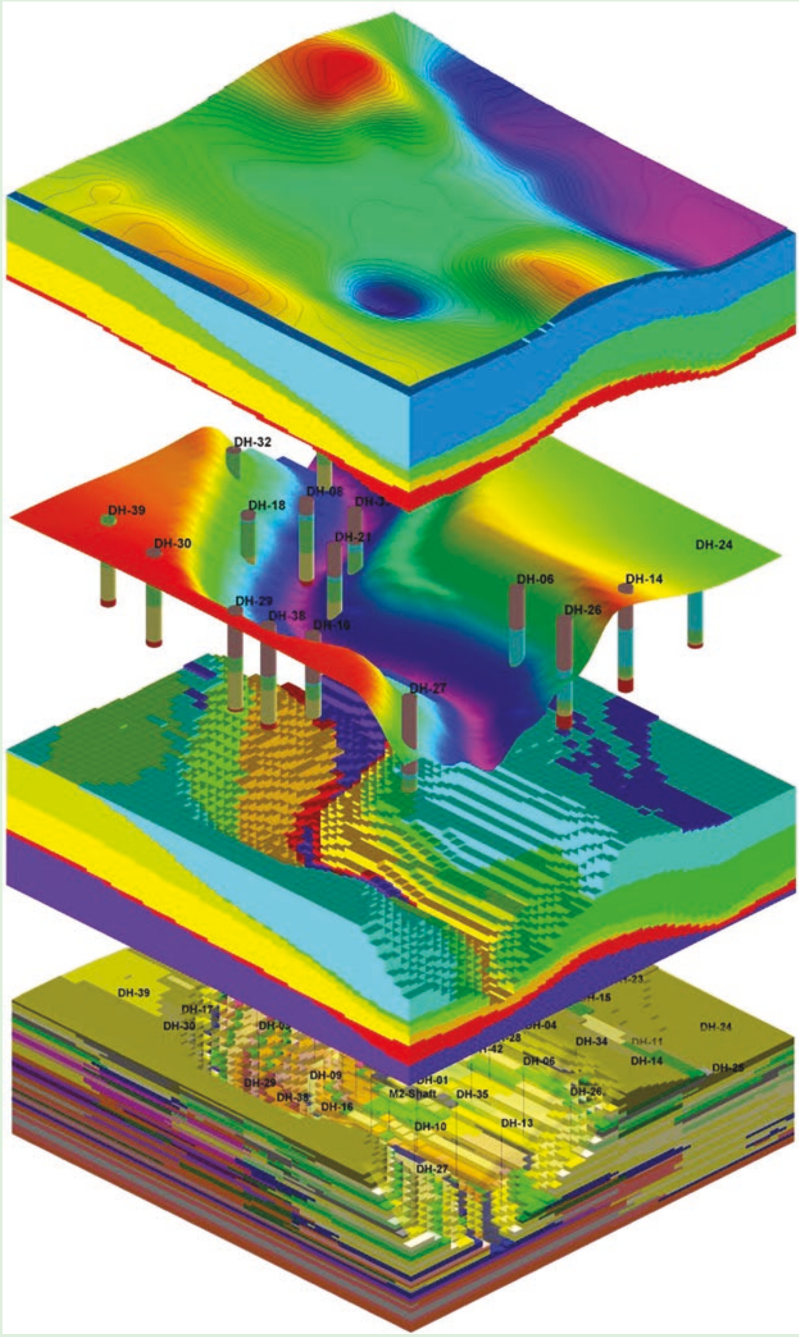


Fig. 5.9 Geotechnical composite from stratigraphic block model (*top*) to lithologic model (*bottom*) using RockWorks 17 (Illustration courtesy of RockWare)

mass weathering, structural data, hydrogeological information, among others) can then be attached. As aforementioned, the readiness of a supportive geotechnical model is the essential basis for all slope designs.

Geotechnical Domains and Design Sectors

Before the slope design and stability analysis can start, the pit is split into several geotechnical domains, each with its own geotechnical features

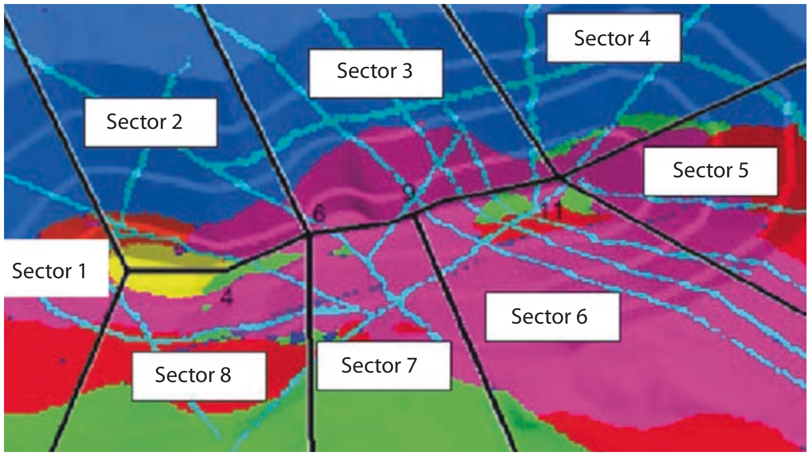


Fig. 5.10 Pit geotechnical domains based on geometric, geological, rock mass quality characteristics, and hydrogeological considerations (Illustration courtesy of KGHM)

that are distinct from those of its neighbors (Fig. 5.10). These features will define the stability based on the orientation of pit slopes. The amount geotechnical domains significant to pit wall design can change depending on the characteristics of the mine. Thus, several domains can be necessary to outline a large mine excavated in a complex geotechnical environment.

Geotechnical Slope Design and Stability Analysis

The geotechnical slope design is the process of determining the optimum slope angles and dimensions for open-pit mines. Designing a geotechnical model is one issue, but implementing the information it includes to the slope design is another (Guest and Read 2009). In open-pit mining, there is a general tendency to increase the slope angle as an attempt to decrease the stripping ratio, which in turn can originate higher return on investment. However, an increase of the slope angle decreases the stability of the slope, and it could lead to safety implications and higher operating costs due to slope failures. Thus, the slopes must be constructed to an optimum angle without compromising both safety and economics.

Regarding the design acceptance, a slope in mining is defined as stable if the forces resisting the potentially shearing, sliding, or toppling mass of material on the slope are greater than the forces driving the mass. The ratio of the resisting forces to the driving forces is termed the factor of safety (FOS) and has been the basis of stability acceptance criterion for many engineering applications. Where FOS = 1, the slope is considered to be in a state of limiting equilibrium, while if FOS > 1, the slope is considered to be theoretically stable. There are no strict criteria that specify the acceptable FOS, but for static loading conditions, values of 1.2–2.0 are commonly used depending on the type of slope and its importance.

Implementation of the Slope Design

The implementation of the design typically involves minimizing unnecessary damage to slopes during blasting, excavation control and scaling, groundwater and surface water control, as well as installation of ground support and reinforcement. These measures are added to the production cost, but they are required to improve stability. For example, poor blasting procedures near mine slopes can originate loose rock on slope faces and batter crests, overbreak in the slope face, and cumulative depletion in the strength of rock mass in which the slope is developed. Performance monitoring of open-pit walls is required to check the geotechnical parameters and assumptions utilized to design the existing walls, to assure that any potential falls of ground are identified previous to them becoming harmful, and to set correct plans where ground movements are detected.

Slope Monitoring

In an active excavation, slope monitoring is crucial in predicting and preventing slope failures, and, when failure is imminent, mitigating the effects of a slope failure. A comprehensive slope stability monitoring program reduces the risk of major production delays or even sterilization of part of a

Fig. 5.11 Ground stability radar device used to monitor the movement on the highwalls of the pit (Image courtesy of Anglo American plc)

reserve permanently as a consequence of slope failure. Moreover, it ensures overall safety of personnel and equipment in operation (Wetherelt and van der Wielen 2011). Another situation in which pit slope monitoring is essential is the presence of active underground workings in close proximity to an open-pit mine. Crown pillar failure or caving-related subsidence can permanently cease surface excavation activities.

Slope stability monitoring techniques can be divided into surface and subsurface monitoring techniques (Wyllie and Mah 2004). Surface monitoring techniques include visual survey, direct measurement techniques, prism monitoring, laser systems, and radar systems (Fig. 5.11). Direct measurement techniques include crack width meters, tilt meters, and other similar devices. Subsurface monitoring techniques include time-domain reflectometry, borehole probers, extensometers, and inclinometers. These techniques rely on measurement of changes of the inclination or other characteristics of a borehole that could indicate deterioration of stability. Additionally, seismic monitoring techniques can be used. These rely on geophones registering acoustic emissions associated with failure events. The most cost-effective approach to slope stability monitoring is generally a combination of several of these techniques where they are used to complement one another. For instance, laser systems or prism monitoring can be used to determine overall stability of pit slopes and identify possible failure zones. If instability of a slope is detected, extensometers or radar systems can be used for more precise determination of movement in this area.

A last important consideration in slope stability is the presence of groundwater. Piezometers are the main tool for determining groundwater level. These, together with rain gauges, can act as an early warning system and serve as a basis for adjustment the rate of water extraction from dewatering wells to prevent groundwater-induced failures.

5.3.2 Surface Production Cycle

The surface production cycle of unit operations for metal and many nonmetal mines commonly consists of drilling, blasting, loading, and hauling (Fig. 5.3). Open-pit mining deals with the extraction of topsoil and overburden, blasting of ore, and the transportation of material using a system of shovels or excavators and haul trucks. Once the haul trucks have been loaded, they transport the material out of the mine to a dumping location where the material will either be stored or further processed. The trucks then return into the mine and the cycle repeats itself. Note that each of the unit operations cannot begin its handling of the mineral product before the previous unit operation has completed his work.

Equipment Selection

In the last 50 years, mining equipment and especially trucks have progressively increased in size and capacity because experience has demonstrated that larger equipment has diminished total cost by enhancing productivity in big mines. To date, in terms of productivity, the mining industry continues to adhere to the «bigger is better» mentality. In mine operations, drills, loading

5

machines, and haul trucks comprise the major cost items (Box 5.3: Equipment Selection Problem).

The increasing size of mining equipment has occurred in parallel with the addition of new technologies that have brought noticeable changes in the mining industry in the last 20 years. An example of these new technologies is the dispatching and global positioning systems for fleet management such as the Dispatch System (Modular Mining Systems, Inc.), which provides optimization of the truck locations in real time, thus decreasing truck queuing and shovel hang time (Fig. 5.13). This process is accomplished with sensors integrated into the vehicle design.

In conclusion, many factors must be considered in selecting the most economic fleet of equipment. Where there are a number of options, the best is to select the one offering the highest degree of flexibility to the mining operations and surviving future crisis (Bohnet 2011).

Loading Equipment

Loading or excavating is the third main stage in the production cycle of a mine. These terms are not synonymous, but they are commonly used interchangeably. However, the term loading is commonly utilized to indicate that the material is placed in a haulage device. Once the rock has been fractured by drilling and blasting, loading process

Box 5.3

Equipment Selection Problem

The purpose of equipment selection is to select optimum equipment with minimum cost. Type, size, and number of units are major considerations, being these three items strongly interdependent. The dimensions of a machine and its production rate are important factors in equipment sizing, although larger dimensions and increased productivity do not necessarily go hand in hand. Optimal fleet size can be estimated based on production tonnage requirements and individual truck productive capability. Thus, the main goal of the equipment selection process is to satisfy the production rate requirements while minimizing the mining cost. The selection of equipment also influences the open-pit optimization process. For example, one of the inputs in open-pit optimization is mining costs, which is influenced by the kind of equipment that is purchased. Consequently, equipment optimization and open-pit optimization are closely linked. Moreover, the size of the operation over time has a direct bearing on the type of equipment selected.

Loading and hauling fleets are dependent fleets and therefore the effectiveness and availability of each affect the fleet requirements of the other. For example, if a mining operation has only one large shovel, the truck production would only be effective where the shovel is operable and broken rock is available at the shovel face for loading. Therefore, one of the challenging problems for surface mining operation is to choose the optimal truck and loader fleet (Fig. 5.12). This problem is called the equipment selection problem (ESP). The inputs to the ESP are (a) a long-term mining schedule, including production requirements at a number of loading and dumping locations, (b) a set of loader and truck types that can be purchased, (c) equipment productivity information and how this changes when equipment operates with different types of equipment, and (d) cost information, including interest and depreciation rates, purchase, maintenance, and operating costs (Burt and Caccetta 2014). The output from an ESP is a purchasing strategy or policy as well as additional information such as how the equipment should be used with respect to defined tasks.

The size selection criteria for loading and haulage machines are not the same. The size of the loading machine is an important factor in selective mining and prevention of dilution. In addition, the loading machine initially determines the productivity of a mining system. In an open-pit mine, the number of loading machines is limited and their reliability and flexibility are very important. Thus, the mining selectivity, productivity, reliability, and flexibility are essential factors for loading machine selection. Regarding the size of haulage machines, it directly influences the mine layout and design, and loading and haulage should be adequately matched. Haulage costs are usually twice the cost of loading; consequently, greater attention must be paid to truck selection.

The number of constraints determining the type of equipment selected is greater for loading machines than for haulage machines. For instance, selectivity and the amount of dilution are important factors for sizing loaders, whereas they are not important for sizing the haulage fleet. Loading machines are also more sensitive to flexibility and reliability than haulage machines. Other relevant factor in the equipment selection process is the compatibility of the loaders with selected truck fleets. For example, some loaders cannot reach the top of the tray on the larger trucks. Conversely, some loader capacities exceed the capacity of the truck.

Fig. 5.12 Optimal combination of a large shovel and truck (Image courtesy of Anglo American plc)

Fig. 5.13 Modular dispatcher, Dispatch System (Image courtesy of Modular Mining Systems Inc.)

begins. Mining systems can generally be classified as continuous or cyclic. Continuous excavation systems used in surface mining include bucket wheel excavators and bucket chain excavators or dredges, commonly applied to brown coal mining for power generation, essentially large-volume surface mining operations. Cyclic excavation systems include shovels, hydraulic excavators, draglines, and wheel loaders that are applicable for a large range of operational scales, commodities, and surface mining configurations. Because all loading tools in cycling excavation systems perform basi-

Fig. 5.14 Bucket wheel excavator (Image courtesy of ThyssenKrupp)

cally the same function – they load trucks – differences lie in features such as capacity, mobility, flexibility, life, and support requirements. The choice of loading unit is dependent on the minimum number of active works areas, ore selectivity, and total production and blending requirements.

Continuous excavation systems are generally matched to continuous transport systems such as belt conveyors or pipelines. There could be also applications where continuous excavation could be matched to cyclic transport with mining trucks, but operational life or production rate cannot justify investment in continuous transport. Cyclic excavator systems can be adapted to continuous transport systems particularly for large-scale long-life deep open-pits where waste and ore transported on conveyors, generally after crushing to conveyable size, is the best economic solution. Most often, cyclic excavation systems are matched to cyclic transport systems, typically conventional loading equipment loading mining trucks.

Excavation equipment can be evaluated in terms of productivity (metric tons per hour) and efficiency (cost per metric ton). Important factors in achieving acceptable productivity and efficiency from excavators are matching the trucks to excavator sizes, ideally three or four loading passes, selecting the right excavator for the bench height, and providing enough working space for the excavator and trucks to operate (Hustrulid et al. 2013).

Bucket Wheel Excavators

The bucket wheel excavator (BWE) is the most powerful tool for mining in unconsolidated and soft rock (Fig. 5.14). It is commonly used in coal seam mines, reaching daily outputs of up to 250,000 m^3. BWE combines three parts of the mining process in one machine: extraction, loading, and transportation to the conveyor. The in-pit conveyor system then transports the excavated material to the dumping site or stockpile. For purposes of comparison, BWE is a high-capital, low operating cost that has limited flexibility and can operate through a limited range of applications with sensitivity to geological variance (Humphrey and Wagner 2011). These machines are highly customized and vary in design more than do any other mining machines, to the extent

■ **Fig. 5.15** Electrical cable shovel (Image courtesy of Codelco)

that nearly every machine is almost unique. They are very robust in design and consequently very long lived, although very expensive. Due to the associated conveyors, BWE requires linear, flat-floored mining faces that advance in straight or radial patters. For this reason, major application of this equipment has historically been large lignite mines.

Shovels

In mines of medium to large size, the principal loading equipment is the shovel. It can be hydraulic or electrical and bucket sizes range from 15 to 70 m^3. The electrical cable shovel or rope shovel (■ Fig. 5.15) continues to be the primary loader of selection for large open-pit mines. Although initially costly, these machines have the productivity, ruggedness, and longevity required by mining operations to reliably load broken rock into large trucks for haulage to processing plants or waste dumps over the life on an operation. The dependence on the trailing cable somewhat limits mobility; its handling can be facilitated with the utilization of special cable handling trucks. The advantages of electrical cable power are the effective use of power, the credibility of the system, and the monitoring equipment (Hustrulid et al. 2013).

The productivity and unit cost efficiency demanded of the mining industry have resulted in substantial increases in electrical cable shovel size in the last decades. Shovel capacities have grown tenfold in the past 50 years. In the 1960s, a typical shovel had a 4.5 m^3 bucket to load a 30 ton truck; actually, machines with as large as a 70 m^3 bucket load trucks of 400 tons or more. Compared with rope shovels, hydraulic shovels have less reach, so they move more often, but they travel at higher speed and do not require assistance with a trailing cable or cable bridge. The growth of loader machines has led to a new delineation in the loader market, with wheel loaders predominant at the lower range of bucket capacity, hydraulic shovels in the middle range, and electrical shovels at the upper end of bucket capacity.

Hydraulic Excavators

For hydraulic excavators, face shovel (■ Fig. 5.16) and backhoe configurations are available. Face shovel configurations are preferred in harder rock and with higher rock faces, whereas backhoe configurations allow for more selective digging and faster cycle times. Hydraulic excavators can be diesel or electrically driven. They are somehow similar to hydraulic shovels and even both terms are often used interchangeably.

Draglines

Draglines (■ Fig. 5.17) are self-contained systems that load and transport material to a dump point. They are highly productive, comparatively low in operating cost and labor requirements, and extremely robust. Consequently, they have very long

Fig. 5.16 Face shovel (Image courtesy of De Beers)

Fig. 5.17 Dragline operating in a phosphate mine (USA) (Image courtesy of PotashCorp)

lives, commonly 30–40 years. Because of their high productivity and capability of direct disposal of material, draglines are favored for area mining (see section «Strip Mining») in areas of flat-lying tabular geology with high production requirements. The most common application for large draglines is overburden removal in coal mining, having up to 125 m^3 bucket size. The bucket is pulled by a dragrope over the face toward equipment itself, hence the name dragline. In most basic dragline operations, the machine removes overburden material to uncover ore that is the most recent in a series of parallel adjacent pits (narrow and relatively long). Overburden material from the current pit is placed in the previous adjacent pit, from which product has been removed by auxiliary equipment.

Wheel Loaders

Wheel loaders are used in soft to hard formations and are forthcoming with small bucket size of 0.5–20 m^3. These units are commonly wheel mounted, but a few models are also offered with crawler mounting for their use in problematic terrain. Large wheel loaders are often favored as support loading equipment because of mobility advantage. They can more readily clean up small quantities of batter trimmings. Wheel loaders can be applied as prime

Fig. 5.18 Special tow truck transport medium for mining equipment (Image courtesy of Eduardo Revuelta)

loading equipment where mobility, in-pit blending, and multi-material selectivity are major issues, particularly in shallow open-pit operations such as lateritic ores, bauxite mining, and the like (Hardy 2007). More recently developed or upgraded large wheel loaders have the advantage of faster digging cycles that more closely approach a rope or hydraulic shovel.

Hauling Equipment

The fourth and last stage of the production cycle in a mine is haulage; hoisting is the term used where essentially vertical transport is accomplished. In surface mining works, truck haulage is the biggest factor in the operating costs, forming from 50% to 60% of the global costs (Ercelebi and Bascetin 2009). Off-highway trucks (they must be translated – as well as other mining equipment (Fig. 5.18) – in a special tow truck transport medium) have dominated haulage in surface mining operations for many years. Some decades ago, a few mines worldwide utilized rail haulage and it is still being used. An essential feature to be considered in terms of rail transportation (Fig. 5.19) is the requirement to ensure almost completely horizontal track placement. This status has largely restricted opportunities to employ this type of haulage. The application of rail haulage only has economic sense if the distance transported is appropriately long (Czaplicki 2009).

Longer haulage distances in many large pits, availability problems with haul trucks, and improvements in technology have revived interest in in-pit crushing and conveying (IPCC). Since mineral and waste transportation costs include the greatest amount of surface's mine working costs, in-pit crusher and a conveyor belt, instead of truck transport, can reduce this cost aspect, mainly where pits have become deeper. This system uses a crusher/sizer unit to process material from a cyclic loader to a size that is suitable for conveyor transport, extending

Fig. 5.19 Rail transportation of ore (Image courtesy of BHP Billiton)

the application of around the pit conveyor systems to include consolidated waste and overburden (Humphrey and Wagner 2011).

IPCC (Fig. 5.20) is preferred for the material handling transportation system where long-term planning is possible. The in-pit crushers systems developed and operated to date have varying degrees of mobility ranging from fully mobile units to permanently fixed plants, which resemble traditional in-ground plants. There are many advantages of IPCC as compared to truck haulage. Some of them, among other, are the following: (a) cost by shortening the haulage distance between the loaders and crushing plant is reduced; (b) other costs such as operating costs associated with fuel, tires, and lubricants or labor costs are reduced; (c) safety risks are reduced; (d) belt conveyors can traverse grades of up to 30° versus approximately 10–12° for trucks; (e) CO_2 emissions are greatly reduced; and (f) conveyors are more energy efficient than trucks and require less skilled labor for maintenance (Utley 2011). Whether IPCC is economically viable is a function of production, duration of the operation, and the distance and vertical lift of the haulage route.

Trucks

Over the years, off-highway trucks, also called mining trucks or haul trucks, are primary means by which both ore and waste are transported in large open-pit mines. Mining trucks haul the material from the loader to a dumpsite. In simplest terms, a haulage truck is a container (the body) on drive wheels. The amount of material transported per cycle relies on the size of the container used. The trucks have capacities ranging from less than 40 tons per load to 400 tons per load in large trucks such as Caterpillar 797 and Liebherr T 284 or 450 tons per load in BelAZ 75710. Haulage truck capacity is usually measured on the basis of weight rather than volume to prevent overloading. It must not be forgotten that loading capacities are measured in volume. In this sense, the speed at which the truck can transport the material is inversely proportional to its capacity.

Rigid-frame haul trucks (Fig. 5.21) have dominated haulage in mining operations, although articulated dump trucks have proven to be a viable alternative. The truck drive systems can be broadly divided into mechanical drive, with automatic transmission, and electric drive (AC or DC) in

Fig. 5.20 In-pit crushing (Image courtesy of Octavio de Lera)

Fig. 5.21 Rigid-frame haul truck (Image courtesy of Anglo American plc)

which the engine drives a generator that powers the electric motor used for traction. Electric drive is currently becoming the predominant type because of its superior drive performance, ease of maintenance, and cost advantages made possible by rapid advances in technology.

Because haulage costs are very high, a thorough understanding of truck haulage is imperative. Most mines are designed to minimize the travel distance between the loading unit and the crusher in order to reduce the number of trucks in the fleet, decrease wear on the truck, and limit the

Fig. 5.22 Open bowl scraper

round trip time for each load. Two important tasks must be undertaken for the proper use of haulage trucks. First, the trucks must be matched in size to the excavator. Second, the number of trucks in the fleet must be matched to the haulage layout so that the system produces in a near-optimal manner.

Wheel Tractor (Open Bowl) Scrapers

One of the oldest concepts of bulk material handling is the wheel tractor or open bowl scraper (Fig. 5.22). Today it is the only machine that can self-load, haul, and dump with a single operator. Mobility and flexibility are key characteristics of this type of equipment, which makes it ideal for small, short-life mining projects. Its capability to remove and place material in controlled lifts makes it the machine of choice for topsoil relocation in reclamation operations.

Auxiliary Operations

Auxiliary operations consist of all activities supportive of but not contributing directly to the production of ore. Since this unit operations do not generate incomes, there is a tendency in mining organizations to assign them a staff function and a low priority. However, efficient operations depend largely on auxiliary operations. Many of these operations common to mining are classified as supportive on the extraction function, but others are associated with development and reclamation operations. Two examples of auxiliary operations equipment are the track dozers and the motor grades.

Track Dozers

Large track dozers are extremely common in all mining operations. They are designed to move the greatest amount of material in the most efficient way and generally are used for both utility and production work. Utility work includes tasks that support a mine's main production fleet such as dumpsite preparation and cleanup, bench preparation, road creation, stockpile work, and reclamation. These machines can also develop production works such as excavation and to rip in situ or blasted material from one area to another. Track dozers are complex machines due to their variety of mechanical, electrical, and hydraulic systems all fitted into a compact design. Most of the industry commonly uses the smaller size of track dozers because of their lower operation costs and flexibility.

Motor Graders

In good road conditions, trucks run faster and more safely, fuel costs are lower, and tire damage is reduced as truck maintenance is. Since loaders and haul trucks are responsible for producing ore, motor graders (Fig. 5.23) generate a clear impact on how productively these machines can operate, especially in their role in haul road maintainability. Thus, they are some of the most productive, and productivity

■ **Fig. 5.23** Motor grader

■ **Fig. 5.24** Control of dust generation (Image courtesy of Anglo American plc)

enhancing, machines on site. Motor graders are designed to meet the specialized requirements of large mining operations. They help to create and maintain constant grade and proper drainage. By using the blade incorporated in the machine skillfully, it is also possible with motor graders to finish a slope. Finally, since dust in a mining operation presents one of the most visible and invasive operational constraints, a haul road dust prevention system is essential to avoid dust lift-off (■ Fig. 5.24).

5

5.3.3 Surface Mining Methods

Open-Pit Mining

Open-pits can be in the form of inverted, truncated, or circular cones where the radius of each circular bench decreases with depth (Figs. 5.7 and 5.25). Thus, mining occurs in successively narrower benches in order to maintain safety and stability inside the mine. It is important to note that each successive bench in the mine is smaller than the last one developed, which causes the depth of the pit to be mined is determined by the size and location of the first cut or bench.

In stratiform deposits, shallow and large open-pits can be designed in the shape of footprints with steep sides and flat bottoms. However, some pits can be very deep, up to 1 km. The objective is to extract both metallic and nonmetallic ores while dumping overburden and tailings at a dedicated disposal site outside the final pit boundary. Open-pit is utilized where the ore body is typically pipe-shaped, vein-type, steeply dipping stratified, or irregular. Iron, copper (Fig. 5.25), and gold mineralization together account for most of the total open-pit excavation volume in the world. Application of this mining method to coal is less common.

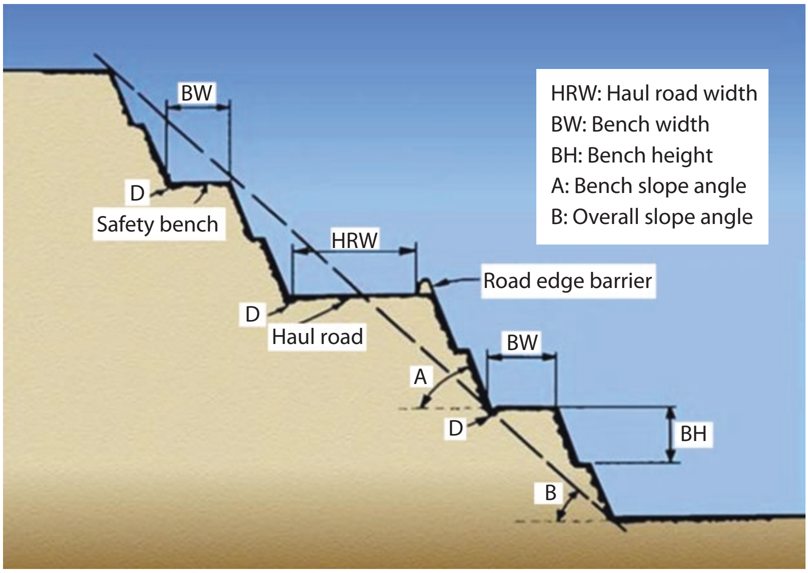
Many features define the size and shape of an open-pit, and these must be correctly understood and utilized in planning of any open-pit operation. The importance of each factor is based on the particular project, but the following are the key characteristics affecting pit design: geology, grade and localization of the mineralization, extent of the deposit, topography, property boundaries, bench height, pit slopes, road grades, mining costs, processing costs, metal recovery, marketing considerations, stripping ratios, and cutoff grades (Armstrong 1990).

Open-Pit Geometry

Mining geometry is a dynamic rather than a static concept because the different geometries are very important so that the needed economic result, revenue and costs, is realized. To evaluate the large number of possibilities, the utilization of computers has become invaluable. In open-pit mining, the ore body and the associated waste are

Fig. 5.25 Bingham Canyon (USA) Copper open-pit mine (the largest and deepest excavated hole in the world) (Image courtesy of Atlas Copco)

■ **Fig. 5.26** Some concepts of open-pit design



extracted from the top down in several horizontal layers of similar thickness called benches (■ Fig. 5.25). Thus, benches are the main extraction components in an open-pit mine and possibly the most distinguishing feature. They are crucial in an operation as they accommodate the active drilling and blasting and excavation areas. Each bench has an upper and lower surface separated by a distance equal to the bench height. Thus, the bench height is the vertical distance between the highest point of the bench (crest) and the lower point (toe) (■ Fig. 5.26). In general, all benches in an open-pit have the same height, but geological conditions can dictate the opposite.

There are several types of benches: (a) working benches, which are in the process of being mined; (b) inactive benches, the remnant of working benches left in place to maintain pit slope stability; and (c) safety or catch benches, with a purpose of collecting the material which slides down from benches above. Constituting one of the busiest areas of an open-pit, working benches have to accommodate large excavators and dump trucks as well as the muck pile formed after a blast. Therefore, maintaining the quality of a suitable working surface is vital to ensure acceptable safety and productivity levels at an active excavation. In addition to leaving the safety benches, berms or piles of broken materials are often constructed along the crest to form a ditch between the berm and the toe to catch falling rocks. A berm is also a horizontal shelf or ledge at the ultimate pit wall slope. The design of all these components is controlled by the geotechnical configuration of the slope.

Several features can influence the selection of bench dimensions. This includes ground competence, existence of water, presence of geological disturbances (e.g., faults, joints, bedding planes), and cutting height of the excavator (Tatiya 2013). Bench height decision is essential since once this value is determined, the rest of the dimensions in the open-pit follow consequently. One of the most usual bench heights in large open-pits is 15 m (Hustrulid et al. 2013), although for smaller pits the value might be 12 m or less. A common guideline is that the bench height must be matched to the loading equipment. Regarding the bench width, it varies according to equipment size and the type of bench. Working benches should at least be wide enough to accommodate the turning radius of the largest haul truck plus the width of the safety berm. Bench width commonly ranges from 30 m to several hundred meters.

Hustrulid et al. (2013) recommended the following steps where considering bench geometry:

1. Mineral deposit characteristics such as total tonnage and grade distribution dictate a certain geometrical approach and production strategy.
2. The production strategy yields daily ore/waste production rates, selective mining and blending requirements, and number or working places.

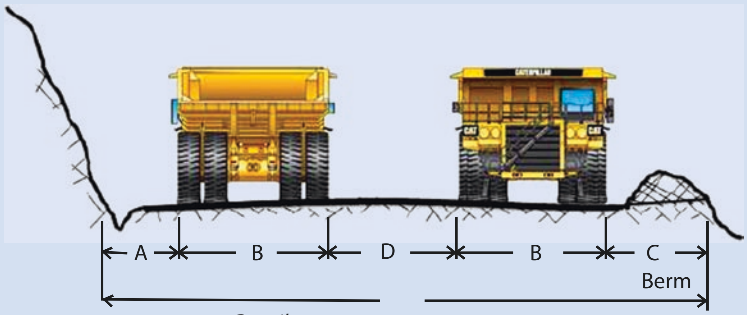


Fig. 5.27 Example of ramp width according the type of equipment

Calculation	*Details*	*Measurement (m)*
A	Safety distance	8
B	Truck width	7.3
C	Berm width	3.7
D	Distance between trucks	3.7
E	Slope width (A+2B+C+D)	30

3. The production requirements lead to a certain equipment set (fleet type and size).
4. Each equipment set has a certain optimum associated geometry.
5. Each piece of equipment in the set has an associated operating geometry.
6. A range of suitable bench geometries results.
7. Consequences regarding stripping ratios, operating vs. capital costs, slope stability aspects, etc., are evaluated.
8. The «best» of the various alternatives is selected.

In turn, the slope of the pit wall is one of the major elements affecting the size and shape of the pit. The pit slope helps to determine the amount of waste that must be moved to mine the ore. It is usually expressed in degrees from the horizontal plane. The global height, from the toe to the crest, is the overall pit slope. Thus, the overall pit slope angle can be established as the angle calculated in degrees constituted while joining the toe of the lowest bench to the crest of the top most bench of a pit where benches achieved their ultimate final designs. Slope angle is clearly an essential parameter that has meaningful economic impact. If the slope angle is too steep, the pit walls can collapse; if it is too shallow, excessive waste rock must be removed. The exposed subvertical surfaces of the benches are called the bench faces. They are described by the toe, the crest, and the face angle. It can vary considerably with rock characteristics, face orientation, and blasting practices. Normally, bench faces are mined as steeply as possible. In most hard-rock open-pits, the face angle ranges approximately from 55° to 80°. Mining starts with the top bench, and after a sufficient floor area has been exposed, extraction of the next layer can start. In most mines, the top few benches are commonly formed by soil and overburden material. Pre-stripping is the term used to refer the removal processing of these materials. The entry to the pit is generally defined utilizing a ramp or road that can be spiral around the pit or situated on one side of the pit with switchbacks at each end.

It is important to note that haul roads and ramps connect the benches, allowing equipment to move freely about the pit and for ore and waste to be hauled out of the pit. Thus, haul roads constitute a key element of an open-pit mine. Haul roads can significantly impact pit angles and stripping ratios depending on the adopted design and geometry. As such, sound haul road design and management can play a significant positive influence on the safety record, profitability, and environmental impact of a mine (Wetherelt and van der Wielen 2011). The width and steepness of the haul road or ramp are based on the type of equipment to be placed (Fig. 5.27). According to the location and use, haul roads are generally around 3–3.5 times wider than the largest truck size on two-way straights. For one-way haul roads, a width of 2–2.5 times that of the largest truck size is generally enough. Any change in the road width will directly affect the overall pit wall slope and increment drastically the stripping ratio, particularly in deep open-pit mines (Bozorgebrahimi et al. 2003).

Topsoil and Overburden Disposal

In mining, overburden refers to all unprofitable material that needs to be excavated to access an ore deposit. If overburden is encapsulated between two layers of ore, it can be referred to as interburden. Overburden forms, by far, the largest volume of material produced by most open-pit mines. In which concerns material handling, there are three important differences between ore and overburden: (a) overburden is not beneficiated and will generally not generate any revenue; (b) overburden tonnages almost invariably exceed ore tonnages in an open-pit mine; and (c) the rock mass characteristics are often different from that of the ore. The first two points imply that handling of overburden and related costs should be kept to a minimum. Furthermore, overburden can contain sulfides or other substances that are potentially damaging to the environment. Consequently, selection of the most suitable site for the overburden embankment involves a trade-off between handling costs related to overburden disposal and the environmental impact of the overburden at a particular site. To minimize costs related to the handling of overburden, it is often blasted to a coarser fragmentation than ore and in many cases excavated and hauled by larger-capacity equipment.

The overburden removal system should be in harmony with the mine planning of the future. Unless this and the other factors influenced by overburden removal schemes are carefully considered, operation can suffer in efficiency and costs (Aiken and Gunnet 1990). Minimizing costs involves selecting and overburden embankment site in close proximity to the mine where the environmental impact is as small as possible. Preferably, this site is close to the projected final pit limit, at the same or at lower elevation than excavation to minimize upslope haulage costs. With these considerations in mind, optimization of overburden management at a mine site can have considerable positive influence on the environmental impact and economic viability of a mine (Wetherelt and van der Wielen 2011).

Topsoil is the near-surface portion of the material lying above the ore so that it has sufficient agricultural nutrients to support varying degrees of vegetation growth. It is included sometimes in the overburden, although the differences are clear. Both are not ore but the topsoil has an essential end use. Depending of climate, topography, and bedrock geology, the layer or layers of soil can be just a few centimeters deep or extend to several meters in depth. In many operations, topsoil storage is required for reclamation purposes at the end of the mine life. Thus, soil and growth media are commonly stockpiled on the mine site for further use in reclamation. To replace topsoil as it originally existed on mined areas requires that each layer be carefully excavated and placed in an area of easy recovery. In some cases, separate storage of different topsoil and subsoil layers can be necessary to ensure quality of the material. Depending on the duration of topsoil storage, revegetation and erosion control can be required.

Because of the unconsolidated nature of the topsoil, it often requires different excavation techniques. As topsoil is generally free digging, scrapers, bulldozers, front-end loaders, and small hydraulic excavators are the most common equipment used in topsoil removing. Bulldozers can be used for pushing materials onto piles for further excavations by front-end loaders or hydraulic excavators. Alternatively, they can support scraper operation by ripping soil. Haulage distance is an important consideration in choice of equipment. At short haul distances, scrapers and bulldozers are the best option, whereas a more conventional excavator/truck haulage operation tends to be more economical at longer haul distances (Wetherelt and van der Wielen 2011).

Open-Pit Design and Optimization

The management of a large open-pit mine is a huge and complex tax, especially for long-lived mines. Thus, one of the most important economic inputs of an open-pit is the calculation of the final pit limit. It is the consequence of mining the amount of material that originates the global maximum benefit while fulfilling the operational needs of safe wall slopes (Cacceta 2007). The ultimate pit limit evidently gives the size of the mine at the end of its life. This limit of the open-pit must be set at the planning stage and defines many significant features such as the amount of mined mineralization, the metal content, and the amount of waste. Other similar terms for this concept are pit outline or pit contour.

Predictably, the size, geometry, and location of the ultimate design of the pit are important in planning tailing areas, waste dumps, access roads, concentrating plant, and all other surface facilities. The open-pit mine design issue is thus to decide the blocks of a mineral deposit to extract to maximize the total profit of the mine while fulfilling digging constraints on pit slope and those that enable underlying blocks to be extracted only after blocks on top of them. The ultimate pit only exists if mining stops, that is, up until that time its final form is uncertain. Consequently, the utilization of the term final pit is discouraged (Whittle 2011).

Open-Pit Design

There are many ways of designing an ultimate open-pit, differing by the size of the mineral deposit, the features of the data, and the accessibility of computer mining software (see ▶ Chap. 8). As a rule, two methods can be broadly outlined: manual or hand methods and computer methods. The manual or hand design method is rarely used nowadays because computers are used worldwide, and relatively inexpensive software is available. A complete description of this technique is shown by Annels (1991). Regarding the computer methods, the growth of their use in the last two decades has enabled to handle huge amounts of data and to study more pit alternatives than with manual methods. In computer methods, again two groups can be defined: computer-assisted hand methods and automated methods or computer methods stricto sensu. In the first, the calculations are done by the computer under the direct guidance of the technician. In the automated methods, the software program designs the ultimate pit limit based on a given group of economic and physical constraints without assistance of the technician. These methods are called in a general sense optimization methods and they obviously use different mathematical algorithms to perform the ultimate pit.

Regardless of the method used, in designing the open-pit, it is necessary to set up values of the physical and economic parameters. The final pit limit will mean the maximum limit of all material matching these criteria. Thus, the material included in the pit will find two objectives: (a) a block will not be mined unless it can pay all costs for its mining, processing, and marketing and for stripping the waste above the block; and (b) for conservation of resources, any block meeting the first objective will be included in the pit (Armstrong 1990). The result of these aims is to establish the design that will maximize the total profit of the pit in terms of the physical and economic parameters applied. As these parameters change, the pit design can also change.

Pit design relies on preliminary analysis consisting of (1) an ore body model in which the deposit is discretized into a grid of blocks, each of which consists of a volume of material and the corresponding mineral properties; (2) the value of each block, which is determined by comparing market prices for ore with extraction and processing costs; and (3) a geometric model of the deposit» (Newman et al. 2010; Amankwah 2011), being the block model produced in a variety of ways depending on the structure of the ore body. This block model can consider millions of blocks based on the size of the deposit and the blocks.

The block dimensions are dependent on the physical characteristics of the mine such as pit slopes, dip of deposit, and grade variability as well as the equipment used. However, the block sizes should reflect the selective mining unit (SMU) to be used. If the ore blocks have heights equal to the bench height, or to some exact fraction of them, then it is an easy matter to locate bench faces to enclose as much ore as possible (Annels 1991). The raw material grade, especially in metal mines, is susceptible to the dimensions of mining blocks (e.g., bench height) and therefore the size of the equipment. For example, in a mine with erratic spatial ore distribution (e.g., many gold deposits), the dimensions of the mining block size have a dramatic impact on the ultimate pit value and must be determined very precisely. The smallest mining unit or selective mining unit (SMU) is the smallest block inside which ore and waste cannot be separated, and grade estimates are utilized to maximize the pit value. In fact, the determination of mineral resources and/or reserves from a block model needs the selection of a block of SMU size.

The second step is carried out computing, based on tonnage and grade data, an estimated profit of extraction for each block in the model. The ore body model is utilized to define the ultimate pit limits that are the limits of the deposit up to which it is economic to extract. Based on the financial, metallurgical, and geotechnical information, the net profit of each block is calculated. This type of block model is commonly referred as economic block model. In the third step, a geometric model is performed based on slope angle computations. These angles are determined by the structural composition of the rocks and change depending on the location and depth. If it is possible to fix the block values and the slopes, an optimal outline can be determined. It is clear that an increment of the values of the blocks generates an increase in the size of the optimal pit, while an increment in slopes means the optimal pit gets deeper.

Open-Pit Optimization

Optimum pit design is done by utilizing mining software that either uses the floating or moving cone method or the Lerchs and Grossmann algorithm (Lerchs and Grossmann 1965). The Lerchs and Grossmann algorithm guaranties the optimality with respect to defining the pit limits that maximize the undiscounted profit, while floating cone routine is heuristic and can give suboptimum results. For this reason, Lerchs and Grossmann algorithm is the most used method in optimization software programs, whereas moving cone method is commonly integrated in low-cost software (Box 5.4: Lerchs and Grossmann Algorithm).

The moving cone is the simplest method for determining the optimal pit shape, being also the most widely utilized of the heuristic algorithms since it is very easy to program and simple to understand. In this method, the optimum pit is a combination of groups of removal cones of blocks. Main problems of the method include the following: (a) the final pit design relies on the sequence in which reference blocks are chosen; and (b) many reference blocks might need to be chosen, and the associated value of the cone computed, to achieve a reasonable although not even necessarily optimal pit design (Annels 1991).

Aside from the algorithm used by the software package (e.g., moving cone, Lerchs and Grossman, Milawa, Korobov, and others), the aim of an optimization mining software is to generate the most cost-effective and most profitable open-pit design from a block model of an ore body. New algorithms

Box 5.4

Lerchs and Grossmann Algorithm

In the optimization of open-pit mine design, the Lerchs and Grossmann algorithm is the industry standard. Helmut Lerchs' and Ingo Grossman's paper (1965), «Optimum Design of Open-Pit Mines,» outlined an algorithm based on graph theory that could help planners determine the ultimate limits of an open-pit mine in three dimensions. At that time, most computers were incapable of performing the large quantities of iterative calculations required by the method. For this reason, a 2-D algorithm was also described. This method provided a first approximation to an optimal pit as it only considers data on one section at a time, ignoring data on adjacent sections. Though effective on sections, the 2-D algorithm lost its optimized quality where sections were combined. Therefore, Lerchs and Grossmann were the first to put forward a method to solve the open-pit mine optimization problem. The aim was to design the contour of a pit that maximizes the difference between total mine value of the extracted ore and the total cost of extraction of ore and waste materials.

In essence, the algorithm works by flagging certain blocks as «strong,» meaning that they are planned to be mined. Blocks that are not strong are labeled as «weak» meaning that there is no current plan to mine them. A block is considered to be strong if it belongs to a group of linked blocks, known as a branch, with a total positive value. Initially, each block is a separate branch and thus only the blocks with a positive economic value are strong. Lerchs and Grossmann indicated that where a check through all the arcs does not detect any possible strong to weak connection, then those blocks which are labeled as strong constitute the optimal pit. Being the first step of this process to generate the optimal ultimate pit, the second step is to create nested pits within the ultimate pit by changing the capacities of the arcs between the nodes of the graph; this process is termed pit parameterization. In the third step, the nested pits are combined to obtain a pushback design, and later on, a production schedule is added. Pushbacks are generated by combining nested pits so as to maximize the net present value of the pit design (pit limit and pushbacks).

In 1986, the Whittle 3-D open-pit optimization package was launched by Whittle Programming Pty Ltd. This package utilized the Lerchs-Grossman algorithm in a commercial software application for the first time. Therefore, the Lerchs-Grossman algorithm was the first optimization method used to design large open-pits in reasonable time, and it is still used in mining optimization software as the industry standard to find the optimal pit. In 1987, Whittle 4-D was released incorporating time, risk, and optimizing around NPV and incorporating sensitivity analysis for long-term planning.

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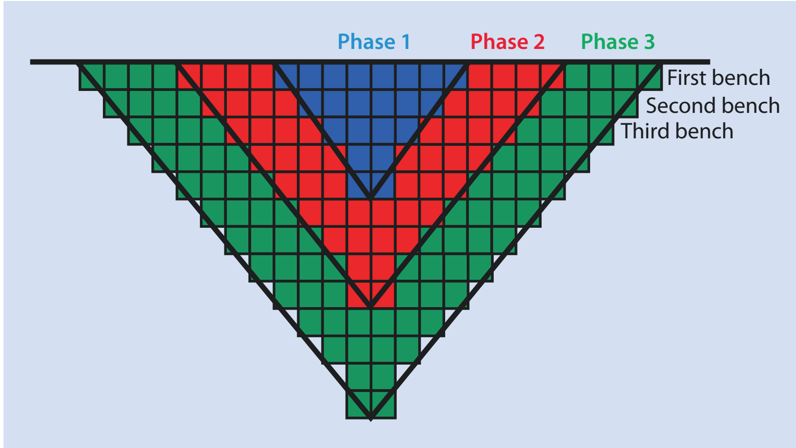


Fig. 5.28 Nested pits

Table 5.2 Example of pushback values for a production schedule

Pit	Blocks in pit	Pushback	Blocks in pushback	Tonnage per pushback (M)	Life of pushback year
1	6756	1	6756	152	1.90
2	20,970	2	14,214	320	4.00
3	34,772	3	13,802	320	4.00
Ultimate pit	53,577	4	18,805	439	5.48
Total	53,577		53,577	1232	15.39

are described recently, many of them related to environmental constraints including ecological costs of open-pit mining such as prevention and restoration costs or cost of carbon emission from energy consumption (e.g., Xu et al. 2014).

Production Scheduling

Production scheduling of the open-pit mines is a difficult and complex optimization problem. It can be outlined as the sequence in which ore and waste of the pit are extracted over the lifetime of the mine and the time gap in which every material is to be extracted. The main goal production scheduling is «to maximize the total discounted profit from the mine subject to a variety of physical and economic constraints; in the process, a set of nested pits is generated, starting with the ultimate pit contour, by varying the economic parameters» (Cacceta 2007). This process assumes an a priori discretization of time into periods and a priori definition of production capacity in each time period. To determine the time of extraction for each block, a subgroup of nested pits from those calculated in the sequencing step is selected. For instance, in Fig. 5.28 the blue area (the smallest pit) is the one that represents the best value that is possible in the early stages of mining as it is the pit that would still be valuable even under the worst economic conditions (i.e., a low commodity price). The green (largest) pit represents the pit with the longest life under the best economic conditions. Each of these pits is called a «pushback or phase» (Chiscoine et al. 2012). By changing the commodity price, for instance, from a low value to a high value, it is possible to originate an increasing size number of pits and a diminishing average value per ton of mineralization included in the open-pit. Since the smallest size open-pit covers the highest-valued ore, the production is scheduled by extracting smallest pit first followed by the production in larger pits. The incremental mining from the smallest pit to larger pit is commonly referred to as pushback mining. Table 5.2 shows an example of pushback values for a production schedule.

Fig. 5.29 The Ekati Diamond Mine (Image courtesy of Dominion Diamond Corporation)

Open-Pit Case Studies

Actual examples of diamond, copper, nickel, and gold open-pit mines are described below.

Ekati Diamond Mine (Northwest Territories, Canada): Courtesy of Dominion Diamond Corporation

The Ekati mine site is situated in the Lac de Gras region of the Northwest Territories, about 250 km northeast of Yellowknife (Canada). The Ekati Diamond Mine (named after the Tlicho word meaning «fat lake») (Fig. 5.29) is the first surface and underground diamond mine in Canada. It officially began production in October 1998, after intensive prospection and development work dating back to 1981. The largest gem quality diamond generated to date at the mine is the 78 carat Ekati Spirit, which was discovered in 2010 and sold at auction in 2011. At the beginning, production was located on six open-pits and two underground operations. The current planning is based on extraction from six kimberlite pipes: Misery Main, Pigeon, Sable, Lynx, Jay open-pits, and Koala underground operations. Currently, Koala, Misery Main, and Pigeon pipes (Fig. 5.30) are being mined.

Bedrock is dominated by Archean granitoids, which intruded by metagraywackes and transected by Proterozoic mafic dykes. There are not conserver younger cover sediments. Bedrock is overlain by Quaternary glacial deposits that are commonly less than 5 m thick. The kimberlite intrusions are of Phanerozoic age. The Ekati kimberlite pipes are part of the Lac de Gras kimberlite field which is located in the central Slave craton. The kimberlites intrude both granitoids and metasediments. They are mainly small pipelike bodies (surface area predominantly <3 ha but can reach as much as 20 ha) that usually extend to projected depths of 400–600 m below the current land surface. Kimberlite distribution is controlled by fault zones, fault intersections, and dyke swarms. Diamond grades are highly variable. Estimated average grades for kimberlites that have been bulk sampled range from less than 0.05 cpt (carats per ton) to more than 4 cpt.

The kimberlite pipes are nearly circular in plain view and are commonly situated within granite, a competent host rock. The ore/waste limit is sharp and is quickly differentiated by the type of rock. The open-pits are mined utilizing classical truck and shovel procedures (the principal truck loading and haulage equipment are diesel hydraulic shovels/excavators with a bucket capacity of 12 m^3 and 90 t capacity off-road haul

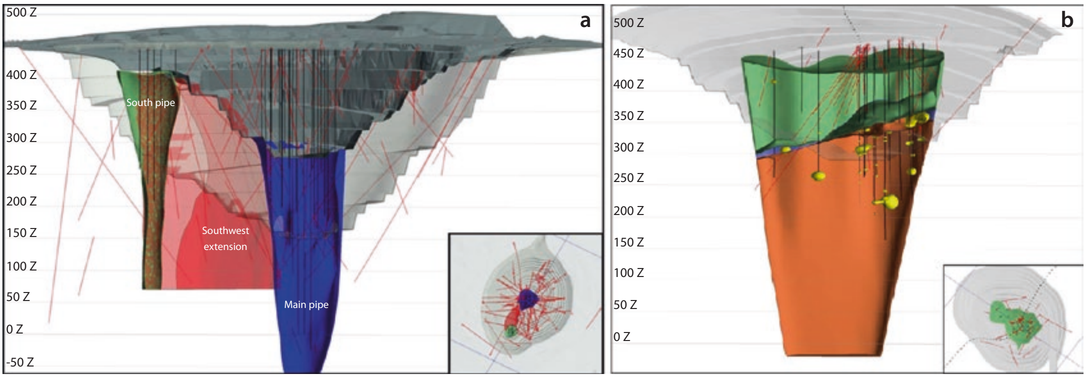


Fig. 5.30 Isometric views of Misery Main **a** and Pigeon pipes **b** (Illustration courtesy of Dominion Diamond Corporation)

trucks) and are carried out in benches usually 10 m high. Pattern of pit slopes changes drastically between ore and waste, being conformed based on detailed geotechnical and hydrogeological investigations and operational specifications for each pipe. A single circular access ramp around the perimeter of the pit is projected progressively as the benches are mined. Waste rock is hauled to an assigned waste rock storage zone and dumped to an engineered design. In general, kimberlite is hauled direct from the pit benches to the process plant. Kimberlite ore is selectively mined on the basis of visual delimitation.

Production blastholes are 270 mm diameter drilled on a 6.5 m by 7.5 m equilateral pattern with 10 m bench heights. Wall control blasting methods including pre-shear firings on the perimeter of the pit excavation improve final highwall stability. Wall control processes on the final pit walls consist of drilling 165 mm presplit blastholes on a 2.0 m spacing on the pit perimeter, followed by a row of 270 mm wall control blastholes on a 3.0 m burden and 4.0 m spacing, then a second row at a 5.0 m by 5.0 m spacing before switching to the standard production pattern. Since the blastholes are commonly wet, a gassed emulsion explosive doped with 30% AN prill is utilized both in waste and kimberlite blasting. The pre-shear holes are loaded with a radially decoupled explosive consisting of a 44 mm diameter continuous water gel product and high-strength detonating cord.

Cobre Las Cruces Copper Mine (Sevilla, Spain): Courtesy of Cobre Las Cruces – First Quantum Minerals Ltd.

The Cobre Las Cruces mineral deposit is located in the eastern end of the Iberian Pyrite Belt, a 300-km-long and 80-km-wide geologic belt that spreads eastward from southern Portugal into southern Spain. Mineralization is formed by syngenetic massive sulfides including polymetallic mineralization, as in most other Iberian Pyrite Belt deposits. Cobre Las Cruces is a blind deposit with no outcrop because of the 100–150 m of sedimentary rocks overlying the deposit. The copper in the ore is primarily found in chalcocite with some minor amounts found in chalcopyrite, tennantite-tetrahedrite complex, and enargite. The ore from the open-pit mine ranges in grade from 5% to 10% copper and the design grade is 6.02% Cu.

The Cobre Las Cruces mine is a medium-sized open-pit mining operation using conventional truck and shovel operations. The mining fleet is basically made up of hydraulic shovels and 90 t haul trucks. The hydraulic shovels have a 7 cubic meter bucket. The support fleet consists of conventional equipment such as graders, tractor dozers, water tankers, etc. From 2015 onward, the mine is expected to produce around 72 kt tons of copper cathode per year. Mine development required a preproduction phase of almost 24 months for prestripping to expose sufficient ore to ensure steady ore production. The overall pit slope angle is about 28° in the upper and lower tertiary marl and sandstone, 45° in the Paleozoic bedrock, and between 32° and 36° in Paleozoic soft rocks (shales). Trucks haul the ore to the blending yard within the north dump complex and then to the primary crusher located near the processing plant. Overburden material (marls and sandstone from the aquifer) is hauled to inert dumping facilities. In the later years of the project, partial backfilling of the pit with marl will occur although CLC is considering delaying the backfill operation in order to maintain access to the primary ore resource.

The mining phases generally comprise wide benches of between 30 and 200 m in width, providing several mining horizons to satisfy the feed requirements for blending. Benches (interval between berms) are mined to a height of 10 m in ore and waste. In general, ore is hauled to a ROM pad located immediately east of the pit, whereas waste is hauled to various dumps around the northern, eastern, and southern extremities of the ultimate pit. The mining sequence broadly follows the sequence of events as follows:

1. Blastholes are sampled and the results are used in conjunction with the resource model to delineate the ore zones.
2. Blast patterns are designed to reduce material throw and ore dilution.
3. Ore and waste are blasted and mined separately in order to minimize ore loss and dilution; the current values used in reserve estimation are mining loss = 2% and mining dilution = 4.3% at 0.1% Cu.
4. The removal of waste in the successive cutbacks utilizes bulk systems of operation.
5. Perimeter blasting is used to ensure pit wall profiles are cut to the correct angle and to minimize wall damage.
6. Diesel/hydraulic excavators load rock into haul trucks.
7. Ore is hauled from the pit to the ROM blending area where finger stockpiles are used to ensure ore blending can be achieved.

Regarding grade control, blastholes are currently sampled for grade control. In-pit mapping and blasthole drill sample analysis are utilized to guide short-term mine planning and design in accordance with the longer-term strategic mine plans. Typically, blast patterns in ore are 4 m × 4.5 m, square or staggered. The holes are drilled vertically and sampled to 5 m depths with some over-drilling to support locally relevant blast designs. Mining software is used to collate the grade control data and to update the geology and grade estimates into a grade control block model. Results are interpreted as in-pit mark outs, according to the short-term mine and blast designs. Mark outs are assigned evaluated grades as per the grade control block model and are assigned to the digging and truck dispatches for the respective run of mine stockpiles and waste dumps. Regular (monthly) pit surveying allows for accurate assignment of tons and grades mined and dispatched to the respective destinations. Accordingly, feed to the plant is known and is verified with a milled measurement for reconciliation of tons and grade. Final metal generated as cathode is reconciled back to the declared tons and grades. Over the life of the operation, final metal processed/sold is 9% below the predicted mineral resource estimates and is 1% higher than the predicted grade control model estimates. Where dilution and mining recovery are taken into account, the variance between actual production and the reserve estimate is reduced to less than 7%. Accordingly, mining reconciliation is good, with limited concerns identified during modeling/planning, mining, and processing.

■ Ambatovy Nickel Mine (Antananarivo, Madagascar): Courtesy of Sherritt International Corporation

Ambatovy is a large-scale nickel and cobalt mining located 80 km east of Antananarivo (the capital of Madagascar) near the town of Moramanga. It develops an open-pit mining operation and an ore preparation plant. From the mine, the slurried laterite ore is sent via pipeline of approximately 220 km in length to a processing plant and refinery situated south of the Port of Toamasina. Project construction began in 2007 and was completed in 2012. The estimated life of the operation is approximately 29 years.

Gneisses and migmatites form part of the high-grade metamorphic rocks that underlie the eastern two-thirds of Madagascar. As Madagascar broke away from the African continent, the breakup was accompanied by volcanism and internal rifting, the latter forming the horst and graben structural features that are pertinent to the Ambatovy mine. A large intrusive, known as the Antampombato complex, cuts the gneissic terrain and dominates the geological setting of the Ambatovy mine. Within this complex, Ambatovy mafic-ultramafic intrusion can be identified. It consists mainly of ultramafic rocks with pyroxenite injections. Since ultramafic rocks are highly unstable in a tropical weathering environment, Ambatovy presents a deep weathering alteration, with a complete lateritic profile capped by a ferruginous duricrust. Thus, the ore deposit is a typical nickel laterite in which enrichment has occurred in the residual soils formed by tropical weathering

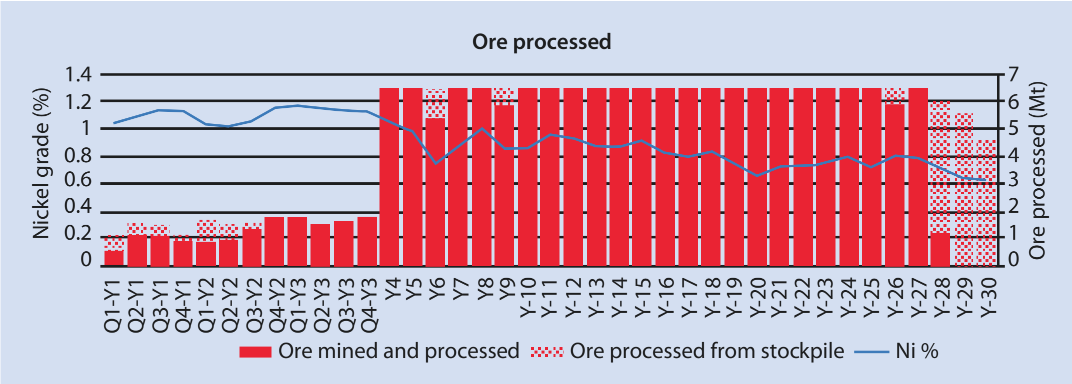


Fig. 5.31 Mill feed production schedule – mined and stockpiled feed and nickel grade by period (Illustration courtesy of Sherritt International Corporation)

of ultramafic bedrock. Prolonged weathering has produced a thick mature laterite profile in which the nickel grades have been enriched from the levels seen in the underlying bedrock.

The Ambatovy mine contains 135.4 Mt. of mineralized material above the cutoff grade with an average nickel grade of 0.834% and average cobalt grade of 0.076%. Waste tonnage in the mine is 45.3 Mt. for a stripping ratio of 3:1. The bench height and width are 6 m and 5.8 m, respectively, being the face angle 45°. The Ambatovy pit dimension is about 4.0 km long and 2.3 km wide. The base of the pit is at an elevation of 978 m, resulting in a pit depth of approximately 75 m. The Ambatovy pit has been split into nine pushback phases. The phases are designed to allow for mining of the higher nickel grade zones first in order to maximize plant throughput during the years of the project, maintain the biological corridors, and provide in-pit waste backfilling opportunities. Thus, the Ambatovy open-pit will produce 190.4 Mt. of mill feed including 2.4 Mt. of ore currently in stockpile and 115.0 Mt. of waste rock over a 30-year mine operating life. Figure 5.31 presents the mill feed summary by direct feed ore, stockpiled ore, and the average nickel head grade by period.

Current mining activities have shown ore and waste to be amenable to free digging; thus, they will not require drilling/blasting, which is reflected in the mining costs. The operating bench height is 6 m in ore and waste with the ability to mine split benches at 3 m height as required, maximizing ore recovery and minimizing dilution and ore loss. Minimal blasting is required in specific locations to bring down ramps into hard saprolite or hard rock. Bulldozers with rippers are utilized in hard zones occurring in the ferricrete materials. Once the ferricrete material is removed, equipment footing on the ferralite and saprolite material will require a layer of aggregate sheeting to maintain equipment efficiency.

Ore control drilling is systematically done to collect assay information on a 20 m × 20 m grid in advance of mining. The main grade control preproduction drilling is carried out with reverse circulation drillhole sample collection. Samples are collected and assayed on a consistent basis. This drilling is completed in advance of production drilling. In locations where the grade distribution is highly variable, a sonic-type drill will bore on a reduced grid of 10 m × 10 m spacing. Ore control models are assembled to assist in the control of the mine operation based on ore grade and quality contacts.

Hauling is done with a fleet of 40 ton articulated trucks. The mine haulage fleet is currently being replaced with larger 100 ton haul trucks. The 40 ton trucks will be kept to perform a number of support tasks around the mine. All excavation is now performed with backhoes in the 4.8 m^3 size class. The excavators are being replaced with larger front-end shovels to match the 100 ton haul trucks, and 17 m^3 hydraulic front shovels are anticipated. The current backhoes will be kept to work with the smaller fleet of smaller trucks. Waste and stockpiled material will be delivered to the appropriate storage locations. Waste and low-grade storage will be accomplished in 6 m lifts. Low-ground-pressure dozers are used at the storage sites to maintain truck efficiency when poor footing conditions occur.

Kişladağ Gold Mine (Usak Province, Turkey): Courtesy of Eldorado Gold Corporation

Kişladağ is a low-grade, bulk-tonnage, open-pit mine that uses heap leaching for gold recovery, being the largest gold mine in Turkey. Kişladağ is a porphyry gold deposit that formed beneath a coeval Miocene volcanic complex in western Anatolia, Turkey. Gold mineralization occurs within zones of quartz-pyrite stockwork and disseminations. Oxidation extends to a depth of 20–80 m, but there is no supergene enrichment.

The ground conditions at Kişladağ mine are highly variable. Zones of geotechnical importance include the weathering profile that divides the oxide and sulfide horizons, the three intrusions, which have different alteration profiles and structural characteristics, and a series of late state brittle deformations called friable zones. These major zones are also affected by a local rock mass fabric, which includes multiple joint sets of varying persistence and orientation. The open-pit slopes have been monitored on a continuous basis since the start of operations. The monitoring program consists of measurements of slope displacement using prisms, changes in groundwater levels using piezometers, regular inspections of the berms and highwalls, and development of a hazard map for mine operations. A slope radar system is also planned, as the mine gets deeper.

The final reserve pit has been designed to economically extract the oxide and sulfide resources that are convertible to ore reserves. It extends about 1250 m from north to south and 1350 m from east to west. The mine was delineated utilizing mining software based on a 10 m bench height with double benching for most pit walls. Thus, pit development will be varying for five geotechnical sectors. Twenty meter high benches will be developed in two 10 m steps. The 20 m face height will be reduced in some locations, where ground type is expected as friable. Slope face angles will change from 65° to 75° depending on the sectors and oxide-sulfide type of ground. Spill berm widths will vary from 6.7 to 9 m and were used to separate bench stacks and satisfy the overall slope angle limitations. Geotechnical berm widths will be in the range of 12.5–28 m. Inter-ramp angles varied from 39° to 56°. The overall slope angle of the final pit design is 41.9° for N, 42.4° for NE, 41.3° for SE, 43° for S, and 46.5° for W geotechnical sectors.

The upper half of the pit (above 750 m elevation) has a double ramp network, and the lower half of the pit is limited to a single ramp. Ramps were designed with a minimum width of 26.3 m for the two-way traffic ramps and a minimum width of 16.0 m for the single-lane ramp used only for the bottom four benches. The pit exposure on surface ranges in elevation from 960 m to 1080 m, and the pit extends down to a bottom elevation of 500 m. The entire pit has a surface footprint of 125 ha. Pit designs have been completed for four mining phases, namely, the initial pit, two intermediate pits, and a final pit (Fig. 5.32). The four phases are based on pit shells while considering practical mining widths for the intermediate pits. Of the total proven and probable reserves (217,470,000 t), a total of 25,694,000 tons is oxide ore at a grade of 0.72 g/t, and 191,776,000 tons is sulfide ore at a grade of 1.01 g/t based on a cutoff grade of 0.35 g/t Au for oxide ore and 0.50 g/t Au for sulfide ore. A 10 m drilling bench height is used with about 1 m subdrill totaling 11 m. Production drilling also provides samples for grade control purposes.

Hydraulic excavators and a front-end loader complimented with off-highway trucks make up the production fleet. The currently selected excavating equipment consists of hydraulic shovels with 21 m^3 bucket capacity loading into a fleet of dump trucks each with a capacity of 136 tons. A front-end loader has been selected as a secondary loading unit. For blasthole drilling, rotary rock drills have proven to be capable of meeting production targets while also providing grade control data. All final walls and long-standing intermediate walls are presplit to a 20 m face height (10 m in the friable zones). The equipment required for constructing and maintaining haul roads, waste dumps, and for in-pit duties includes a wheel dozer, two dozers, two graders, three water trucks, and a rock breaker.

Strip Mining

Strip mining or opencast mining is a surface method that resembles open-pit mining. Opencast mining seems to be a more descriptive generic term for the method (Hartman and Mutmansky 2002). It is used for large, tabular, flat-lying mineralization that is close to the surface. Although a range of commodities such as phosphate, bauxite, tar sands, manganese, and even industrial materials from quarries have been recovered by this method, the most common deposits worked by

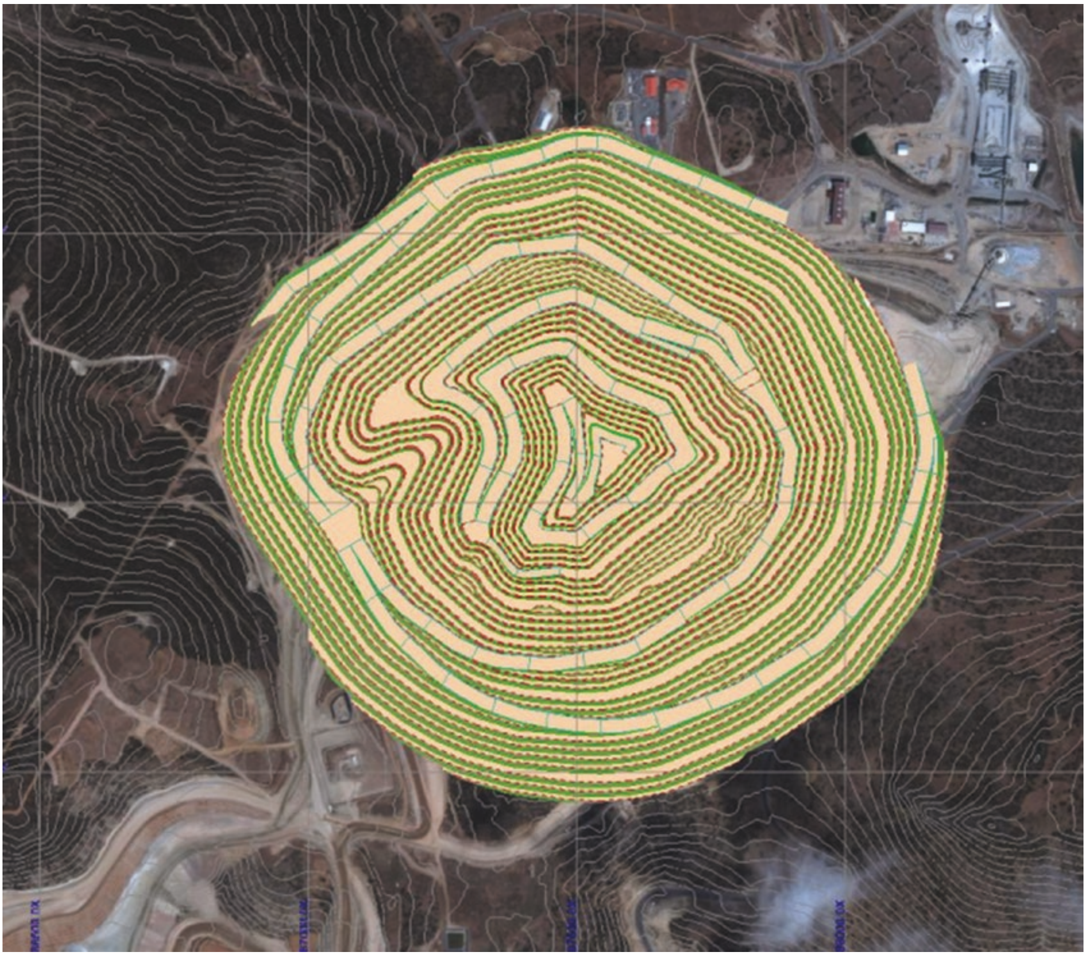


Fig. 5.32 Kişladağ final open-pit design at 2030 (Illustration courtesy of Eldorado Gold Corporation)

strip mining are coal deposits (Fig. 5.33). The main difference between strip mining and open-pit mining lies in the overburden disposal. In strip mining, overburden is dumped directly onto mined-out panels rather than outside the final pit boundary, as typical of open-pit mining. This process is commonly established in one unit operation and carried out by a single machine. Therefore, this method offers an additional advantage of utilizing the same land which is taken up by the deposit for locating the waste rock and hence a minimum land degradation (Tatiya 2013).

Backfilling of strip mining is often economically feasible and desirable as part of the concurrent reclamation requirements. In open-pit mining, this procedure generally cannot take place until the extraction is completed; even then, the very high cost of filling the pit with all of the waste rock extracted at the end of the mine life would seriously jeopardize the economy of the mining project. Therefore, strip mining is characterized by its method of waste material movement, which is placed almost entirely inside the pit. Thus, strip mining includes a progressive and quick process of reclamation; each mined cut is reclaimed arranging the waste rock, overburden, and fertile medium from the next cut to the mined strip and then revegetating the disturbed land.

In this method, an initial cut is made on the subcrop, called the boxcut, and the overburden is placed on a natural surface updip of the subcrop line. The exposed material (e.g., coal) is mined out, and successive cuts, or strips, are taken to progress the mining downdip with the overburden from each strip placed inside the previous mining void. Thus, waste rock recasting goes at the same time with mineralization mining that enables high production rate and almost continuous muck flow

Fig. 5.33 Coal strip mining (USA)

under appropriate conditions. In the case of coal seams, individual strip geometry is typically from 30 to 100 m wide and to the economically recoverable basal coal seam. Final landforms in strip mines can range from voids whose batters have been regraded to voids that have been fully backfilled to the original topographic levels.

Strip mining is unique in that the drill and blast process itself can be employed as an overburden removal process. As the overburden is to be placed into the mined-out void, immediately adjacent, certain pit configurations and operating methodologies lend themselves to cast blasting. Cast blasting is where a powder factor and delay design are selected to purposely cause the fractured rock mass to heave in the direction of the mined-out void with large quantities of overburden, up to 30%, resting in the final position. It therefore requires no further handling by mining equipment. This is a particularly economical method of overburden removal.

Strip mining is a bulk earthmoving operation making use of large-scale mechanized equipment. This enables high-productivity and low-mining charges allowing extraction of even low-grade and deep-seated mineral deposits with higher values of the stripping ratio (Tatiya 2013). Thus, the use of highly productive equipment such as bucket wheel excavators and high-capacity belt conveyors is feasible. Dragline equipment, supplemented by truck and shovel systems, are also observed in strip mines. In this method, stripping ratios can be relatively high, and slope angles can be steep, largely due to the relatively low overall height of these slopes (Carter 2011).

The two main variations of strip mining are area mining and contour mining. Area mining is performed on moderately flat terrain with flat-lying seams; mining cuts are made in straight, parallel panels, advancing across the property. Contour mining (Fig. 5.34) is usually carried out in mountainous terrain, with cuts located on the contours of the topography. The mining proceeds around the hills extracting the seam to a depth fixed by the stripping ratio. This method is commonly practiced in the Appalachian coalfields (USA) (Hartman and Mutmansky 2002).

Quarrying Mining

Quarrying is the extraction of rock (e.g., industrial minerals and rocks) from the ground. As such, the geology of a country or region determines where a quarry is located. Quarries are very similar in design and operation processes to open-pit mines. Commodities mined in quarries include aggregates, dimension or natural building stone, raw materials for Portland cement and lime manufacture, clays for bricks and tiles, and many

Fig. 5.34 Contour mining (USA)

other, especially industrial minerals (e.g., calcite, talc, feldspars, and silica sand).

In crushed stone for aggregates, the excavated rock is crushed, screened, and separated into different size fractions for subsequent sale and use. Since the products are usually of relatively low value and for local markets, they are transport cost sensitive. Hence, wherever possible, quarries of crushed stone are located as close as possible to the market (e.g., a big city). If site investigations for a new quarry of aggregates either sand and gravel or crushed stone are carried out, local factors to study include depth of overburden, size of reserve, water table (dewatering is sometimes necessary), rock type, visual impact (landscaped amenity banks must be constructed and/or large number of trees planted), the presence of roads and railways close to the plant, and distance to the market. Special measures are required to minimize adverse environmental impacts such as noise from drilling, vibrations from blasting, and dust from crushing and screening to the neighboring areas.

Natural building stone quarries are also common examples of this type of surface mining, although with some specific characteristics. Thus, the majority of dimension stone quarries are conducted according to a regular bench design. The rock is commonly cut in the quarry using diamond wire. Other techniques such as explosive splitting or flame jet burner are sometimes used in hard rocks (e.g., granite), although flame jet burner damages the rock to a considerable depth. The marketable dimension blocks obtained by drill-and-shear techniques are then transported to the factory, where the blocks are again cut and sliced in different sizes and shapes. In this type of quarries, bench faces are commonly vertical due to the good geotechnical features of the rocks. These products are frequently high value in comparison with crushed stone for aggregates. For this reason, they can be transported and sold worldwide as building stone. An example are the alabaster panel windows used in the Cathedral of Los Angeles (USA) since the raw material to obtain the panels was previously mined near Zaragoza (Spain) and then transported to the USA.

Auger Mining

Auger mining (Fig. 5.35) is a comparatively low-cost method of coal mining. It starts in the West Virginia coalfields (USA) in the 1940s, being

Fig. 5.35 Auger coal mining (Australia) (Image courtesy of Coal Augering Services)

Fig. 5.36 Auger mining equipment (Image courtesy of Coal Augering Services)

in use today (e.g., the USA and Australia). Auger mining is used on mountainous terrain and needs a surface cut, extraction of overburden, and a fraction of the coalbed to enable the auger access to the bed. The auger method involves boring horizontal or near-horizontal holes in a face of the coal and loading the coal extracted by the auger. It is usually utilized to add value at contour or strip mines where the overburden becomes too great to be economically extracted in a determined pit design. It is also applied where the terrain is too steep for overburden removal because retrieval of coal using underground methods can be unfeasible or unsafe or to extract a proportion of the coal left from underground methods. In case of physical constraints, auger mining is usually the only choice to increment the amount of coal produced. This method uses large-diameter drills mounted on mobile equipment to bore into a coal seam (Fig. 5.36). Holes are horizontally drilled at regular intervals to depth of as much as 300 m and with diameters of up to 2 m. Where the hole is

Fig. 5.37 Namdeb walking jack-up platform that includes a purpose-designed dredge pump to extract sand and gravel and pick out diamonds in Namibian coastline (Image courtesy of De Beers)

mined to its defined depth, the auger equipment is translated laterally 1 or 2 m and another hole is drilled.

Aqueous Extraction

Aqueous extraction encompasses several methods that are used in special circumstances. They have in common the use of water or a liquid solvent as the basic component in the mining process, either by hydraulic disintegration or physicochemical dissolution. Examples of these methods are dredging (Fig. 5.37), hydraulic mining, in situ leaching, and evaporite processing. Dredging is the most common method of large-scale mining of placers, which involves the extraction of the unconsolidated materials from a body of water without the use of explosives or any other significant means of rock-breaking force (Bullock et al. 2011). This method is particularly suitable if adequate water supply is available and the mining operation can comply with the applicable environmental regulations. Modern dredges can produce between 600 and 1500 tons per hour (Haldar 2013).

The dredging process is usually performed from a floating vessel called a dredge, which can include many processing facilities. The concentration of minerals is performed using jigs, cyclones, spirals, and shaking tables (see ▸ Chap. 6). The body of water used for dredging can be natural or human made. Dredges are often classified by method of excavation and material transport. Mechanical dredges are those that mechanically excavate and transport the mineral. Hydraulic dredges, also called suction dredges, are designed to transport the mineral in slurry form, using water as the transport medium. The valuable minerals or metals obtained with this method are meaningful: gold, diamonds, cassiterite, heavy mineral sands (Box 5.5: Heavy Mineral Sands Dredging), and precious stone.

Hydraulic mining (Fig. 5.39) or hydraulicking is a method of mining placer deposits that was utilized in the past but actually is not applied due to environmental issues. It is a low-cost method to extract large amounts of unconsolidated material. In this method, a high-pressure stream of water is directed against a bank to undercut and cave it. The loosened particles are then washed and transported by gravity to a concentrating device.

In situ mining is the extraction of the meaningful elements of a mineral deposit without physical removal of the solid material (Bates and Jackson 1987). It is commonly carried out by dissolving the mineral in an adequate liquid that is later removed

Box 5.5

Heavy Mineral Sands Dredging at Cooljarloo Mine (Perth, Australia): Courtesy of Tronox Ltd.

The Cooljarlo heavy mineral deposit lies within the Perth Basin in Western Australia. The detrital heavy minerals of the Perth Basin include ilmenite, rutile, and zircon, which were derived from igneous and metamorphic rocks in the adjacent Archaean shield to the east in the interior of Western Australia, concentrated in nearshore sediments through multiple phases of weathering, erosion, and deposition. Most of the high-grade heavy mineral deposits at Cooljarloo occur as shoreline accumulations comprising detrital ilmenite, rutile, leucoxene, and zircon with subordinate monazite and a gangue of aluminosilicates, kyanite, staurolite, andalusite, and tourmaline.

The dredging operation at the south mine excavates the deeper deposits located below the water table. A contract overburden removal fleet handles up to 4.5 million bank cubic meters (BCM) of overburden per annum; in 2015 4.2 million BCM of overburden was removed. The overburden is generally between 2 and 15 m thick. Equipment used to remove this overburden includes one 250 ton excavator with a fleet of five 100 ton dump trucks. The excavator can move up to 800 BCM/hour.

Two dredges (◘ Fig. 5.38) operate in a pond up to 25 m deep and mine ore between 22 and 30 m thick. They together mine 23 million tons of ore per year, which is delivered to the shared wet processing plant. The floating dredges pump slurried ore to a floating concentrator that recovers heavy minerals from the sand and clay using a series of gravity spirals. The pond is usually 1 Km long and 400 m wide. The pond water is natural groundwater and is fresh. These facilities are controlled by six operators via computers and GPS satellite navigation.

Heavy mineral concentrate is pumped to a central stockpile where it is stacked ready for rehandling into triple-trailer road trains for transport to Tronox Chandala Processing Plant for separation and processing (see ► Chap. 6). Tailings from the plant comprise washed sand and clay at 2900 tons per hour. The tails is directed either via a floatline and floating tails stacker back to the dredge pond to form stable beaches and to enable the return of stripped overburden or sent via external tails pipelines up to 6 km in length to backfill previously mined out-pits. Thus, as the ore body is mined, overburden and sands with little mineral content are returned to fill the void, clay residue is pumped to solar drying cells, and the surface is contoured to resemble the original landscape, prior to re-spreading topsoil and seeding for rehabilitation. No chemicals are used in the process.

◘ **Fig. 5.38** Dredging operation (Image courtesy of Tronox)

Fig. 5.39 Hydraulic mining at Sierra Leone (Image courtesy of Dove)

for recuperation of the needed constituent. In situ mining includes in situ leaching, solution mining to extract water-soluble salts, brine extraction, sulfur extraction using the Frasch process, and others. In situ processes could potentially deliver the highest goal: a zero environmental footprint.

The application of commercial scale in situ leaching to sedimentary uranium deposits has been around since the 1960s (Albanese and McGagh 2011), being copper, gold, and silver deposits other common examples of minerals mined by this method. Surface leaching commonly uses heap leaching of mineral values. The key to successful leaching of uranium is the identification of suitable, below water table sedimentary deposits in which uranium is confined in permeable rocks by impermeable layers. Thus, the process leaves the ore in the ground and recovers uranium by pumping a leachate solution into boreholes drilled into the deposit; the pregnant solution from the dissolved minerals is then pumped to the surface.

Regarding solution mining, it is likely to be more economical and is inherently safer than conventional underground mining. It will increase in the future as more effective reagents are developed and application methods are improved. Solution mining is used to the exploitation of easily dissolved materials, for instance, sodium- and potassium-bearing evaporates or sulfur, and has also been applied to the extraction of uranium ores hosted in porous sandstone. In a wider sense, coal gasification by underground combustion can be included in this type of surface mining.

A good example of this technique is the extraction of natural sodium sulfate in glauberite mines of Spain (Box 5.6: Glauberite Solution Mining). The mining method employed begins with removal of overburden in an approximate 100 × 100 m pool that is then drilled and blasted over the whole area. A system of wells is then developed and water is injected to the glauberite body and recirculated, being the mineral dissolved. The rich sodium

Box 5.6

Glauberite Solution Mining (Burgos, Spain): Courtesy of SAMCA

The sodium sulfate (glauberite) deposit is located in the Rio Tirón-Belorado subbasin of Spain's northern Ebro Basin, near Burgos. The glauberite ($Na_2SO_4 \cdot CaSO_4$) ore is very pure, and the beds are flat (a maximum slope of 1–2°) and free from faults. The deposit consists of six major zones that contain some interbeds of shale and gypsum, and there are thicker layers of barren rock between the glauberite zones. As an example, in one area the six glauberite zones totaled 39.6 m in thickness or an average of 6.6 m/zone. Each glauberite zone averaged one interbed of barren rock, making 5.4 m of pure glauberite and 1.2 m of interbedded barren rock. The exploitable reserves based upon the 2000 economy were 162 million t of 35% Na_2SO_4 glauberite or 57 million t of Na_2SO_4.

The mining plan has been developed to match the deposit's individual stratigraphy, consisting of first removing and storing the topsoil and sending the 5–15 m of overburden from a proposed leaching pit to the tailings pile. Then, the ore from the first two glauberite zones is selectively mined and stockpiled, with the barren rock between the glauberite zones sent directly to the tailings area. Finally, the ore in the third glauberite zone is blasted in place to the desired rock size and interrock porosity. Blasting for all the rock is done with ANFO and some dynamite by first detonating a row of holes around the edges of the pit to form a fairly smooth wall and then blasting the remainder of the rock. Once the bottom ore zone has been fragmented, the previously mined ore is placed on top of it. The pit size (commonly called pool) is nominally 200 m × 150 m (◘ Fig. 5.40), the average ore thickness 20–21 m, the bed's porosity 25%, and the rock size less than 400 mm (Garret 2001).

Brine (or water) injection and withdrawal wells then constructed 5–30 m apart on opposite sides of the 200 m dimension of the pit, with gravel packing somewhat similar to that of water wells. The injection wells discharged weak brine (or water) 3–4 m from the surface, and the withdrawal wells removed a nearly saturated brine from the bottom of the pit. The piping manifolds had valves on each well so that the brine concentration could be controlled by the flow rate. After being completed, a new pit would be filled with water and the leaching process would commence.

When the leaching of the upper three ore zones was completed, the spent ore would be removed and stockpiled to be later returned to a pit. The remaining three ore zones would next be mined and prepared in the same manner as the upper three and then leached. When they were depleted, the spent ore from the upper zone would be returned, the pit filled with the overburden from a new pit, and topsoil paced over that to return the mined area to its original or an improved condition. Strong brine from the pits was sent to the two parallel processing lines in the plant to form Glauber salt (Garret 2001).

◘ **Fig. 5.40** General view of open-pit mine (Image courtesy of SAMCA) sulfate brine obtained is pumped and sent to an evaporation plant where the brine is converted later into high-quality anhydrous sodium sulfate salt.

In evaporite/evaporation operations, the valuable minerals are produced from a saline solution by evaporation in a closed basin. Halite, potash, and trona are typical examples of this category. The minerals can be recovered by conventional mining operations or by solution mining. In the latter case, recovery is often accomplished by evaporation of the water from brines in solar ponds (◘ Fig. 2.50). Thus, saline solutions are pumped into large, shallow ponds to allow the water to evaporate, being essential a warm and dry climate. Evaporation conditions, volume of solution to be processed, and the expected low rainfall in the area are commonly the major parameters of concern in this type of mineral extraction (Hartman and Muntmansky 2002).

5.4 Surface Mining to Underground Mining

Mineral deposits can be so close to the surface that their extraction by surface methods can easily be carried out. In contrast, some mineral deposits occur so far from surface that only underground mining is allowed. Besides these contrasted situations, there are some deposits that start at surface or near surface and continue to great depth. In such vertically extensive ore bodies, a combination of both surface and underground methods could result in a higher net profit than only one (Bakhtavar 2013). Within this transition zone, it is necessary to consider issues such as the production rate or the economic and risk features because these factors can decide the open-pit to underground mining point that is the best for the project. However, as the costs of underground mining can be many times that of surface mining, only moderate- to high-grade deposits can be mined by underground methods below an open-pit. Accurate estimation of the depth in mines where both methods are utilized is of significant interest (Bakhtavar et al. 2009). The point at which economic considerations define the change from open-pit to underground method is referred to as «transition depth.» To take the decision of where to end the open-pit method and begin the underground method is referred to as the «transition problem,» and it has originated some attention in the literature since the 1980s (◘ Box 5.7: Venetia Transition Surface to Underground Mining).

Box 5.7

Venetia Transition Surface to Underground Mining (South Africa): Courtesy of De Beers

Venetia is South Africa's biggest diamond producer, contributing 40% of South Africa's production and about 10% of De Beers production of 31 million carats last year. By 2021, the diamond-bearing ore at the current Venetia Mine is expected to be depleted. In order to extend the life of the mine, in 2007 De Beers and Anglo American took the decision to construct a new underground mine beneath the open-pit. The project team is currently developing the decline from the surface to a depth of 900 m and sinking two vertical shafts to a depth in excess of 1000 m and is on track for production to commence in 2021. Thus, extending production at the site to 2043, with the potential to deliver an estimated 96 million carats from 130 Mt. of mined kimberlite material over its 22-year life span, the project is the biggest capital investment in South Africa in the company's history.

The Venetia project will build an underground diamond mine beneath the existing open-pit mine (which is among the top eight diamond mines in the world) replacing about 3.2 million carats a year of production by late 2021, as the surface-stripping mining ratio becomes too expensive. The scope of works comprises the building of an entire underground mine. This includes the sinking, equipping, and commissioning of a decline (◘ Fig. 5.41) and two vertical shafts and horizontal tunnel development to provide the establishment of, and access to, loading levels. The work includes associated ventilation, ground, and water-handling infrastructure. Both 7 m diameter vertical shaft bottoms will stretch 1080 m although the ore body will be exploited to about 900 m. Although the ore bodies extend beyond 1000 m, this was determined the optimal depth, particularly with regard to development completion timeframes.

Two main ore bodies will be mined through underground mining. The first ore body will be mined by means of a sub-level caving mining method, producing 4 Mtpa (average of 3.5 million carats per annum). The second ore body will be mined by means of a modified sublevel caving mining method producing 1.9 Mtpa (average of 0.9 million carats per annum). This equates to 5.9 Mtpa, which matches the throughput capability of the existing main treatment plant.

Fig. 5.41 Venetia decline to underground mine (Image courtesy of De Beers)

This combination of initial surface mining and further underground mining is called sequential mining. It is selected on the basis of the ore deposit geometry (dimensions, shape, and depth), rock characteristics and conditions, productivity, capacities of machineries, capital requirements, operating costs, investments, amortization, depreciation, ore recovery, safety, and environmental aspects, among other aspects. It is important to keep in mind that extension of an open-pit with a new pushback often involves removal of millions of tons of material generating huge capital investment. Thus, decisions to expand or deepen an open-pit, instead a transition to underground mining, required extreme care. Detailed planning and modeling before reaching the transitional depth of the mine is essential as many problems can arise influencing the production flow (Fig. 5.42). It is important to bear in mind that only moderate- to high-grade mineral deposits can be mined using a combination of surface and underground mining because the costs of underground methods are commonly many times that of surface methods.

After feasibility of underground mining has been proved, timing of transition to underground mining must be decided. There are two major considerations for this decision. First, it is important to maintain continuity of the operation because underground mining should supplement and eventually take over production from the open-pit without major permanent changes in tonnages of ore shipped to the mill. Differences between the open-pit and underground ore grade and composition can complicate this issue. Second, while a smooth transition requires a production overlap, neither of the two operations should compromise the safety of the other (Wetherelt and van der Wielen 2011).

The main issues to be evaluated in determining the optimal transition depth are the availability of

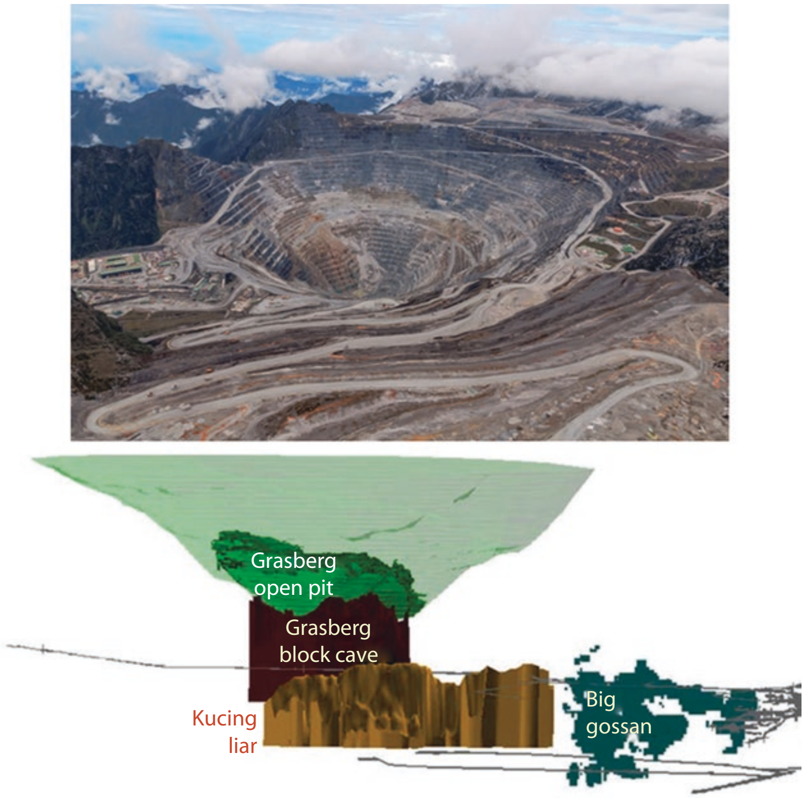
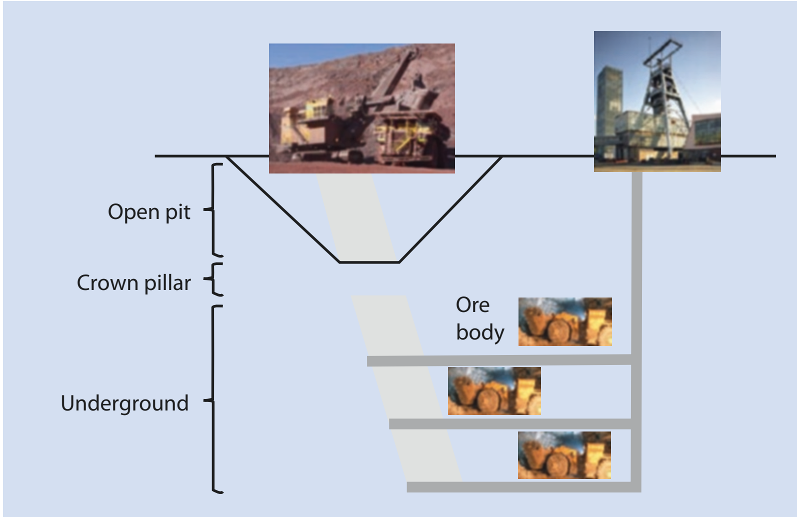


Fig. 5.42 Grasberg (Indonesia) copper and gold open-pit mine at 2014 and proposed transition to underground mining (Image and illustration courtesy of Freeport-McMoRan)

feed, feed grade, and resource utilization impact. Consequently, there is a range of parameters that can be checked to estimate the sensitivity of the optimal transition depth. It is important to specify the ultimate depth of an open-pit mine as early as the planning stage considering economic efficiency of underground mining of the remaining mineral reserves later on (Ordin and Vasilev 2014).

The optimization of the transition depth is also a complex topic. It is defined as the process of determining what part of the ore body (e.g., what blocks in a block model) should be mined by open-pit, what parts should do by underground methods, and when they should be extracted so that operation can maximize the long-term NPV of the project. The problem of optimizing a simultaneously producing open-pit and underground mine plan is really complex, and for a current technology, an iterative process must be embraced in attempts to establish the optimal solution. To consider the transition depth as an essential issue, a number of algorithms were developed in the last two decades (e.g., Nilsson 1982; Bakhtavar et al. 2009; Ordin and Vasilev 2014; among many others). The solution uses the Lerchs-Grossman algorithm, floating cone technique, dynamic programming, neural network, etc., and based on these methods, different software packages are widely used (e.g., Datamine, Vulcan, MineSight, and Gemcom).

Thus, the open-pit to underground transition problem is one of the hot topics in the mining industry that has not been mathematically solved since there is not a mathematical algorithm that can successfully optimize the transition depth by considering the life of the mine schedule of both open-pit and underground all together. Due to the complexity of the problem and its size, often the transition depth is defined by considering the open-pit and the underground separately. Defining the transition depth by comparing the costs of these two mining methods, the economics of the mining project cannot be optimized in terms of the net present value of the project. The underground development work and the value of the underground mine are not properly considered; therefore, the costs and the value of the overall project cannot be correctly estimated. Where correctly defined, the transition depth can significantly enhance the discounted net present value of the mining project (Traore 2014). The final design



■ Fig. 5.43 Illustration of a crown pillar between an open-pit and underground mine

of the combination surface mining-underground mining frequently includes a crown pillar left in place while underground mining is developed (■ Fig. 5.43). The height of the crown pillar is commonly established equal to the maximum width of stopes to be extracted promptly beneath.

5.5 Underground Mining

Underground mining consists of the extraction of material in excavations below the Earth's surface (■ Fig. 5.44). This type of mining employs its own and distinctive nomenclature. Thus, ■ Fig. 5.45 shows the main terms commonly used to describe underground working and other aspects of underground mining. Underground mining exists where a surface mine becomes cost prohibitive to operate by different reasons: (a) the ratio of extracted waste to ore becomes too high; (b) waste storage space is insufficient; (c) pit walls fail; (d) environmental considerations outweigh extraction benefits; and (e) environmental or social factors limit the viability of surface mining. In such cases, underground mining can be the only choice for a given deposit.

However, it is important to note that the economic feasibility of an underground operation depends on more or less the same economic studies as an open-pit mine. If the appeal of surface mining lies in its mass production and minimal-cost capabilities, the attraction of underground mines derives from variety of ore deposits that can be mined and the versatility of its methods to meet conditions that cannot be approached by surface mining. Moreover, underground mining is a method with less environmental impact to gain the access to a mineral deposit. In contrast, it is usually more expensive and involves greater safety risks than surface mining. In general, an underground mine is more complex and generally more expensive than a surface mine because the development openings of an underground mine can be considerably more costly than surface mining on a tonnage basis.

The social, economic, political, and environmental factors of underground mining are often quite different from those of surface mining. A more skilled labor force can be required, financing can be more difficult because of increased risk, and subsidence can become the most important environmental concern (Hartman and Muntmansky 2002). In underground mining, overburden extraction to gain access to mineralization is kept to a minimum, being this access obtained by tunnels or shafts. Thus, there is only a small amount of waste rock generated (development waste), and consequently limited excavation and relatively small openings are necessary for most underground mines. The waste can even be useful since it can be used as backfill in the mine.

Underground mines are generally utilized to exploit high-grade, deep mineralization, usually with mining production rates lesser than 20,000 tons per day. For instance, a 10,000 ton mining production rate is a typical production of highly

Fig. 5.44 Aguas Teñidas underground mine (Spain) (Image courtesy of Matsa, a Mubadala and Trafigura Company)

mechanized and large-capacity underground mines. A particular case should be block caving underground method, since it can achieve mining production rates much greater than 20,000 tons per day. In addition, the use of smaller equipment in underground mining means production rates that are obviously much lower than at a surface mine. In terms of ore tonnage, underground mining is relegated to a secondary role for many commodities. However, it is possible to assume that underground mining will continue to play an important role in supplying mineral resources in the future, with many large underground mines in operation around the world.

Underground mining methods are always selected below 1000 m depth, because it becomes difficult in a surficial exploitation to maintain the stability of a 1000 m high rock slope. Large tabular mineral deposits with long vertical or horizontal dimensions or mineralization lying more than 300 m below the Earth's surface are commonly mined utilizing underground methods as well. In this sense, Mponeng and TauTona (Fig. 5.46) gold mines, located in South Africa, are currently the two deepest mines in the world, respectively (Mponeng exploits at depths of between 2400 and 3900 m and TauTona sinks to depths of between 2900 and 3480 m).

5.5.1 Geotechnical Considerations in Underground Mining

Obviously, geotechnical features of the ore and waste rocks are essential to develop a safe underground mine. Thus, the main goals of geotechnical consideration in underground mine design, independently of the mining method applied, are (a) to ensure the overall stability of the complete mine structure, defined by the main ore body, mined voids, ore remnants (pillars), and adjacent country rock, (b) to protect the major service openings and infrastructure throughout their design life, (c) to provide safe access and working places in and around the centers of ore production, and (d) to preserve the mineable condition of unmined ore reserves (Brady and Brown 2006).

The characteristics of the ore body itself constitute the basis to the geotechnical study, including the thickness and orientation of the mineralization, the ore and rock strength, the distribution of mineralization within the ore body, and the depth

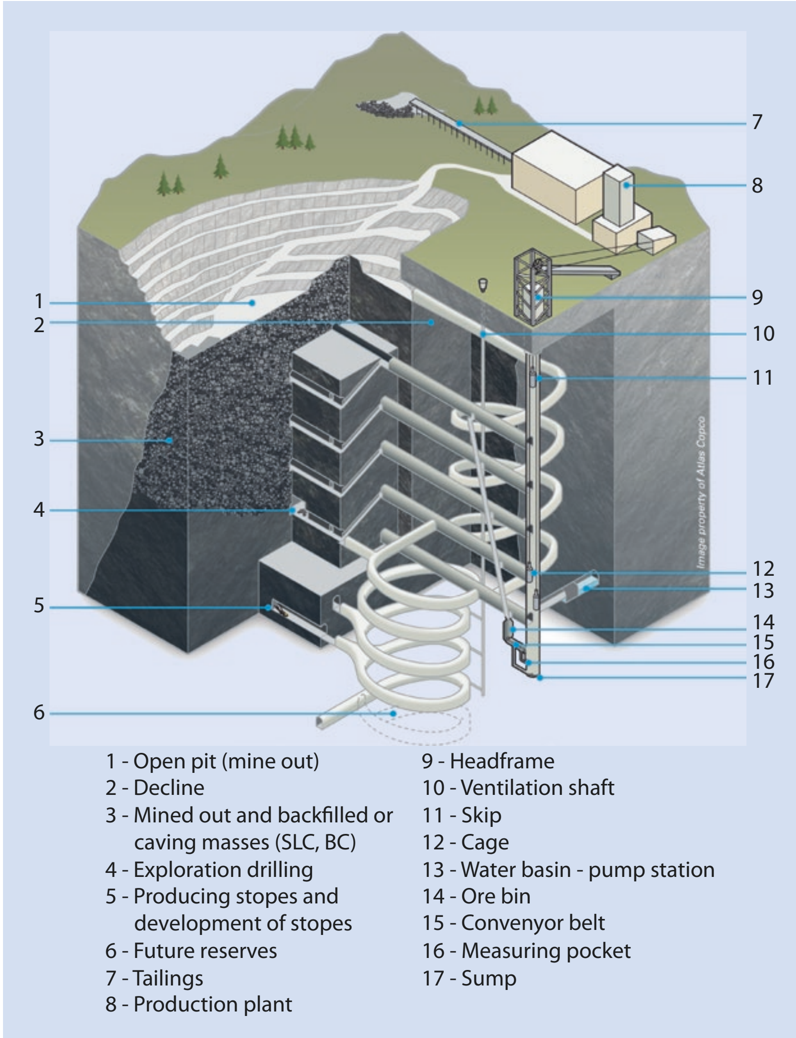


Fig. 5.45 Main terms used in underground mining (Illustration courtesy of Atlas Copco)

Fig. 5.46 TauTona gold mine (South Africa) (Image courtesy of AngloGold Ashanti)

Fig. 5.47 Checking fall of ground (FOG) lights underground at central shaft in Bathopele platinum mine (South Africa) (Image courtesy of Anglo American plc)

of mineralization and surface conditions. Thus, geotechnical data are needed to decide: (1) most economical method of excavating ore and waste rock, (2) pillar sizes and extractions ratios, (3) features to control the subsidence, and (4) where to locate the accesses to the mine. As early as possible in the mine feasibility assessment process, it is essential to understand and fully consider the interrelationships between the local geotechnical environment and the mining process (Fritz and Coldwell 2011). There are important benefits linked to an early prioritization of geotechnical evaluation and impact on underground mine planning. Regarding the hydrogeological conditions, groundwater commonly concerns upper areas of a shaft and must be controlled by grouting to prevent water from entering the shaft.

With deeper level mining and higher overburden pressure, it is very important to give geomechanical validation to engineering decisions to be in accord with the ground conditions. Collapse of mine structures, rock bursts, and higher cost of ground control and mine support directly influence mining productivity. Instability of such structures results, as a rule, in severe accidents and long-term suspension of production up to mine closure, which causes social tension and high economic loss. For this reason, geomechanical monitoring in underground mineral mining to evaluate the stress state and properties of rocks is of paramount importance. Therefore, a monitoring program should be implemented in order to get better understanding of the rock mass deformation mechanisms.

This geomechanical monitoring structure can be outlined utilizing a package of instrumental (Fig. 5.47), visual, and numerical methods for the evaluation of mechanical condition and its alteration in rocks and in structural components of mines. The information support of monitoring systems is based on instrumental and theoretical methods allowing (a) acquisition of reliable source data on natural stress state and mechanical properties of rock masses, (b) determination of mechanisms of change in the stress-strain state of structural elements in the course of deformation under natural or induced forces, and (c) experimental-analytical justification and estimation of limit state criteria in rocks and other materials (concrete, backfill) (Baryshnikov et al. 2014).

The uncertain geotechnical environment in which an underground mine operates is among the prime reasons for geotechnical accidents. Accidents in the form of roof collapse, fallouts, uncontrolled caving, etc., can lead to loss of lives and machinery along with substantial ore loss and loss in productivity. For instance, one of the worst underground mine accident in the world was the so-called «Mufulira disaster,» recorded in Zambia in 1970 when 89 miners died due to flooding. The accident took place in the morning of 25 September 1970, when half of the mine was flooded because mud and water from the slime dam seeped through cracks in an old slope, causing a section of the overhanging wall to give way. Thus, the mud and water rushed into the eastern section of the mine and flooded all shafts below 500 m.

Geotechnical risk assessment at early stages such as mine design can even help to make changes in the design, for example, the use of support methods in risky areas of the mine. The risk assessment process can be defined into four sections: hazard identification tool, risk assessment approaches, risk assessment parameters, and risk representation tool (Mishra and Rinne 2014). Once a geotechnical risk assessment is completed, the result should be analyzed to test if the risk must be mitigated or completely avoided, for example, switching to a different method or abandoning the area. Geotechnical risk assessment process in a mine should be subjected to continual improvement through feedbacks from the mine and via lessons learned during every assessment.

To summarize, the methodology for the implementation of a rock mechanics program can include the following steps:

1. Site characterization: definition of hydromechanical properties of the host rock mass for mining
2. Mine model formulation: conceptualization of site characterization data
3. Design analysis: selection and application of mathematical and computational schemes for study of various mining layouts and strategies
4. Rock performance monitoring: measurement of the operational response to mining of the host rock mass
5. Retrospective analysis: quantification of in situ rock mass properties and identification of dominant modes of response of mine structure (Brady and Brown 2006)

5.5.2 Underground Infrastructure

An underground mine has different components that ensure the extraction of ore and the safety and movement of people and equipment. Therefore, each mining method requires different underground infrastructure such as access drifts to sublevels, drifts for longhole drilling, loading drawpoints, and ore passes. Together, they form an intricate network of openings, drifts, ramps, shafts, and raises. The mine requires three groups of physical plant installations: the surface plant, the shaft plant, and the underground plant. The first consists of a variety of facilities to provide the mine with necessary services such as access roads and parking, transportation facilities, power and water supply, service and maintenance buildings, mineral processing plant, bulk storage, and waste disposal facilities for air, water, and solids. The shaft plant includes the facilities installed for material handling of ore and associated waste and the means of transport of miners and material. It generally incorporates systems for ventilation, drainage, power supply, and communications. Regarding the underground plant, it covers various installations to make the system work efficiently and safely, including storage bins, loading pockets, power distribution equipment, underground maintenance facilities, and numerous other installations that provide auxiliary services to the underground operations (◘ Fig. 5.48).

Mine ventilation is one of the most important facilities of underground mining. Air quality in mine workings is an area of particular concern to the underground development. It must be maintained at an acceptable health standard. A continual and adequate supply of fresh air must be made available to working areas. Underground mines use networks of fans, gates, and surface openings to move fresh air into the mine and remove exhaust air. High-pressure fans on surface extract exhaust air through the upcast shafts and ventilation doors control the underground

Fig. 5.48 Station for underground maintenance equipment (Spain) (Image courtesy of Iberpotash)

airflow, passing fresh air through active work areas. As most of the infrastructure is located on the footwall side of the ore body, the fresh air is normally channeled via the footwall toward the hanging wall, from where the exhaust air is routed to the surface (Nord 2007). It is particularly important to clear the air after an underground blast, because harmful gases such as carbon monoxide or oxides of nitrogen can build up. A good ventilation system will rapidly clear the air around a blast as blasting reduces the concentration of oxygen in the air. Figure 5.49 shows an aerial view of ventilation equipment at Aguas Teñidas Mine (Spain).

Underground development openings, which are designed so that the ore bodies are easily accessible and transportable after excavation, usually can be ranked in three categories by order of importance in the overall layout of the mine: (1) primary or main openings (e.g., shaft or slope), (2) secondary or level or zone openings (e.g., drift or entry), and (3) tertiary or lateral or panel openings (e.g., ramp or crosscut). The construction of underground openings is specialized and expensive, and consequently, this phase of mine development has become increasingly mechanized and efficient in order to reduce costs. A number of initial decisions related to the primary development openings of a mine must be made early in the mine planning stage and include the type, number, shape, and size of the main openings. Factors to influence this decision include the depth, shape, and size of the deposit, the surface topography, the geological conditions of the ore body and surrounding rock, the mining method, and the production rate, among others. Sometimes underground development openings double for exploration purposes and vice versa. Those openings driven in advance of mining can provide valuable exploration information and afford suitable sites for additional exploration drilling and sampling. Likewise, openings driven for exploration purposes can be utilized to develop the deposit; some shafts and drifts would almost certainly serve subsequently to open up the deposit.

Underground Access

The access method to underground works is an important aspect of underground mine development and operation because it is required for people, equipment, and ventilation as well as for transporting ore to the surface. Underground mines usually have several access points with different objectives such as a ramp for equipment and personnel and a shaft for transporting ore out of the mine and form ventilation. There are generally three methods of accessing an underground mine: shaft, adit, and decline or ramp. The shaft remains the mine's main artery, and downward development is by ramps to allow access for the

Fig. 5.49 Aerial view of ventilation equipment at Aguas Teñidas Mine (Spain) (Image courtesy of Matsa, a Mubadala and Trafigura Company)

machines. A decline ramp from surface can facilitate machine movements and transport of people and materials. It can also be used for ore transportation by truck or conveyor, eliminating the need for hoisting shafts.

Shafts

A shaft is a vertical excavation in which elevators are used to transport people and ore in and out of the mine. It is used where the deposit is located deep within the ground. Most shafts are divided into a number of compartments each with a different use. For example, one compartment for moving people, a second for skipping ore to the surface, and other compartments for ventilation and electrical infrastructure. The main factor to establish the shaft size is the estimation of reserves in the sector to be mined by the shaft. Thus, the ore body size will define the rate of mining, and this will determine the tonnage (ore and waste) to be hoisted, the number of persons, and the material to be moved in a given shift. Figure 5.50 show De Beers' Venetia Mine in South Africa with two headgears. One is the production shaft, used to lift kimberlite (containing diamonds) and waste rock. The second is the production services shaft, used to transport employees and equipment in and out of the underground mine.

Because shafts are essential in the general planning of mine development, their localization is commonly predetermined, being this position changed where adverse geotechnical conditions are identified. Ground conditions and water-bearing structures also govern the ultimate localization of shafts. The decision to locate the shaft is critical if the terrain is moderately flat because the process to develop a shaft is very expensive, and only a vertical shaft, well located with respect to the ore deposit, will be helpful later in the production work. Thus, the correct configuration of the shafts will provide optimum operational benefit. The shaft can be rectangular, circular, or elliptical in profile, although almost all hard-rock underground mines commonly have circular section shafts because this shape generates a correct geometry for airflow and suitable rock support characteristics.

Raises

Raises are steeply inclined openings linking the mine sublevels at several vertical elevations. They are normally placed near the stopes employing specialized cyclic or continuous operations. Specific applications of bored raises are transfer of material, ventilation, personnel access, and ore production. Inclination varies from 55°, which is

■ Fig. 5.50 Venetia mine shafts (South Africa) (Image courtesy of De Beers and Anglo American plc)

the lowest angle for gravity translation of blasted rock, to vertical, with cross sections from 0.5 to 30 m^2. Since manual excavation of raises is a very dangerous job, the raise boring machine is currently utilized for boring ventilation raises, ore passes, and rock fill passes. It provides safer and more efficient mechanized excavation of circular raises up to 6 m diameter because this method eliminates the need of explosives.

Raise boring is the procedure of mechanically boring a vertical or inclined shaft between two or more levels. In conventional raise boring, a downward pilot hole is drilled to the target level by the raise boring machine, where the bit is removed and replaced by a reaming head (■ Fig. 5.51). The machine then reams back the hole to final diameter, rotating and pulling the reaming head upward. The cuttings fall to the lower level and are removed by any convenient method. The capital cost of a raise boring machine is high, but the return on investment is very worthwhile. Advantages of raise boring are that miners are not required to enter the excavation while it is underway, no explosives are used, a smooth profile is obtained, and manpower requirements are reduced. Above all, an operation that previously was classified as very dangerous can now be routinely undertaken as a safe and controlled activity.

Adits

An adit is a horizontal excavation that is used in mountainous areas where the ore body is located near or above the valley floor. This type of development is the most difficult to design in certain aspects, being commonly considered only where topographic relief is considerable. In this opening, the ore and waste can be taken down and out of the mine at minimal operating cost. All the horizontal openings are developed by a process called drifting or tunneling. The traditional method of performing this operation is to drill and blast the face, load the material into a haulage device, and then provide support and ventilation to the newly advanced face. Thus, drilling and blasting are the standard excavation method for drifting. The exceptions to the use of blasting are underground mines in relatively soft rock such as coal and salts where the rock can be removed without the need for blasting (Stevens 2010). In addition, using explosives in underground coal mines creates a significant safety hazard because methane gases and dust associated with the coal can ignite.

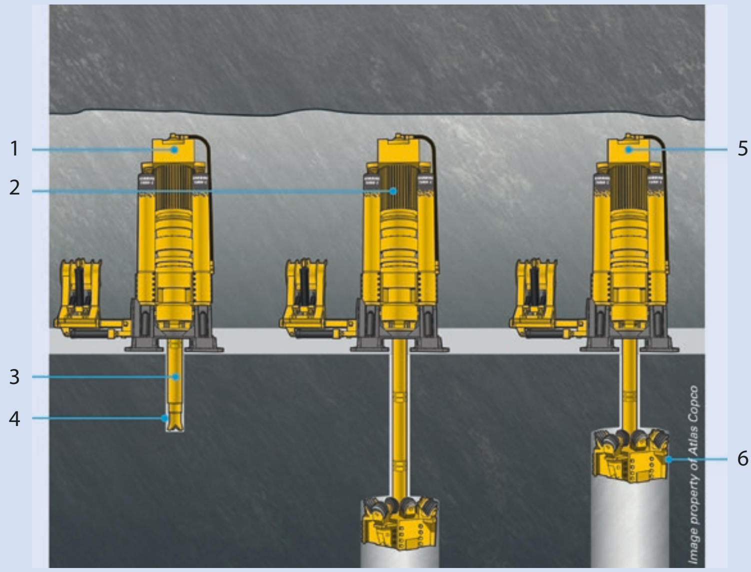


Fig. 5.51 Raise boring process (Illustration courtesy of Atlas Copco)

Declines

A decline or ramp is a tunnel (Fig. 5.41) usually sunk at a low slope angle (<20° dip). The design of declines is considered as one of the main issues in underground mine development. They are straight, spiraled, or a combination of both. Ramp access is the common selection in shallow ore bodies, especially where the mineralization is near horizontal. A ramp from surface can facilitate machine movements and transport of people and materials. It can also be used for ore transportation by truck or conveyor, eliminating the need for hoisting shafts. Ramps are sizing to include machines that pass through or operate inside. Space must incorporate a rational margin for clearance, walkways, ventilation ducts, and other facilities. Cross sections vary from 2.2 m × 2.5 m in mines with a low degree of mechanization to 5.5 m × 6.0 m where heavy equipment is used (Nord 2007). In many mines, the decline is used to transport ore to the surface through a conveyor belt, being associated with grade limits. For instance, if utilized for conveyor belt haulage only, the maximum grade of the decline could be from 15° to 25° depending on material to be conveyed.

5.5.3 Underground Load and Transportation

The fragmented ore is removed from the mine by loading it – called mucking in underground terminology – onto transportation equipment and hauling it out of the mine. The load, haul, and dump processes are carried out using a load-haul-dump (LHD) truck (hence its name), also known as Scooptram. LHD units are commonly used to move ore from the stope to a crushing plant or waiting truck to be transported to the surface. They are adequate for small and large tunnels, chambers, and stopes. In ramps and adits, the LHD will dump its load onto a haul truck or onto a conveyor for transportation to the surface. In mines with a shaft, the LHD will commonly dump its load directly into an ore pass where the ore will fall near the bottom of the shaft into a crusher. From there, it will be hoisted or skipped to the surface. For long ramp operations, the LHD/truck combination generates lower operating costs than LHD alone, being considered on any haul more than 500 m in length. LHD or Scooptram can be used with remote control technology, which utilizes a transmitter and radio receiver to control and monitor the operations of the LHD (Fig. 5.52).

Fig. 5.52 LHD used with remote control (Image courtesy of Matsa, a Mubadala and Trafigura Company)

Another possibility to transport the ore in underground mines is where a continuous miner is utilized to cut soft materials continually. Where drilling and blasting are not required, the focus of the operation is the continuous miner (Fig. 5.53). This machine consists of a central body to carry all other components mounted on some type of drive mechanism to provide mobility and a cutting head usually rotating drums equipped with tungsten carbide teeth that cut into the rock. An internal gathering system then loads the broken ore onto an onboard conveyor, and it feeds onto a shuttle car or articulated hauler, which takes the product to an optional mobile belt feeder. If present, the feeder puts the product onto a conveyor belt, which in turn carries the ore to the surface.

5.5.4 Rock Support

Rock support is the term utilized to outline procedure and materials used to enhance the stability and maintain the load bearing capacity of rock near to the limits of an underground mine. Thus, the primary aim of support processes is to conserve the intrinsic strength of the rock mass so that it becomes self-supporting. Rock support is essential in underground workings for both the safety and the productivity of the mine. It is still the bottleneck in the working cycle in underground mining. The selection of the support type installed in an underground excavation is based on the extent of the zone of loosened or fractured rock surrounding the

Fig. 5.53 Different types of continuous miner (**a** image courtesy of Iberpotash; **b** image courtesy of PotashCorp)

excavation. The support of excavations is commonly classified as primary or secondary. The former is applied during or immediately after extraction to ensure safe working requirements during further excavations, whereas the latter is applied as any additional support or reinforcement at a later stage. Support can also be separated into active or passive; active support (e.g., tensioned rockbolts) means a predetermined load to the rock surface at the same time of installation while passive support (steel arches) is not installed with an applying load and develops its load as the rock mass deform. More commonly used surface rock support methods are rockbolts and grouted cables as active rock supports, and mesh, shotcrete and steel sets as passive rock supports.

Mechanically anchored rockbolts are probably the earliest type of rock reinforcement utilized in underground operations to prevent major ground

Fig. 5.54 Installing rockbolts at Bathopele mine (South Africa) (Image courtesy of Anglo American plc)

failure (Fig. 5.54). Moreover, they are yet the most usual way of rock reinforcement utilized in mines worldwide. In this method, «holes are drilled into the roof and walls and long metal bars are inserted to hold the ground together; point anchor or expansion shell bolt is a metal bar of 20–25 mm in diameter and 1–4 m in length and, as the bolt is tightened, the expansion shell located at the top end expands and the bold tightens holding the rock together» (Haldar 2013). Tensioned rockbolts are most useful to retain loose blocks or wedges of rock near the surface of the excavation.

Rockbolts can be substituted by cable bolts (Fig. 5.55) grouted with cement. They are utilized to bind large masses of rock in the hanging wall and around large excavations, being much larger than standard rockbolts (e.g., between 10 and 25 m long). The main advantage of these cables is that they are installed in openings with very low headroom. Grouted cables are very effective in applications such as the reinforcement of ore or waste passes. Grouting serves two main purposes in rockbolt installations. First, it bonds the bold shank to the rock making it an integral part of the rock mass and enhancing the interlocking of the components of the rock mass; second, grouting offers protection against corrosion. For this reason, rockbolts installed for long-term use must be grouted.

Regarding the passive rock supports, the installation of mesh on the backs and sidewalls of an excavation is a method that can largely remove unintended fall of small rocks. However, this type of support system is not developed to support large static or dynamic loads. In this case, it can only be utilized in combination with other components such as rockbolts and grouted cables to constitute a global integrated system. There is a great variety of mesh forthcoming, but the three major types are welded wire mesh (10 × 10 cm openings), chainlink mesh, and nonmetallic mesh. Galvanized or nonmetallic mesh is recommended where corrosive conditions exist.

Sprayed concrete (gunite or shotcrete) (Fig. 5.56) has a long history of being used as a surface support in mines. There are two application methods for sprayed concrete: dry mix and wet mix, having each type its special utilization in

Fig. 5.55 Installing cable bolts at Dishaba Mine (South Africa) (Image courtesy of Anglo American plc)

Fig. 5.56 Spraying shotcrete onto the walls of a drift underground at Snap Lake Mine (Canada) (Image courtesy of De Beers)

Fig. 5.57 Installation of canopy jack at Tumela Mine (South Africa) (Image courtesy of Anglo American plc)

surface rock support. The present tendency is to utilize fiber-reinforced shotcrete or Fibercrete. It forms actually a very versatile support technique with the addition of microsilica to the mortar mix. The mixture coats 50–100 mm thick layers on the roof and walls anticipating smaller fragments from falling (Haldar 2013).

Regarding the ancient methods of support, steel set has commonly substituted timber as the traditional passive support technique in underground mining. In general, steel or timber sets only generate support instead of reinforcement. In hard-rock mining, steel sets have restricted utilization because most support duties can be carried out more efficiently using rockbolts, shotcrete, or combination of these systems. The main exception is in extremely broken ground related to the presence of faults of shear zones. In such cases, it can be unable to anchor the rockbolts in the rock mass, being thus steel sets needed in order to carry the dead weight of the failed material surrounding the opening. Thus, the subsidence of the roof can be supported by steel sets. A wide range of rolled steel sections are available in the market. Figure 5.57 shows the installation and inspection of canopy jacks to secure a brow at Tumela PGM Mine (South Africa). The image also displays timber passive support.

5.5.5 Underground Methods

There are many different underground methods that have been developed to respond the needs of differing geometry and the geotechnical features of the host and surrounding rock. These underground mining methods, called stoping by the American miners, are difficult to classify rationally since each method depends not only on ore body geometry but also includes other consider-

Fig. 5.58 Underground mining using unsupported method (Spain) (Image courtesy of Iberpotash)

ations such as ground conditions, hydrology, grade distribution, the presence of structures (e.g., faults or dykes), scale of operations, economic factor, availability of labors, and materials/equipments as well as environmental considerations.

The reason why the choice of a method is crucial is that it largely governs the type and placement of the primary development openings. If disturbance of the surface due to subsidence, inevitable with caving methods and possible with other methods, is anticipated, then all the access openings must be located outside the zone of fracture bounded by the angle of draw. The angle of draw is the angle between a vertical line drawn upward to the surface from the edge of the underground opening and a line drawn from the edge of the opening to the point of zero surface subsidence. The larger the angle of draw, the wider will be the area on the surface in which subsidence should be present.

To show the significance of ground support, underground mining methods can be classified in three main types based on the extent of support required: (a) methods generating openings that are naturally supported or requiring minimum artificial support, (b) methods requiring substantial artificial support, and (c) caving methods in which failure of the back roof is inherent to the extraction process. Underground mining method can also be separated in selective and bulk methods. The former are utilized to recover ore without dilution, whereas the latter are used to extract large tonnages of ore with low cost. Evidently, selective methods are more expensive per ton of rock extracted than bulk methods, but the revenue per ton of ore is greater. Selective methods typically apply to narrow precious metal vein deposits and high-grade base metal veins such as those hosting lead and zinc, whereas bulk methods are used for mining low-grade large ore bodies which cannot be extracted profitably using selective mining methods. In this section, the goal is to summarize briefly the main characteristics of the major underground mining methods according to the first classification (ground support).

The unsupported methods (Fig. 5.58) of mining are generally utilized to mine mineral deposits that are roughly tabular, plus flat or steeply dipping, and are commonly related to high competent ore and waste rock. They are termed with this name

Fig. 5.59 Two rooms and one pillar in a room-and-pillar underground system (Spain) (Image courtesy of Pedro Rodríguez)

since they do not utilize any type of artificial element to help in the support of the openings. However, a great number of roof bolting and localized support measures are commonly needed. In room-and-pillar method, a classical unsupported method, the support of the roof is generated by natural pillars of the mineral that are left standing in a systematic configuration (Fig. 5.59).

Supported mining methods need important amount of artificial support to keep stability in openings as well as systematic ground control throughout the mine. They are utilized in mines with ground conditions ranging in competency from moderate to incompetent. In fact, the supported method is basically used where the other two types of methods, unsupported and caving, are not appropriate. Cut-and-fill stoping is the most typical of these methods and is utilized in steeply dipping metal deposits.

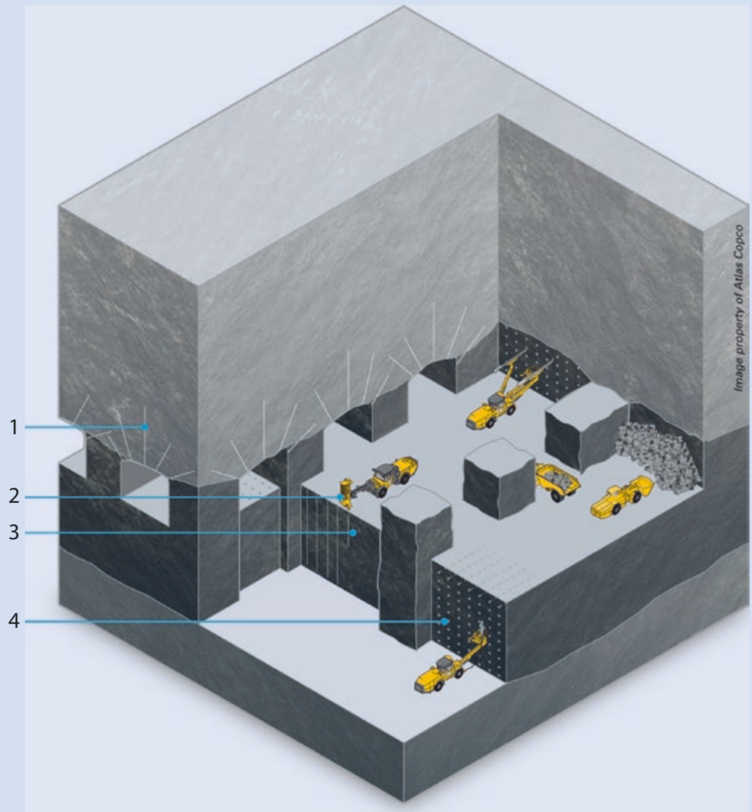
The third group, caving methods, is varied and involves induced, controlled, or massive caving of the ore body and/or the overlying rock. The mining workings are defined to collapse with intentional caving of the ore and/or host rock. Subsidence of the surface normally occurs afterward. Two methods of this group widely applied due to their high productivity are longwall mining and block caving.

Unsupported Methods

Room and Pillar

Room and pillar is the most classical unsupported method (Figs. 5.59 and 5.60). It is planned for mining of flat, bedded deposits of limited thickness, normally showing an inclination that does not exceed 30°. Examples are sedimentary deposits such as limestone or sandstone containing lead, salt layers, phosphate, some base metal deposits (Box 5.8: Rudna Copper Mine), limestone, magnesite, and dolomite. This method recovers the mineralization as completely as possible in open stopes, called rooms, leaving pillars of ore to support the hanging wall (hence the name room and pillar) but without jeopardizing working conditions and personal safety. The dimensions of rooms and pillars depend upon factors such as the stability of the hanging wall and the ore, the thickness of the deposit, and the rock pressure. In this respect, the stability of the ore and the hanging wall is a flexible concept. Increasing the number of pillars and reducing the room width can compensate for poor ground conditions, but ore recovery is sacrificed since a larger portion of the ore body is left to support the back. Although it is not common, sometimes areas of waste can be utilized as pillars.

Fig. 5.60 Room-and-pillar method (Illustration courtesy of Atlas Copco)



1 - Bolting and cablebolting
2 - Vertical benching
3 - Pillar
4 - Flat benching

Box 5.8

Rudna Copper Mine (Polkowice, Poland): Courtesy of KGHM

The Rudna mine is the largest copper ore mine in Europe and one of the largest deep copper ore mines in the world. Rudna mine is located in Lower Silesia, north of Polkowice city. Industrial resources of Rudna mine (31.12.2015) in four operated deposits are 432 million Mt. of copper ore with an average grade of copper of 1.88%. Average thickness of Rudna deposit is over 4 m nowadays, and over 70% of resource is over 3 m thick. The deposit series includes three lithological links: Upper Permian carbonate rocks, clay-dolomite shale, and white sandstones of White Footwall Sandstone. The share of individual lithological types of the ore in the balance ore resources is as follows: carbonate ore, 11% of ore resources; shale ore (Kupferschiefer), 6% of ore resources; and sandstone ore, 83% of ore resources. Copper-bearing shale (Kupferschiefer) contains the highest grade of copper (6%). The depth of copper ore body ranges from 844 m up to 1250 m in depth.

The Rudna deposit displays varying and differentiated mineralization. The main ore minerals are chalcocite, digenite, bornite, chalcopyrite, covellite, and tennantite. The carbonate-shale ore contains both the distributed forms in the form of grains and aggregates as well as ore pockets and veins. The ore minerals are usually dispersed in the sandstone ore and are present in either binder or ore laminate form. The highest concentration of the ore minerals is observed in the shale ore. The accompanying elements include mainly silver, lead, cobalt, nickel, vanadium, and molybdenum.

Rudna deposit is mined using room-and-pillar underground mining method. Primary access to production areas is provided by main development headings driven from the shaft. Each production area is divided into mining sections, and each section is prepared for mining by driving tunnels on all four sides to verify geological continuity and ore grade. Mining sections are located primarily beyond the limits of the major pillars required to protect shafts, permanent underground installations, and surface facilities. For the extraction of ore in a mining section, a series of parallel tunnels (rooms) are driven in an updip or cross dip direction, with support provided by roof bolts. Connections are made between the rooms by driving tunnels, essentially at right angles to the rooms, at regular intervals. The result is that a series of more or less square or rectangular pillars are left in place between the rooms and crosscuts. This phase of mining is referred to as primary extraction.

After primary extraction has been completed in a number of adjacent sections, or in a complete production area, and provided that there is no adverse impact on other areas, the ore that remains in the pillars can be partially recovered in a phase of secondary extraction. Thus, secondary extraction involves removing ore from all sides of the pillar, thereby reducing its size. Mining areas are sealed following secondary extraction in order to prevent further access and are then allowed to cave naturally. It is understood that the current system of primary and secondary mining is capable of extracting 75–90% of the in situ ore.

Where the ore is less than about 6 m thick, single-level mining of rooms is carried out, with pillar dimensions varying from 5 to 7 m by 5–7 m to as large as 17 m by 17 m. If the ore is thicker, rooms are excavated in two levels. The upper level, under a dolomite roof, is mined first, with a subsequent extraction of a lower-level bench. The resulting void is filled with hydraulically placed sand. In recent mining operations, backfilling of the mined-out void, with hydraulically placed sand, is carried out. Underground mining operations are fully mechanized and, generally, employ sufficient units of equipment of appropriate size. Mining is conducted in the following cycle: (a) drilling the blasting holes with the support of self-propelled drilling rigs, loading of blasting material to drilled holes by drilling rigs, group blasting of the ore, followed by the ventilation of the areas blasted (from 30 min. to 2 h; in seismic areas this time is longer) and (b) loading of the ore using self-propelled loaders into haulage vehicles and its transport to dumping stations and protection of the exposed face by anchor bolts using bolting rigs; the crushed ore is then transported mainly by conveyor belts (◘ Fig. 5.61) to the storage sites by the shafts and is hoisted to the surface.

Given the extent of the developed underground area, water inflow to the workings is extremely low. Underground inspection confirms that there is minimal evidence of any significant water flow. The installed pumping capacity provides a substantial margin of safety in comparison with average inflows. In certain portions of the mining area, the overlying dolomite and limestone beds form an aquifer that has the potential to release significant short-term water flows into the underground

◘ **Fig. 5.61** Conveyor belt to transport ore (Image courtesy of KGHM) workings. In such areas, retention reservoirs have been constructed to provide storage in the event that short-term inflows exceed pumping capacity. Since Rudna mine employs hydraulic backfill in certain mining areas, drainage from the backfill contributes to the overall mine pumping requirement. It is understood that all water pumped from the underground workings are delivered to the tailings storage facility, from which it is either recycled to the concentrators or treated and released to the river.

In this method, the ore is blasted and the material loaded in the room where it was extracted and transported to a point where it will flow, either by gravity or mechanical means, to a central gathering point to be taken out to the mine. This is because the direction of excavation (angle of dip) is below that which would cause the dry material to flow by gravity to a drawpoint or gathering point. The loose rock is then translated by dump trucks or LHD vehicles to the surface for waste disposal or processing in the case of mineralization. In thin ore bodies, loading points can be necessary for transferring ore from loader to hauler. As all activities are carried out on one or very few levels covering a large area, there are many faces available at any time, so high equipment utilization is possible. Thus, this method of extraction is well adapted to mechanization. All tunnels are excavated by drilling and blasting, and the production rate ranges from 500 to 35,000 tons per day, being the recoveries of extraction obtained in mining in advance as high as 85%. In soft-rock deposits such as salt or coal seams, drilling and blasting are not required and the valuable mineral is extracted using machines such as continuous miners. Mineralized heights greater than about 6 m are commonly operated by multiple passes. Barren rock originated during extraction can be easily disposed in the mined voids.

Rooms and pillars are commonly disposed in regular configurations to simplify planning, design, and operation, being designed with circular or square pillars and elongated walls dividing the rooms. Mining the ore body creates large openings where machines can travel on the flat floor. Since personnel works continuously under exposed roof, close observations of the performance of roof and pillars are needed. Rockbolts are used extensively as rock reinforcement. Usually, the pillars remain after mining is complete and they are not recovered because it is difficult and expensive. However, where all the ore in the openings has been mined and translated to the surface, in a second phase of this method, several pillars can be mined out prior to abandoning the stope. This is because they still have valuable grade content. In this case, some pillars must be left standing to maintain active support for the hanging wall. It is common in this method to collapse the rock mass into the rooms sometime after the extraction process has finished.

The main advantages of the room-and-pillar method are the high degree of flexibility and the high degree of mechanization since many aspects of the mining cycle are repetitious. It is a very selective mining system leaving waste material on pillars, being also relatively inexpensive. It can be operated in multiple fronts and does not require much anticipated development. Regarding the disadvantages, the method requires maintenance of the roof and eventually the pillars, the loss of ore in pillars, and the need of significant capital investment for extensive mechanization. If the method progresses in depth, the tension in the open space increases significantly.

Sublevel Stoping

Sublevel stoping (▪ Fig. 5.62) is an unsupported method used for mining mineral deposits with steeply ore bodies and regular boundaries, stable rock in hanging wall and footwall, and competent mineralization and waste rock. This method requires extensive ore body development with relatively high capital expenditures, but production costs are comparatively low because much of the development is in ore (Lawrence 1998). The thickness of the deposit between the hanging wall and footwall usually varies from a few meters to tens of meters wide. In this method, mining starts at the bottom of a level and proceeds upward. The ore body is vertically divided into levels, and between two levels, the stopes of convenient size are formed, hence the name sublevel stoping. Leaving a crown pillar at the top of the stope safeguards the level above while lower level is utilized as haulage level to collect the mineralization from the stopes. The level developments, commonly in the footwall, range from 50 to 150 m, based on the vertical extent of the ore body and the number of production openings that can be extracted in each level. Between the

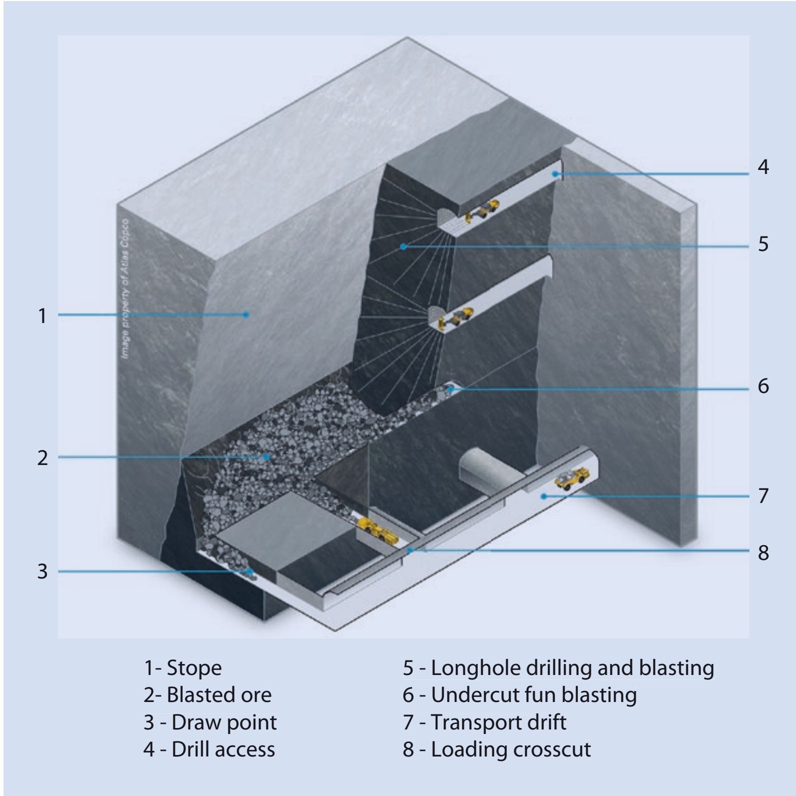


Fig. 5.62 Sublevel stoping method (Illustration courtesy of Atlas Copco)

main levels, ramps are usually driven for haulage transport. These ramps also give access to the sublevels, which are developed at intervals to remove blocks of ore. Dilution with waste rock can occur if ore boundaries are irregular or if caving occurs, but usually 100% of the ore of the stope is recovered.

The ore is blasted from different levels of elevation. Production mining is achieved almost exclusively by longhole drilling, and the length of the holes depends on the shape of the ore body and the predetermined sublevel spacing. Longholes do not exceed 25 m because hole deviation and control become major problems beyond this length. In blasthole open stoping, the ore is blasted in vertical slices, whereas in vertical crater retreat (VCR), the ore is blasted in horizontal slices. Broken ore reports to the drawpoints for extraction. The loading can be carried out using remote control LHD working in the open stope, which reduces the amount of drift development in waste rock. Once the stope is definitely mined, a backfilling process is performed with a mixture of sand and rocks, waste rock with cement, or dewatered mill tailings. The backfill material must have a lot of strength to support the roof of the empty stope. This process allows for recovery of the pillars of unmined ore between the stopes, producing a high recovery of the mineralization; pillar recovery is a common practice in this method. Successful ore recovery would then require draw of fragmented ore from beneath less mobile, barren country rock (Box 5.9: Aguas Teñidas Polymetallic Sulfide Mine).

Box 5.9

Aguas Teñidas Polymetallic Sulfide Mine (Huelva, Spain): Courtesy of Matsa, a Mubadala & Trafigura Company

The Aguas Teñidas Mine is based on one of an east-west striking chains of volcanogenic massive sulfide (VMS) deposits on the northernmost limb of the Iberian Pyrite Belt. The mine geology is comprised of heavily tectonized volcano-sedimentary sequences, with crosscutting thrust faults and shear zones. The main lithological units at the mine comprise a footwall rhyodacitic unit, massive sulfide mineralization, and a hanging wall volcano-sedimentary unit. The deposit includes four mineralization types: polymetallic lead/zinc rock, massive cupriferous, barren pyrite, and a cupriferous stockwork. The principal ore minerals are sphalerite, chalcopyrite, and galena. Pyrite generally forms 50–80% of the massive sulfide. Both massive sulfide ores (polymetallic and cupriferous) are hosted in a massive pyrite structure and are identified from the pyrite host rock by grade rather than any physical differences.

Geotechnical practices may be summarized as: (a) geotechnical mapping: lithology, RQD, rock strength, and various joint factors are recorded in detail for all excavations and geotechnical diamond drillholes; (b) Q class: all of the information obtained from the mapping is combined to make a single encompassing Q class number; this number is decimalized and ranges from very small (<0.4) for very poor ground to +10 for very good ground. (c) geotechnical domain: the criteria described are applied to assign a single domain number based on the derived Q class number; the domain number ranges from 1 for very poor ground to 5 for very good ground. For planning purposes, support requirements have been defined according to the geotechnical domain and whether the development is permanent or temporary. These requirements have been defined in terms of rockbolting, mesh, shotcreting (gunite), and cable bolting. For the poorest ground (Domain 1), the support required includes Split Set rockbolts, mesh, shotcrete and cable bolts on both the floor and walls. For the best ground (Domain 5), in temporary excavations, only Split Set rockbolts are required.

The access to the mine is principally provided by two-ramp systems. The eastern ramp provides access for all service vehicles, personnel, and temporary mucking access where required. The western ramp provides the principal ore haulage system. This ramp is approximately 3.7 km long and has a portal at the extreme western side of the property, very close to the processing plant. Ore haulage is currently provided by a fleet of mostly 30 t trucks. Stope mucking is achieved by direct loading of trucks by Scooptrams from the stopes as well as a system of ore passes. The ore passes are generally 3 m diameter raised bored excavations, or 4 m x 4 m developed raises. In poorer ground, one of these ore passes has been steel lined. At the current time, the ore from these ore passes is loaded into trucks at the base, by dedicated Scooptrams. There are currently three operational ore passes.

The basic level vertical spacing is 30 m. An independent spiral system allows access to all parts of the mine from the eastern service ramp, independent of the mine ore haulage ramp. The haulage galleries are currently laid out with an offset of approximately 50 m from the massive sulfide northern contact. The ore body is basically partitioned into 20 m wide stope panels. However, instead of individual perpendicular crosscuts on each panel line, the crosscuts have been designed as 'Y's, with the perpendicular part off the haulage drives every 40 m and then with branches off to the different stopes. This has advantages in terms of requiring less development and less disruption off the haulage drives where the secondary stopes need to be accessed. The orientation of design of these 'Y's is customized according to local ground conditions. In general, the waste crosscut drives have been designed as being relatively flat. However, in some cases due to local stope block variations, the crosscuts have been inclined, up to a maximum of 20%.

On any 20 m wide panel, stopes are generally mined out from bottom to top. If there is more than one stope on the same elevation for any panel, the stopes are mined out from south to north. All longhole drilling provides 89 mm diameter drillholes. Most of the production drilling is from the top down, in general from a central upper crosscut. ANFO is used for stope blasting. Longholes are typically spaced with 2.6 m burden, spaced laterally at 3.2 m. Blasting results to date have been very good.

For primary stopes, a high-strength (HS) paste fill is used (Fig. 5.63). As the mine evolves, more secondary stopes will be mined out, which can be filled with a mixture of low-strength (LS) paste fill and waste. For the earlier years, the ratio of paste fill types will be 2:1 between HS/LS. In later years, as a higher proportion of secondary stopes are mined out, the ratio will switch round to approximately 1:2 HS/LS. In most years, a total amount of paste fill (HS + LS) of approximately 400,000 m^3 will be required. Most stope muck is first transported to the ore pass systems via the haulage galleries. From there, it is loaded into trucks for haulage up the main haulage ramp to the surface stockpiles.

The scheduling work was completed using mining software. The main objective of the stope production was to achieve MATSA production targets, leading to annual production of 2.2 Mt. of ore, combined polymetallic and cupriferous. In the schedule setup, stope production is built up from the different mine activities: stope development, cable bolting, longhole drilling, slot development, production, and paste filling. Individual rates are applied to the different activities, and from these the overall stope production is built up.

Fig. 5.63 Paste fill used in the stope (Illustration courtesy of Matsa, a Mubadala and Trafigura Company)

Shrinkage Stoping

This underground method is a flexible technique for narrow ore bodies that do not need backfill during stoping. Shrinkage stoping (Fig. 5.64) is usually employed in the extraction of medium to steeply dipping veins with well-defined ore/waste boundaries and where the walls are necessarily hard to support themselves during the extraction process. Shrinkage stoping is a traditional, labor-intensive, low-productivity method that requires a long lead time for total extraction of a zone. It is not clear the allocation of this method because shrinkage stoping is considered an underground mining supported method (e.g., Hamrin 1998) and also an unsupported method (e.g., Tatiya 2013). The method involves vertical or subvertical advance of mining in a stope, with the fragmented mineralization utilized as both a working platform and temporary support for the stope walls. This requires considerable planning and coordination.

The method is comparable with cut-and-fill method (next heading), with broken ore temporarily fulfilling some of the functions of backfill. Shrinkage operations follow the sequence of drilling and blasting, ore extraction, and scaling and supporting. Once the stope has been mined to the full design height, mineralization is drawn until either the stope is empty or until dilution due to stope wall collapse becomes excessive. In principle, there is not provision for support, so the wall rocks must be strong and competent.

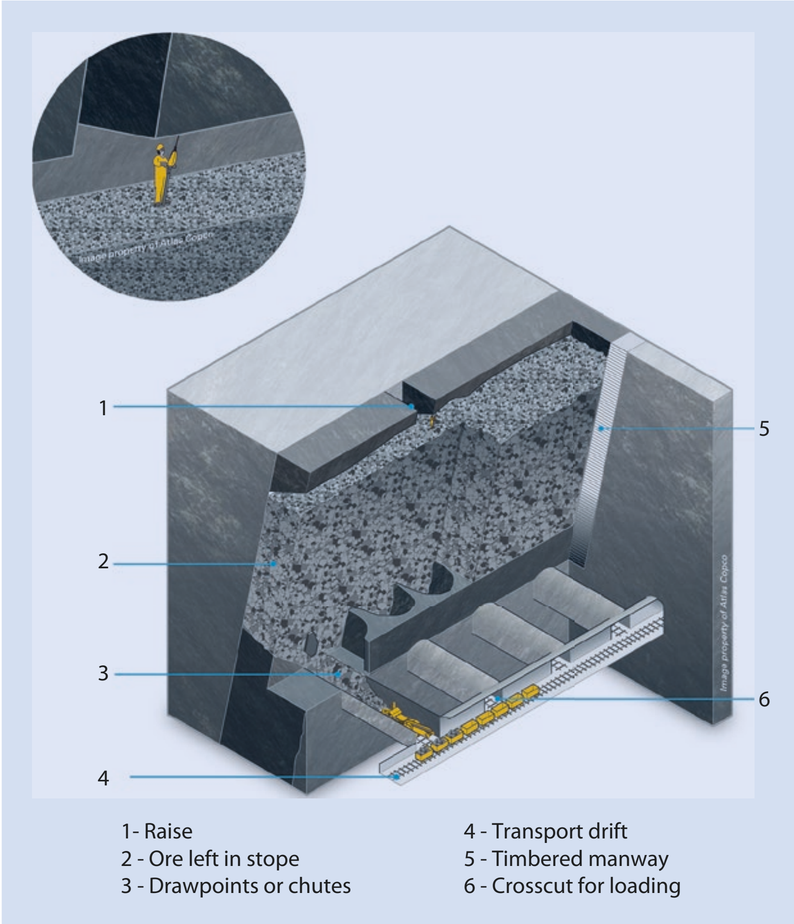


Fig. 5.64 Shrinkage stoping method (Illustration courtesy of Atlas Copco)

Narrow vein shrinkage stoping is not a selective method provided that once initiated the whole ore has to be mined. Blasting swells the ore by about 50% or more, which means that a substantial amount has to be left in the stope to keep a suitable working distance between the back and the top of the broken ore. When the stope has advanced, it is discontinued and the remaining 50% of the mineralization can be extracted, hence the name of shrinkage stoping. As a general rule, the mineralization must be strong and resistant to crushing and degradation during draw because it is necessary to assure that once the ore is mobilized by blasting, it remains mobile and suitable to flow during its residence time in the stope.

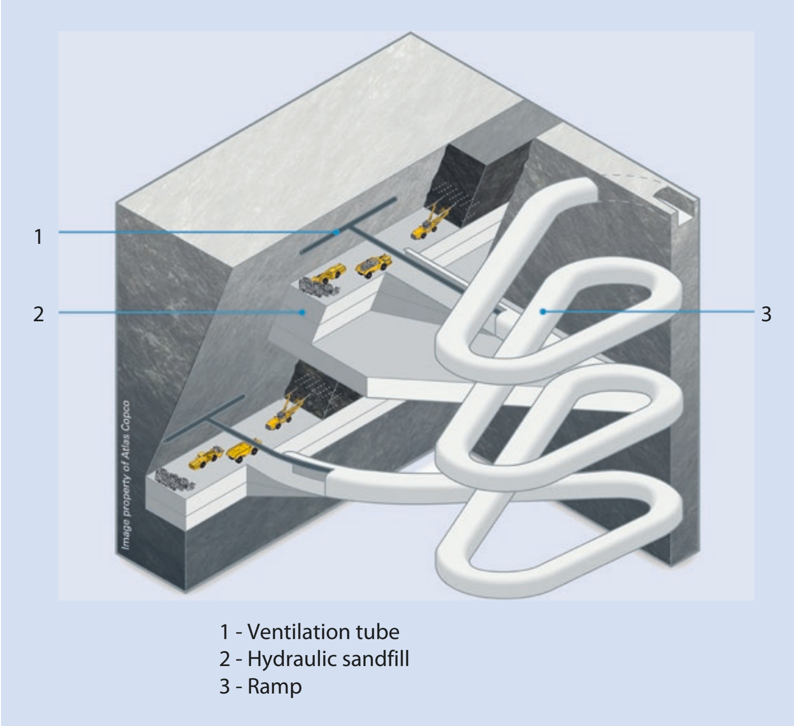
Supported Methods

Cut and Fill

Supported methods are commonly utilized in mines with weak rock structure and cut and fill (Fig. 5.65) is certainly the most common of these methods (Box 5.10: Efemcukuru Gold Mine). For many years, it was probably the main mining method used in underground metal mines, especially those in poor ground conditions. It is frequently applied in vein deposits where the vein is moderately to steeply dipping with considerable vertical extent, although the method is readily adaptable to almost any ore body. The ore body however must be accessible at both top and bottom as well as at regular intervals throughout vertical extent. In general, the cut-and-fill method is preferred for vertical or subvertical mineral deposits at great depths or within relatively weak rocks that need support. Mine planning and supervision are concerned with the geotechnical properties of the fill and their effects on mine and stope stability.

It is preferred for ore bodies with irregular shape and disseminated mineralization where high-grade sections can be mined separately,

Fig. 5.65 Cut-and-fill method (Illustration courtesy of Atlas Copco)



Box 5.10

Efemcukuru Gold Mine (Izmir, Turkey): Courtesy of Eldorado Gold Corporation

The Efemçukuru deposit is located near the west coast of Turkey, approximately 20 km from the provincial capital city of Izmir on the Aegean coast, in a mountainous area known as Tepe Daği. The immediate project area is comprised of a late Cretaceous- to Paleocene-age volcano-sedimentary sequence, which has been regionally metamorphosed to greenschist facies. Narrow rhyolitic dykes cut the immediate host rock. These are unmetamorphosed and largely undeformed and therefore postdate the regional metamorphic collision-related event. Age is reported as a Late Miocene age (11.9 Ma, K-Ar) for rhyolitic rocks (dikes) in the region, and they are thought to be related to the post-collisional extensional magmatism. The rhyolite dikes are thought to be the surface expression of a deeper intrusive body, which is not exposed in the vicinity of the deposit.

Gold and base metal mineralization in the Efemçukuru deposit is hosted in three north to north-west-trending epithermal veins, being the main vein the Kestane Beleni vein, which is a low-sulfidation epithermal vein. The known Kestane Beleni vein structure extends over 1200 m on surface. The deposit comprises two ore shoots, Middle Ore Shoot (MOS) and South Ore Shoot (SOS), with an average dip angle of approximately 60°. The vertical extent of the currently defined resource from surface is approximately 350 m. An overall cutoff grade of 4.5 g per ton has been used for all mining methods. Overall dilution from all mining methods is estimated at approximately 11%. Mining recovery of ore is estimated at 92% including mining losses due to pillars and ore in narrow vein structures.

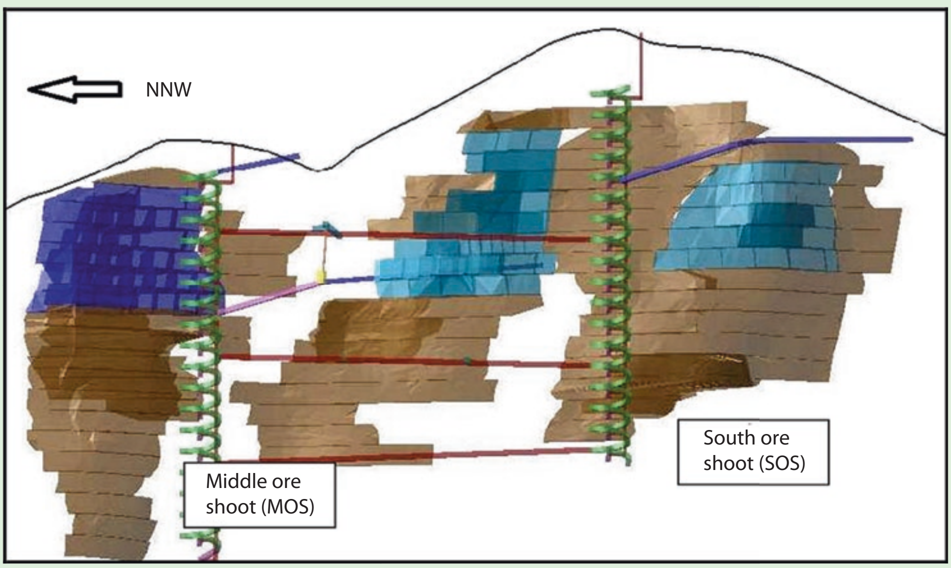
The mine design has been developed to allow flexible access to both the MOS and SOS. Two spiral footwall ramps at each ore body provide access for moving men, equipment, and supplies underground. Advantages of the two-ramp system include increased stope availability, more robust ventilation with increased equipment, and labor productivity. One disadvantage of this approach is the additional cost of waste development for the ramps. Ore is truck hauled to a central ore pass system above the underground crusher before being conveyed to surface via an 800 mm belt conveyor. The ore pass provides 1500 ton surge capacity for underground production with a further 2700 ton capacity in bins on surface. Waste rock is hauled to surface via the South Portal.

Factors taken into account where selecting the mining method at Efemçukuru included (a) continuity, size, and shape of the ore body, (b) local ore body ground conditions (ground support requirements), (c) dip angle of the ore body, (d) achievable production rate based on mucking requirements, and (g) value of in situ ore, mining dilution, and recovery. To minimize development and allow flexibility between mining methods, all mining methods will utilize mining block heights of 16 m, floor to floor.

Mechanized cut and fill is the primary stoping method used for widths between 2 m and 8 m and accounts for 45% of the total production (Fig. 5.66). This method allows selective recovery of ore within the ore body although it is more expensive, has lower productivity, and requires more working faces to meet production targets. Mechanized cut-and-fill lifts are typically the width of the ore body and 4 m high × 4 m deep. The block height will be 16 m floor to floor. Sublevel development will provide access for ore body drilling and blasting, ore and waste haulage, materials and services supply, and ventilation. A one-boom jumbo drills the face, advancing an estimated 4 m per round. Two-boom jumbos are used as required. Blastholes are 45 mm diameter, drilled on a standard overhand heading pattern. ANFO explosives are initiated by dynamite primers with non-electric detonators. Emulsion is required for loading wet holes.

Longhole stopes are used in the SOS where the ore body is wider than 8 m. Mining from longhole stopes easily achieve the full target production rate. The key will be maintaining balance between the longhole and mechanized cut-and-fill production to minimize operating costs and labor requirements. Ore from the MOS and SOS ore bodies will be blended to balance high- and low-sulfide ore and provide a consistent head grade to the mill. The transverse longhole stope access is planned in ore, limiting the number of working stopes available but reducing waste development. Paste backfill is used as a «free standing» structure to control stability of walls, dilution, and safety for the longhole stopes. In the mechanized cut-and-fill stopes, paste backfill is used to stabilize the working floor. In general, rock support includes (a) rockbolts in the side wall up to approximately 1 m, (b) support cables with anchor pins in the side wall, (c) reinforcing with screening, and (d) high-strength cement (8% cement content for working floor).

Conventional trackless equipment is employed to extract ore from mechanized cut-and-fill (MCF) as well as longitudinal longhole (LLH) and transverse longhole (TLH) stopes. Ore from stopes is mucked using 6700 kg capacity LHDs with 3.7 m^3 buckets. Ore is directly loaded into 20 ton articulated dump trucks before being hauled to the central ore pass and crusher system. Remote mucking is required for the longitudinal longhole mining and for mechanized cut and fill when extracting the cut directly below a sill mat. Productivity will be reduced when remote mucking. Underground waste rock is loaded into articulated haul trucks by LHDs and hauled to surface via the South Ramp.



Dark blue: transverse longhole Light blue: longitudinal longhole Brown: mechanized cut- and- fill

Fig. 5.66 Mining method (Illustration courtesy of Eldorado Gold Corporation) leaving the low-grade mineralization in the stopes. Cut-and-fill method is a relatively labor-intensive technique, needing that the value of the ore body be high. Therefore, it is carried out only in high-grade ore where there is a need to be selective and avoid mining of waste or low-grade mineralization, offering better selectivity than sublevel stoping and vertical crater retreat mining. The method is very flexible since multiple activities can be performed at the same moment, for instance, drilling in one level while other levels are being filled.

Cut-and-fill mining excavates the ore in horizontal slices, usually 2.5–3 m thick, starting from a bottom undercut and advancing upward. The ramps are excavated to link the surface to the underground rock. Mining can also proceed with slices mined downward, and the fills form the roof for each subsequent cut. Because the miners in the stope work under freshly blasted areas, the amount of ground control must be great. Since the volume of rock that is broken during one section of mining is relatively small and the amount of nonproductive work required is high, this resulted in limited productivity for the stope. The production from the stope can be quite cyclical because the nonproductive work must be done on a regular basis (Waterland 1998).

The ore is drilled, blasted, loaded, and removed from the stope, which is then backfilled with deslimed sand tailings from the mineral processing plant or waste rock carried in by LHD from development drives. The fill serves simultaneously to support stope walls and as a working platform for mining the next slice. In modern cut-and-fill operations, the fill is distributed by hydraulic means as a slurry. Cement is sometimes mixed in to provide harder and more durable support characteristics. As no rib pillars are left, most of the ore can be recovered with a minimum of waste dilution.

In this method, the development is minimal before mining starts and the equipment investment is relatively small. It is a selective mining method that can also be used to reduce dilution. The main disadvantages of the method include the following: (a) ore production is cyclical; (b) the method is labor intensive and required skilled miners; (c) it is not as suited to mechanization as other methods, so there is lower productivity; and (d) the personnel must work under freshly blasted ground, which creates a safety problem.

Caving Methods

Caving methods rely on the rock breaking into pieces that are small enough to be retrieved from the deposit and to flow into a recovery location without blasting all the ore. Although longwall mining is a classical caving method, sublevel caving and block caving are the most characteristic caving methods. They are bulk mining techniques with high production rates that approach or equal those of a medium-sized open-pit mine. There is little or no opportunity for selective mining parts of the ore body, so they are only used in large tabular deposits with a uniform and generally low-value ore (Stevens 2010). As an example, both methods are commonly used for massive low-grade porphyry copper deposits where the stripping ratio for an open-pit mine is too high or the deposit is too deep for surface mining.

Longwall Mining

Longwall mining (◘ Fig. 5.67) is a classical underground mining method being practiced worldwide to mine thin-bedded, soft rock deposits with uniform thickness and large horizontal extent, particularly coal seams (◘ Fig. 5.68). Not all soft rock ores are suited for longwall mining, which works best in laterally extensive, flat-lying deposits that are primarily free of discontinuities such as faults. Coalbeds deeper than 300 m are usually extracted by longwall mining because the room-and-pillar method would require the use of much larger pillars to support the roof, reducing thus the amount of coal that can basically be extracted. Since a long face (about 100 m or more) defines the method, hence the name longwall mining. Longwall mining requires and ore body dip of less than 20°, with a reasonably uniform distribution of grade over the plane of the ore body.

A variation of the classical longwall method is also applied to hard-rock gold and platinum thin reef-type deposits in Southern Africa. There, drilling and blasting break the rock, and low-production conveying systems clear broken ore from the face. Pillars of timber and concrete are installed to support the roof in up to 3.5 km deep mines. For instance, the sequential grid mining method was adopted in Mponeng Gold Mine (◘ Box 5.11: Mponeng Gold Mine). This has been proven as the best method suited to the deep-level gold mining often associated with seismicity.

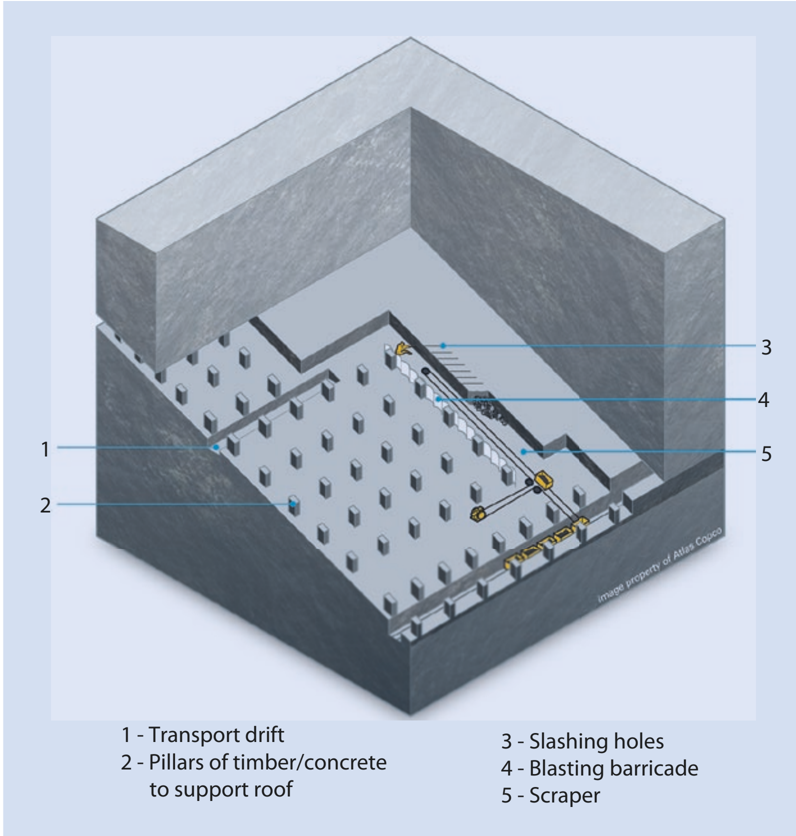


Fig. 5.67 Longwall mining method (Illustration courtesy of Atlas Copco)

Fig. 5.68 Coal seam mined by longwall method using continuous miner (Image courtesy of Anglo American plc)

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Box 5.11

Mponeng Gold Mine (Carletonville, South Africa): Courtesy of AngloGold Ashanti

Mponeng mine (the name means «look at me» in the local Sotho language) is a deep-level gold mine operating between 2800 m and 3900 m below surface and is currently the deepest mine in the world, with grades at over 9 g/t. It is near the town of Carletonville and approximately 65 km west of Johannesburg (South Africa). Formerly known as the Western Deep Levels South Shaft, or No. 1 shaft, Mponeng mine is the most recently sunk of the three mines in the West Wits Operations of the company. The original twin shaft sinking from surface commenced in 1981 and was commissioned along with the gold plant complex in 1986 when mining began. Production started through the use of two hoisting shafts, a sub-shaft and two service shafts. The name changed to Mponeng mine in 1999.

Mponeng is located on the northwestern rim of the Witwatersrand Basin. There are seven gold-bearing conglomerates within the lease area, of which two are economically viable at present. The Ventersdorp Contact Reef (VCR) is the reef horizon mined at Mponeng mine. The VCR forms the base of the Ventersdorp Supergroup which caps the Witwatersrand Supergroup through an angular unconformity. The overlying Ventersdorp lavas halted the deposition of the VCR preserving it in its current state. The VCR consists of a quartz-pebble conglomerate, which can be up to 3 m thick in places. The footwall consists of series of sedimentary layers from the Central Rand Group of the Witwatersrand Supergroup, which, due to the VCR's erosional nature, exposes the youngest sequences in the west to the oldest in the east. The VCR conglomerates are characterized by a series of channel terraces preserved at different relative elevations, and the highest gold values are preserved in these channel deposits. The different channel terraces are divided by zones of thinner «slope» reef, which are of lower value and become more prevalent on the higher terraces and on the harder footwall units.

The other gold-bearing reef with a reported mineral resource for Mponeng is the Carbon Leader Reef (CLR). This reef has been mined at the adjacent Savuka and TauTona mines, and plans are being made at Mponeng to mine the CLR in the future. The CLR at Mponeng consists of a 20 cm thick, tabular, auriferous quartz-pebble conglomerate formed near the base of the Central Rand Group. The CLR is about 900 m deeper than the VCR. In recent years, extensive work has been done in refining the estimation model for CLR.

The VCR has been subjected to faulting and is intruded by a series of igneous dykes and sills of various ages that crosscut the reefs. There is an inherent risk in mining through these faults and intrusives, and a key objective of mine geologists is to identify these geological features ahead of the working face to assist with deciding on the best way to approach and mine through these structures. The VCR reef that Mponeng mines dips at 22° and has an average channel width of 78 cm. Mponeng started stoping in the mid-1980s using a longwall mining method. In the mid-1990s, the mining method was changed to a sequential grid mining method. This is because, since the grade at the operation varies considerably, a sequential grid mining method allows for selective mining and increased flexibility in dealing with changes in grade ahead of the stope. Moreover, this has been proven as the best method suited to the deep-level gold mining often associated with seismicity. The mine utilizes a twin shaft system housing two vertical shafts and two service shafts.

This mining method is a controlled adaptation of scattered mining layouts for a deep mining environment. Planned dip pillars are left systematically and geological structures bracketed. Access development is done in virgin stress conditions. Main haulages are developed on strike of the reef and crosscuts are developed from the haulages to access the reef. Raises are developed on reef from level to level (Stander 2004).

The stope, comprising the hanging wall, face, side walls, and footwall, must be made safe by installing the required support (Split Sets, ◘ Fig. 5.69) and barring of the face and hanging wall. Once the workplace is declared safe, preparation for drilling of the face can begin. The face is marked down in a specific pattern, and holes are drilled accordingly using rock drills. The rock is then scraped into box holes, where it is drawn off into hoppers, or small railway cars, hauled by locomotives. The gold-bearing ore is then hoisted from the lowest underground level of the mine in skips and transported to the gold plant by rail or conveyor belts.

When mining at depths of more than 3000 m, where rock temperatures can reach up to 55°c, cooling systems form a crucial part of the operation. Mponeng mine is currently cooled by four large ice refrigeration plants. Ice is pumped through large pipes to help cool the underground environment. Moreover, ultradeep mines such as Mponeng mine use seismic monitoring stations to transmit signs of movement on the scale of several cm to mine managers.

Fig. 5.69 Installing Split Sets (Image courtesy of AngloGold Ashanti)

Regarding the advantages of the method, it is very effectively and has noticeable production rates and low operating costs. The operation is near definitely mechanized and recuperates a very high amount of the mineralization (Nieto 2011). Electronic controls and automation allow personnel to position themselves away from most of the recognized hazards. In contrast, it needs intense capital investment to cover the highly specialized equipment to create a longwall section, having little selectivity or flexibility after mining commences.

Sublevel Caving

The original application of sublevel caving (Fig. 5.70) was in ground so weak that it would collapse even in small headings where the support was recovered (Cokayne 1998). Sublevel caving can be adapted to large ore bodies with steep dip and continuity at depth. The hanging wall has to fracture and collapse, following the cave, and subsidence of the ground surface above the ore body has to be tolerated. Caving needs a rock mass where both ore body and host-rock fracture are under monitored conditions. As the mining extracts rock without backfilling, the hanging wall carries on caving into the voids. Thus, continuous mining results in subsidence of the surface, where sinkholes can be produced. Sublevel footwall drifts must be stable, requiring only occasional rockbolting.

The ore body is usually divided into sublevels with close spacing at approximately 8–15 m vertical intervals, depending on the plunge of the deposit. Each sublevel is developed with a regular network of parallel drifts that penetrate the complete ore section. Development to prepare sublevel caving stopes is extensive as compared to other mining methods and mainly involves driving multiple headings to prepare sublevels. Ore is fragmented using blastholes drilled upward in fans from these headings. Since the ore is blasted against the caved waste, explosive consumption is very high. Ore is extracted selectively, with a LHD operating in the drill heading. This vehicle transports the rocks to an ore pass where they are elevated to the surface.

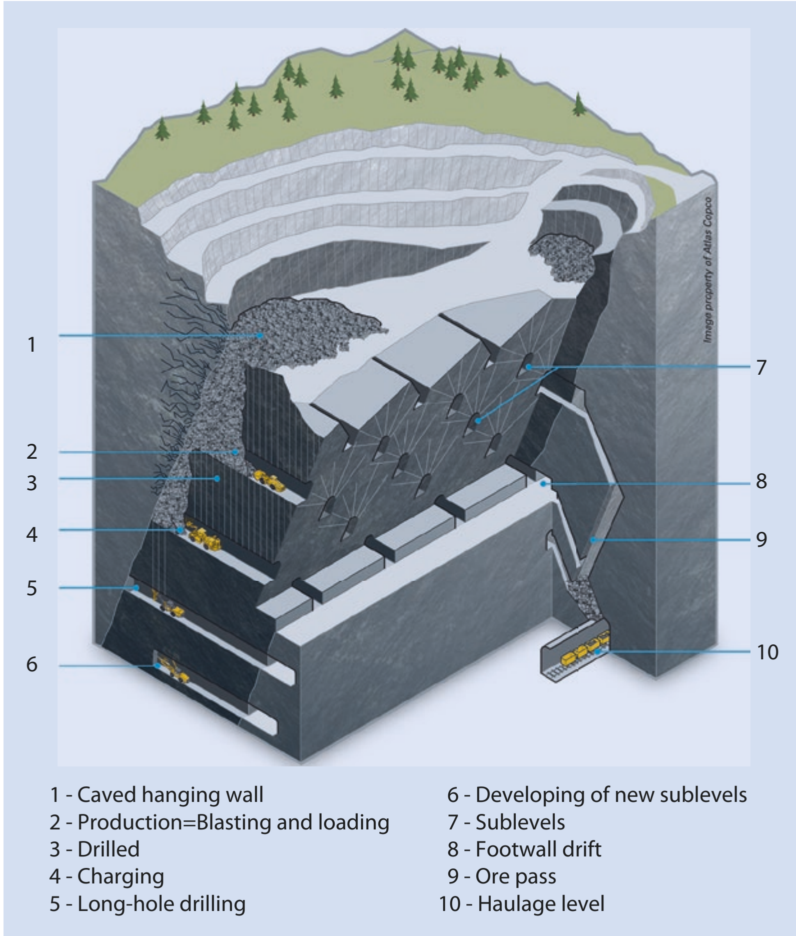
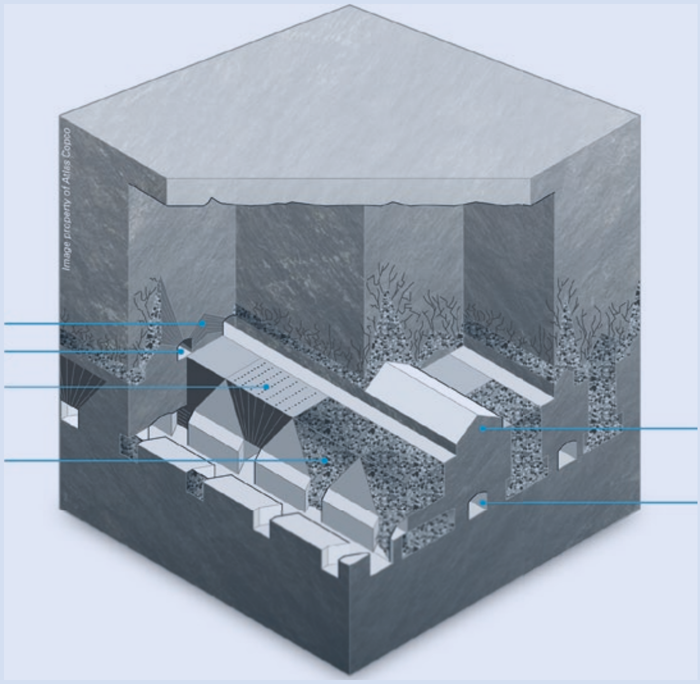


Fig. 5.70 Sublevel caving method (Illustration courtesy of Atlas Copco)

Waste dilution in sublevel caving is relatively high, ranging from 15% to 40%, and ore losses can be 15–25%, depending on local conditions (Fernberg 2007). Thus, in this method the ore must be of sufficient grade to accept the high dilution arising from entrainment of barren country rock in the mineralization. Dilution is of less influence for ore bodies with diffuse boundaries where the host rock contains low-grade minerals. There is always a place for the machines to work, which integrates mechanization into efficient ore production. Consequently, the method is well suited for a high degree of automation and remote operations with corresponding high productivity. The method generates important disturbances of the ground surface, imposing some possible limitations on its applicability, from considerations of local topography and hydrology.

Block Caving

If the ore is wide and steep enough, block caving (Fig. 5.71) would be selected because the cost is normally lower than that for sublevel caving. This method, sometimes called «an upside-down open-pit,» is applied mostly to large, massive ore bodies in which areas of sufficient size can be removed by undercutting, so that the mass above will cave naturally. Where adequately used, this method offers a lower mining cost per ton than any other underground technique (Tobie and Julin 1998) (Box 5.12: Cullinan Diamond Mine). It is applicable only to very large ore bodies in which the vertical dimension exceeds about 100 m. The rather unique conditions limit block caving applications to certain mineral deposits such as iron mineralization, low-grade copper and molybdenum ores, and diamond kimberlite pipes (Fernberg 2007).



1 - Blastholes
2 - Undercut level
3 - Blastholes
4 - Drawbell
5 -Major apex
6 - Production drift

Fig. 5.71 Block caving method (Illustration courtesy of Atlas Copco)

Box 5.12

Cullinan Diamond Mine (Gauteng, South Africa): Courtesy of Petra Diamonds

Cullinan diamond Mine is located in the Gauteng Province of South Africa, being also known as Premier Mine. The site is situated about 40 km east of Pretoria city in Cullinan. The open-pit mining at Cullinan Diamond Mine began in 1903 and is one of the major sources of blue diamonds in the world. Historically, Cullinan Diamond Mine has generated 25% of the world's diamonds over 400 carats. This famous landmark is the source of the most famous diamond ever unearthed, the 3106 carat Cullinan Diamond found in 1905. The stone was so large it was cut into 9 major pieces and 96 smaller brilliant cut diamonds. The mine production will increase up to 2.6 million carats by 2019. The Cullinan ore body has a reserve base of 203.7 million carats (mcts), suggesting a potentially long life for the operation of +50 years. The reserve estimates are based on block cave depletion modeling and external waste. The planned expansion will increase the reserve estimate.

Cullinan diamond Mine is located on a diamond-bearing kimberlite pipe. The carrot-shaped pipe, with a volcanic neck, is considered to be the largest diamondiferous kimberlite pipe of the region. It was first mined in 1871; the mining site is currently referred to as Kimberley's Big Hole. The pipe has a surface area of 32 ha and decreases to a size of 13 ha, 1073 m below surface. The Cullinan kimberlite pipe occurs within the stable, 3-billion-year-old Kaapvaal Craton and intrudes rocks of the Transvaal Supergroup (Pretoria and Rooiberg Groups), Bushveld Complex, and the younger Waterberg Group. The pipe has numerous facies, but there are three dominant facies, namely, the brown TKB, the gray TKB, and the hypabyssal facies, which is contained within the gray TKB. The pipe has intruded through a variety of rocks, the most important of which is norite, the rock type in which most of the current mine haulage system is based. The norite has been correlated with the main zone of the Bushveld Complex. Quartzites, shales, sandstones, and dolomitic shales of the Transvaal Supergroup occur both above and beneath the norite. A unique feature of this kimberlite is the occurrence of an approximately 70 m thick diabase sill (varies from gabbro to norite) that cuts across the occurrence at approximately the 500 m elevation (Chadwick 2012).

Regarding the three major kimberlite facies recognized within the pipe, the brown kimberlite represents the first phase of intrusion and generally has the highest diamond grade of all the kimberlite facies in the Cullinan pipe; the gray kimberlite represents the second phase of intrusion; and the hypabyssal kimberlite represents the final phase of the major facies of the intrusion. The pipe exhibits a wide range of strength characteristics, from uniaxial compressive strengths of some 40 MPa for the brown kimberlite to more than 150 MPa for the hypabyssal.

Petra Diamonds uses the block cave mining method to develop the underground resources. In this method, a drilling level is built up through which the ore body is cut by drilling and blasting.

Once a large area is undercut, caving is started. Mining works are also done at the undercut level. The mining work includes longhole drilling, blasting, charging, and tramming of required ore. The production level is located 15 m below the undercut level and tunnels are bored into the ore body at the level. Drawpoints are developed on these tunnels and raise bored. A draw bell is built up to receive the caved ore, which then flows into the drawpoint. Load-haul-dump trucks load the ore and carry it out of the ore body. Ore is loaded from the drawpoints by a mixed LHD fleet. These are dumped into a series of ore passes sited in the country rock surrounding the ore body. A conventional rail system is used to draw the ore from the ore passes and deliver it to either one of two underground crushing stations on the 805 m level.

Petra Diamonds has planned for a major expansion of the diamond mine. Thus, the C-Cut expansion plan includes the deepening of shafts 1 and 3, related infrastructure to the shaft, and planned level development to the Cullinan ore body (Fig. 5.72). The C-Cut phase 1 area is located 200 m below the existing operations and is designed to develop a new block cave on the western side of the ore body to access the higher-grade BAW and AUC south areas of the ore body. The 350 m deep existing shaft 1 is deepened to 920 m. It will lift up the ore and waste from the new C-Cut phase 1 block once the process of deepening completes. The 60 m deep existing shaft 3 is also deepened to 904 m. It will take workers and material to and from the new C-Cut phase block 1. The South Decline will establish production at 830 m and then on to the bottom of the newly deepened rock shaft at 930 m. Development of the North Decline creates further access to the 830 m production level. For ore handling, the C-Cut will not be using trains and the winze system (e.g., the belts from 804 m to 500 m level). Once the rock shaft has been deepened, LHDs will tip directly into ore passes that will feed into crushers. The crushed feed will discharge onto horizontal conveyor belts at the ground handling level, which will take discharge into the silos from which the skips will be loaded. This will mean a considerable increase in efficiencies and a reduction in costs.

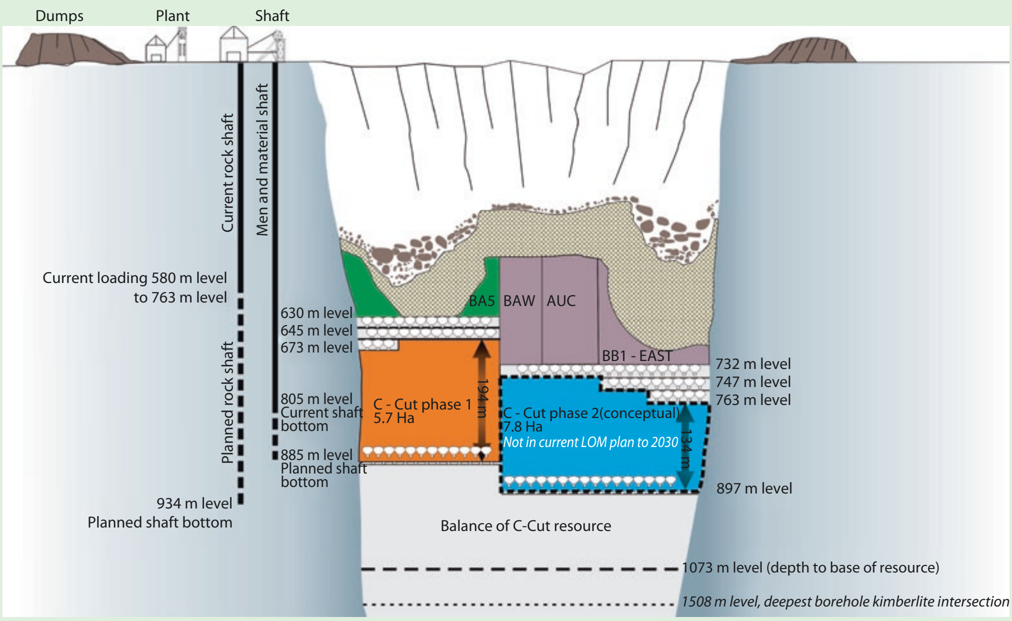


Fig. 5.72 Block cave method in Cullinan Diamond Mine (Illustration courtesy of Petra Diamonds)

Block caving is based on gravity combined with internal rock stresses to fracture and break the rock mass. Caving is induced by undercutting the block by blasting, destroying its ability to support the overlying rock. Thus, gravity forces act to fracture the block. Continued pressure breaks the rock into smaller pieces to pass the drawpoints where the ore is handled by LHD loaders or trains (◘ Fig. 5.73). This method is therefore distinguished from all other commented previously in that primary fragmentation of the ore is carried out by natural mechanical processes. Thus, the elimination of drilling and blasting has advantages in terms of ore body development requirements and other direct costs of production. As fragmentation without drilling and blasting is uneven, a substantial amount of secondary blasting and breaking can be expected at the drawpoints. In this type of underground mining, the rock size and the rate at which rock passes through the drawpoints as well as continuously controlling the stability of the mine are essential (Newman et al. 2010).

Where the ore block breaks up successfully and the extraction is carried out evenly from all of the drawpoints, block caving becomes a low-cost, high-productivity method with good ore recovery and moderate inflow of waste dilutions. Risks are high but the result can be extremely favorable. This method is often used to convert an open-pit operation into an underground mine where surface production can continue while the underground infrastructure is prepared. In fact, the block caving method generates production rates that can approach those of an open-pit (e.g., 100,000 tons per day).

◘ **Fig. 5.73** Ore handling using trains (**a** Image courtesy of AngloGold Ashanti; **b** Image courtesy of Eldorado Gold Corporation)

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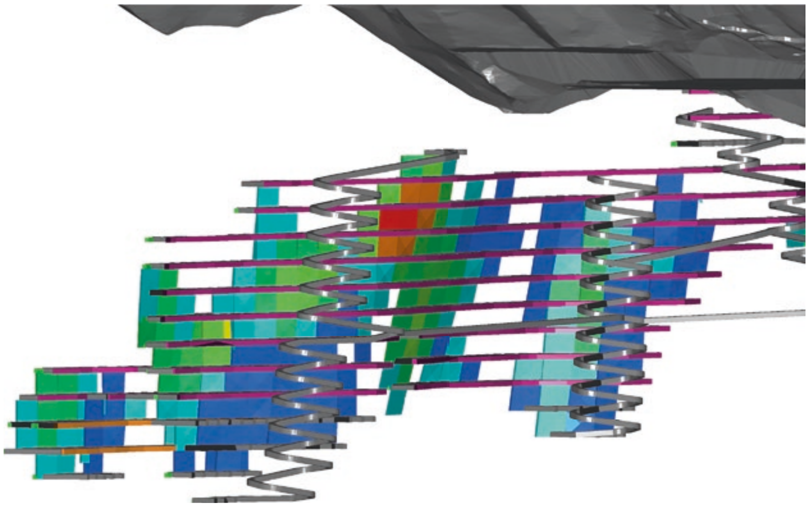

Fig. 5.74 Optimized stopes using Mineable Shape Optimizer (Illustration courtesy of Datamine)

5.5.6 Underground Mining Optimization

As aforementioned in open-pit mining section, effective methods for modeling and optimizing the layout of open-pit mines have been understood in a long time (e.g., Lerchs-Grossman 1965). Although the underground mine design issue is more complex and less restricted than the open-pit problem, it has similar potential for optimization. A meaningful issue in designing a global framework for the optimization of an underground mine is that there is a broad range of mining methods, so that each mineral deposit has a comparatively specializing solution. Thus, there will never be a simple procedure similar to that which is present for open-pit mining. However, by decomposing the design problem into tractable subproblems such as infill drilling design, stope definition, topological network design, and decline design, highly effective though non-globally optimal solutions can be found (Alford et al. 2007).

Infill drilling can be improved through optimization of drillhole pattern and optimization of the physical infill drilling program development. The latter can be managed as a network optimization matter where the aim is to optimize the cost of drilling in combination with the cost of drives and infrastructure to support the drill stations (Brazil et al. 2003). In stope optimization, the different variables can be decreased to dimensional restrictions on the minimum and maximum stope size, suitable stope shape and orientation, and pillar width. In narrow and steeply dipping deposits, the first choice is the width of ore to be extracted. This diminishes the stope optimization issue from a 3-D to a 1-D optimization issue. Mine development network design and its optimization can be approached representing the mine using a weighted network model, which is coordinated according to the coordinates of the mine. Finally, the decline design can also be optimized by using a network model. The ultimate goal of the process is to cover the design of the drilling program, cut-off grade objectives, stope definition, infrastructure development, and mine scheduling in one comprehensive model (Alford et al. 2007).

Regarding mining software for underground mine optimization, there are several options in the market. For instance, Datamine offers several programs for the strategic planning of underground mining operations. For instance, Mineable Shape Optimizer (MSO) generates optimized stope (Fig. 5.74) designs to maximize the value of recovered ore within the given ore body geometry and design constraints, being other tools Mineable Reserves Optimizer (MRO) and Mine Layout Optimizer (MLO).

5.6 Drilling and Blasting

Drilling and blasting are the most cost-effective method to mining mineral resources from the earth. They comprise the first two stages in the production cycle of a mine and the most common method of rock breaking. Most surface mines,

Fig. 5.75 Loading the borehole with explosive

excluding those operations that extract soft rock, require the rock to be fractured using explosives prior to be loaded onto haul trucks. Reliable procedures for rock blasting are well established in mining engineering practice.

Drilling and blasting outcomes cause great impact on different processes of a mine, being essential to find the right combination of drill pattern, explosives, and blast design to contribute to the economic achievement of the global mining process. The primary objectives in rock blasting are the fragmentation of rock masses and moving these rock masses to reduce the mechanical work required. Thus, rock breakage utilizing explosives implicates drilling blastholes, loading the borehole with explosives (Fig. 5.75), and then detonating the explosive in each hole.

Rock breakage is taken into account as the most essential feature of production blasting because of its immediate impacts on the cost of drilling and blasting of the rock and on the economics of loading, hauling, and crushing. In general, the discontinuities in the rock mass, which includes bedding, jointing, and partings, are the main items that dictate how a rock fragments. The closeness of the separation of these determines the maximum block size in the pile of broken material. Consequently, the effect of blasting is to reduce the size distribution of those «preblast» blocks (Lusk and Worsey 2011).

Knowledge of the fragmentation mechanisms is essential to develop accurate techniques for extracting rock quickly. The major elements of the fragmentation process in rock blasting include shock, gas production, extension of fractures, and rock mass movement. When properly initiated, commercial explosives are quickly translated into gases at high temperature and pressure. Following detonation, high-pressure gases compress and beak the material surrounding the explosives. The liberated energy by the explosive can be separated into two principal types, the shock energy and the heave or gas energy. The shock energy causes the conditioning of the rock and initiating mechanisms that originate fractures. As for the gas energy or heave energy, it is generated in the later expansion of the explosives into the crack pattern of the material. Once a fracture network is developed, the gas is able to expand into the network, both spreading the fracture process and causing movement of the rock.

5.6.1 Blasthole Drilling

The hole produced for filling explosives is the so-called blasthole and the procedure of drilling such holes is the so-called blasthole drilling. Most boreholes drilled for mine production are blastholes for explosives. The machine utilized for drilling the hole is called blasthole drill or merely a drill (◘ Fig. 5.76). Blastholes are drilled one after the other, commonly hundreds, then charged and blasted more or less at the same time. The holes are drilled to a depth just below the bench height defined in the planning process of the mine. In order to improve blasting operations, the driller has to measure and log the conditions of all holes. Measure While Drilling (MWD) is an optional instrumentation that logs a number of parameters at requested intervals while drilling such as hole depth, penetration rate, percussion, rotation pressures, and many others. This information obviously provides interesting inputs for the analysis of the rock properties. Utilizing the MWD information, it is probable to define the ideal blasting and obtain a uniform breakage of the rock by adjusting an individual hole charging and blast design. From this, choices can be taken about the most adequate type and quantity of explosive charge to situate in a per blasthole or optimizing the inter-hole timing detonation design of diverse decks and blastholes (Segui and Higgins 2002).

In comparison with the other objectives of drilling such as waterwells or mineral exploration, blasthole drilling shows some peculiarities: (a) the holes are drilled at the same location; (b) blastholes are very near to each other and they are drilled in rock masses that have a high degree of uniformity; (c) they are shallow in depth and drilled in the same environment; (d) no testing is done in blastholes except for grade control (see ► Sect. 5.7); and (e) blastholes are always straight (Gokhale 2011).

For the best overall blasting result, the drillhole needs to follow a designed path along its entire length. While drilling, deviation should be avoided as far as possible. Geological conditions are a major cause of in-hole deviation during drilling, but deviation can also result from faulty setup, hole alignment, as well as bad collaring. The main consequences of hole deviation are (a) uncontrolled fragmentation of blasted material; (b) possible misfires due to intersecting holes firing at undesirable intervals; (c) excessive burden and spacing between adjacent blastholes; (d) secondary breaking leading to higher costs for loading,

◘ **Fig. 5.76** High diameter rotary drills at Aitik mine (Sweden) (Image courtesy of Atlas Copco) haulage, and crushing; and (e) uneven bench floors, resulting in higher equipment maintenance costs. Using positioning lasers, angle indicators, and guide tubes will aid operators to control and manage deviation (Chinedu 2015).

According to the way of rock attack, blasthole drilling is performed by two primary methods: percussive and/or rotary drilling. In percussive and rotary drilling (e.g., top hammer drilling), the rock is broken by a combination of rotation of the bit and high-frequency percussive impacts transmitted by the bit to the rock. These impacts create shock waves that move from the bit to the rock mass through the cutting edges or points on the bit. As a consequence, cracks are created in the material and produce rock chips. The primary difference between rotary drilling and the rest of methods is the lack of percussion, being the tricone bit the preferred to most rotary applications. In rotary drilling, the drill bill is rotated by applying torque at the end of the drill string, which results in removal of chips from the face of the hole. The drill forces the bit into the material mass strongly, being transmitted to the rock mass by means of the cutting points of the drill bit. In both methods, cuttings are extracted from the hole using a circulating fluid bottom-up.

Main factors controlling the choice of drilling method are, most importantly, the continuity of operations, the diameter and depth of the hole, and the features of the rocks to be drilled. Selecting the correct method is essential in mining because the blasthole drilling commonly continues for many years, and blasthole drills especially developed for a method are to be obtained before starting the process.

There are certain limitations to each method of drilling. The effectiveness of top hammer methods decreases quickly as further drill rods are attached to achieve greater depth. In surface mining, holes deeper than 30 m with a top hammer is complex because of the energy lost at the connections of the drill rods. Rotary drilling is still the main technique to drill 230 mm diameter or greater, up to 450 mm actually (◘ Fig. 5.76). Another benefit of this type of drilling is that rotary rigs are big enough to operate a long tower that allows drilling of the complete bench height in a single operation; «at the largest open-pit mines, rotary units are drilling 20 m deep holes in a single pass» (Fox 2012).

Percussive Drilling

The percussive method using a top hammer is mainly utilized to drill hard rock for hole diameters up to 140 mm, being the principal advantage the high penetration rate in sound solid rock conditions. The percussive impact is delivered by either pneumatic or hydraulic pressure. Percussive drills were originally powered by compressed air, but hydraulically powered drills have supplanted pneumatic ones since the mid-1970s (Rostami and Hambley 2011). The advantages of hydraulic drills over pneumatic drills are the fewer moving parts and the significantly higher penetration rates.

In the DTH method, the hammer is located immediately behind the bit and compressed air activates the hammer, which impacts directly to the bit. This eliminates the already commented loss of impact in joints, being a more efficient mechanism of percussive drilling. DTH method is a reliable way to drill in hard to soft rocks and competent to broken or abrasive to nonabrasive rocks. It is also an easy way to produce deep, straight holes with minimum deviation and a very good hole wall stability, even in fissured rocks. DTH is preferentially applied to drilling holes for different objectives on small surface and underground mines. Bits for hammer percussion drills come in various shapes (e.g., chisel or button). Button bits are preferably used in harder rocks, and the shape of the buttons is selected based on the application and the type of rock to be drilled. Underground drilling is usually carried out by using percussion drilling with holes up to 115 mm. According to the underground mining method selected, the holes can be guided in many directions. The holes are usually established horizontally or vertically, being drilled in a symmetrical pattern. Recent significant technological advances in underground drilling include the use of computer-controlled equipment and remote access.

Drilling rigs for underground mining applications can be divided into face drilling and production drilling. Face drilling is performed by mobile rigs equipped with drills mounted on one boom or multiple booms such as two-boom jumbo (◘ Fig. 5.77), which can work on face of the tunnel, roof, side, and floor. The number of booms and drills depends on the opening dimensions and rock mass properties, the number of holes to be drilled per blast round, and the number of faces to be drilled in a shift (Rostami and Hambley 2011). For

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Fig. 5.77 Two-boom drilling rigs performing face drilling (Spain) (Image courtesy of Magnesitas de Rubián, S.A.)

instance, penetration rate in underground blastholes can be considered to vary inversely with the rock strength, other variables being equal. In hard-rock metal mining, two-boom or three-boom jumbos are used, whereas in soft-rock mining, for example, limestone, two-boom or single-boom drill jumbos are common. Multiboom jumbo drills can be programmed to drill the desired blasthole patterns automatically, through coordination with an automated surveying and guidance system. The system simultaneously monitors the drilling parameters and optimizes the control parameters.

Although drill jumbos were historically powered by compressed air, since the late 1970s, electric/hydraulic and diesel/hydraulic units have almost completely supplanted the older pneumatic units. In underground metal mines, ring-drilling production drills are used to drill the long inclined or vertical blastholes used in sublevel stoping, sublevel caving, and vertical crater retreat mining. In such operations, drilling can include not only long production blastholes but also ground support installation such as primarily cable bolts.

Rotary Drilling

It is important to bear in mind that rotary drills can display two methods of drilling, although the majority of the machines work as pure rotary drills, driving tricone, or fixed-type bits. Tricone bits rely on crushing and spalling the rock. This is carried out by transferring downforce to the bit while rotating to drive the teeth, commonly tungsten carbide type, into the rock as the three cones rotate around their respective axis producing cuttings of rock. The softer the rock, the higher the rotation speed. Most drilling functions are hydraulically driven.

Once the cuttings of the rock are created, they must be evacuated, commonly with compressed air. If the cuttings are not removed from the hole, the bit will be eroded because of the abrasiveness of the rock chips. In most rotary blasthole drills, cuttings are lifted between the wall of the hole and the drill rods by compressed air. The compressed air is also needed to dissipate the heat mainly originated by friction between cones and the rock. Two types of drilling bits, drag bits and tricone bits (Fig. 5.78), are utilized, but since their intended direction introduction in the early 1900s, tricone bits have been the traditional type of bits used in rotary drilling. They remain the most popular bits for blastholes ranging from 150 to 450 mm in diameter. Bit selection is based on hole size and depth, rock type, and operational requirements.

Most rotary drills are diesel powered for good mobility. The most important advance in drilling equipment since 1990 has been the development of computer-controlled drilling systems. These systems automatically locate and collar the holes based on a preprogrammed blast round design and incorporated real-time monitoring and optimization of the drilling. To ensure that the blasthole is exactly located and is drilled to the right depth, GPS hole navigation has been developed. This navigation system utilizes antennas mounted on the tower rest and radio antennas on the cab to produce a correct bit position.

Fig. 5.78 Tricone bit (Image courtesy of Atlas Copco)

5.6.2 Explosives

An explosion is a physical or chemical process in which energy is liberated in just a short time. It is commonly accompanied by formation of a great amount of hot gas. There are many types of explosions: mechanical, nuclear, electrical, or chemical. In this section, only chemical explosions, originated by decomposition or very quick reaction of a substance or a mixture, are considered. Thus, an explosive is a substance or mixture of substances which, when started using heat, impact, friction, or shock, undergo rapid chemical transformation, releasing tremendous amounts of energy in the form of heat, gases at high temperature, and shock. The energy released by an explosive is used in mining to rock fragmentation and displacement. The majority of explosives used in modern mines are manufactured using fuels, oxidizings, sensitizers, energizers, and subordinate substances behaving as stabilizers, thickeners, or flame retarders. Regarding the two main components, an oxidizing is a chemical which provides oxygen for the reaction whereas a fuel is the component that reacts with oxygen to produce heat.

For a chemical to be considered an explosive, it must produce quick expansion and liberation of heat, fast reaction, and request to initiate the chain of reactions. To understand the different concepts, the distinction between deflagration and detonation is required. These are the two distinct types of rate of reaction. Detonation takes place when the rate of reaction in the explosive product clearly exceeds the speed of sound creating a shock wave. The speed of detonation for commercial explosives ranges from 1500 to 9000 m/s, which is much higher than the speed of sound (Table 5.3). On the contrary, deflagration is a process where the reaction takes place at much lower rates than the speed of sound, so that no shock wave is produced in the explosive material. Deflagrating explosives include black power, which burn relatively slowly and generate comparatively low blasthole pressure, whereas detonating explosives such as penthrite are characterized by superacoustic reaction rate and comparatively high blasthole pressure.

The speed of propagation is based on the intensity of heat with which the procedure initiated and how quickly and how much oxygen the burning process needs. Thus, explosives can be classified as high explosives (e.g., nitroglycerin) and low explosives (e.g., black powder): high explosives detonate and require a detonator, while low explosives deflagrate and do not require a detonator. Low explosives cause heavy push or powerful lift of the surrounding material but do not cause a shatter. High explosive substances decompose very quickly through detonation under particular situations to develop a large volume of gases, extraordinarily high amount of heat, and quickly translating shock waves in atmospheric gases. When the explosives detonate, they mainly originate common and harmless chemical compounds such as water, carbon dioxide, and

Table 5.3 Velocities of detonation of some explosives

Explosive	VOD (m/s)	Explosive	VOD (m/s)
Lead azide	4630	Nitroglycerin	7700
Mercury fulminate	4250	Dynamite (65% gelatine)	6500
Picric acid	7350	Ammonium picrate	7150
Trinitrotoluene (TNT)	6900	Black powder	400
PETN	8400	Lead styphnate	5200
RDX	8750	Nitrocellulose	4492
HMX	9100	Nitroglycol	8250

5

nitrogen with subordinate harmful gases such as nitrous oxide, carbon monoxide, ammonia, and methane. This is because the majority of explosives are chemicals constituted by carbon, hydrogen, oxygen, and nitrogen. It is important to note that nitrogen is presumably the most essential component of a chemical to achieve the explosive nature.

Explosives can also be classified according to their sensitivity as primary, secondary, and tertiary. Sensitivity of an energetic material can be seen as the quantity of power that the material requires to absorb to achieve a specific probability of making an explosive reaction (Matyas and Pachman 2013). Thus, the most sensitive energetic substances are primary explosives, less sensitive are secondary explosives, and very insensitive are tertiary explosives. Primary explosives (e.g., mercury fulminate, lead styphnate, and lead azide) can be specified as materials that respond to stimuli-like shock, impact, friction, flame, etc., and pass from the state of deflagration, at high rate of burning, to detonation. They are also called initiating explosives, being used in the manufacturing of detonators, detonating fuses, and boosters. Secondary explosives are relatively insensitive to heat, shock, or friction, and they are also termed base explosives. The most typical example of secondary explosives is pentaerythritol tetranitrate (PETN or penthrite). Secondary explosives have a high rate of detonation and commonly require a small device including small amount of primary explosive for their detonation. These substances are utilized in the production of detonators and constitute their base charge.

Tertiary explosives, also called blasting agents, are very insensitive to shocks, and they cannot be reliably detonated by a limited amount of primary explosive. Therefore, the detonation device contains a small quantity of secondary explosives. Tertiary explosives are commonly used in mining and construction operations. The most commonly utilized tertiary explosive is ANFO, acronym of ammonium nitrate and fuel oil. Near 80% of mining blasts are carried out utilizing this explosive. Besides the above classifications, explosives are also ranked based on other parameters such as their consistency, packaging (cartridge versus bulk), or their chemical nature. The latter include two groups, those classed as substances that are explosive and those that are explosive mixtures, for instance, black powder.

Properties of Explosives

Properties of the explosives are relevant because they are the ultimate reasons for their choice. Obviously, the ingredients of the explosives influence directly on many of their properties such as resistance to water, detonation speed, or cost. The utility of an explosive can only be defined where the properties are completely understood. The properties of explosives are summarized in Table 5.4 (Gokhale 2011), being some of them briefly described below.

Velocity of Detonation

Velocity of detonation is a measure of the speed at which the detonation front moves, for example, along an explosive column. It is the most important property of an explosive. Two explosives with same strength but different velocity of detonation can perform quite differently in a blast. The velocity of detonation depends on components of the

Table 5.4 Properties of explosives

Explosive property	Meaning
Velocity of detonation	Velocity in m/s at which the shock front of the detonation layer travels within the column of explosive.
Detonation pressure	Pressure developed by detonation of the explosive in the detonation zone. It is usually measured in GPa.
Blasthole pressure	Pressure exerted on the wall of the blasthole immediately after the detonation.
Strength	Total amount of energy released by the explosion in MJ for each kg of explosive. This includes the energy released in the form of heat as well as the pressure exerted by the gases generated in detonation.
Heat of explosion	Total amount of heat released by the detonation in kcal for each kg of explosive.
Specific gas volume	Amount of gas generated by detonation of one kg of explosive under normal conditions.
Sensitivity	Possibility of causing detonation by such means as friction, pressure, heat etc.
Transport and handling safety	How easily the explosive can be handled and transported through different modes of transport.
Brisance value	Brisance value indicates the shattering effect of the explosive.
Charging density	Weight of the explosive in kg, contained in each liter volume of the blasthole.
Toxic fumes	Volume of poisonous gases generated in terms of liters per kg of explosive detonated.
Water resistance	Whether the properties of explosive remain unchanged by mixing the explosive with water.
Hygroscopicity	Hygroscopicity is a measure of water-absorbing capacity of an explosive.
Minimum hole diameter	What is the smallest diameter of blasthole in which the explosive can be charged and detonated to get the desired explosion effect.
Storage life	How long the explosive can be stored in originally packed and unpacked condition without a change in it's properties.
Volatility	How much is the volatility of the explosive.
Material coexistence	This is the ability of the explosive to coexist with other materials.

explosive, density accomplished when the blasthole is charged, blasthole diameter, type of confinement, the presence of cavities in the rock mass, rock mass temperature, and temperature originated at the initiation element of the detonators that are utilized for firing the explosive. In general, the higher the velocity of detonation, the better will be the shattering effect and rock fragmentation process. Some military explosives have velocities of detonation reaching up to 10,000 m/s or more, but velocities of detonation of explosives used in mining rock blasting range from 2000 to 7000 m/s.

Strength

Strength of an explosive means the energy released by unit weight (weight strength) or unit volume (bulk strength) of the explosive. The energy of an explosive shows the ability of the explosive to do work. The strength is commonly well correlated to density and detonating velocity as well as heat and gas volume released in the detonation of the explosive. The global energy liberated by an explosion can be separated into two main components, shock energy and bubble energy. The former is generated by the shock wave, which moves from the place of its origination as a strain wave, and the latter is produced by the heat developed by the chemical reactions included in the detonation process.

It is complex to estimate the strength of explosives in terms of absolute units. Several tests allow the effect of the strength of an explosive to be monitored easily, offering indications of the strength of an explosive with respect to the strength of a common explosive, which is considered as 100 (nowadays taking ANFO as standard). Thus, strength expressed in terms of such an indicator is called relative strength. High strength is required to shatter the hard rocks, but the utilization of high-strength substances in soft and fractured rocks will be wastage of the excessive energy produced by this explosive.

Density

Density of the majority of commercial explosives is in the range of 0.5–1.8 g/cm^3. A dense explosive liberates more energy per volume unit because increasing density leads to an increase in velocity of detonation and detonation pressure. Thus, dense explosives are very useful to break hard rocks. Primary explosives are usually manufactured as crystalline or powdery material with low densities and large specific surface. Where higher pressures are used to achieve higher densities – the compaction process is reflected in the density of the explosive – «a phenomenon called 'dead-pressing' can occur, leading to a material which is hard to ignite and, if ignited, only burns without detonation» (Matyas and Pachman 2013). Therefore, pressing a primary explosive to a point where it loses its capability to detonate is not desirable. For example, the optimum density range for ammonium nitrate is between 0.8 and 1.0 g/cm^3. In general, it is desirable to press explosives to densities as close to the critical density as possible. Explosives are supplied by the manufacturers in different densities to control the total energy released in a blasthole.

Sensitivity

Sensitivity of an explosive is an estimation of the ease with which it can be detonated. Since explosives utilized in older days (e.g., nitroglycerin) were very sensitive and exploded without reason, today explosives utilized for rock blasting are far less sensitive and have become far safer. Naturally, a perfect explosive to be utilized in rock blasting should be very insensitive so it does not detonate in all the storing or transportation processes. Nevertheless, if the sensitivity of the explosive is too low, the detonation within a blasthole can be interrupted if there are gaps or obstacles among the charges.

Water Resistance

Water reduces the effectiveness of an explosive largely since one or more ingredients of the explosive can be dissolved in water and becomes ineffective. «In low temperature regions, the water can cool a water-resistant explosive to such a low temperature that a much higher detonation energy is required to ensure its detonation» (Gokhale 2011). The water resistance of explosives varies considerably. According to some results of tests performed on samples, the manufacturers define water resistance of the explosive as «excellent, very good, good, limited, or poor.» The water resistance of an explosive is essential because the blasting process can often take place in wet conditions. As an example, emulsions have excellent water resistance, heavy ANFO has some water resistance, and ANFO has poor or negligible water resistance.

Types of Industrial Explosives

Industrial (or commercial) explosives are designed, produced, and utilized for commercial applications rather than for military purposes. The principal explosives utilized in mining are generally multicomponent type, containing fuel, oxidizer, and in many cases a sensitizer as well. Fuel is used to burn and generate heat. The oxidizer accelerates the process of burning, and where high-energy output is required, a sensitizer is supplemented to the explosive mixture. From an industrial viewpoint, four main groups of explosives used in mining blasts can be considered: (1) dynamites, (2) blasting agents, (3) slurries and water gels, and (4) emulsions. According to the Federation of European Explosives manufacturers, the US and Europe explosive consumption in 2014 was about 3,600,000 tons, with ANFO products representing the more consumed group in the market.

Dynamites

The original dynamite made by Alfred Nobel was a mixture of nitroglycerine and kieselguhr (diatomaceous earth). The kieselguhr absorbed the oily nitroglycerine and the mixture became quite insensitive to shock. Thus, it could be used far more safely than nitroglycerine. Over the years, formulations of dynamite have changed, but nitroglycerine has still remained the main detonating component. There are three basic types of dynamites according to their consistency: granular/powdery, gelatine, and semi-gelatine. They are offered in cylindrical paper, cardboard, or plastic cartridges.

Gelatine dynamites are powerful explosives (nitroglycerin, 92%, and/or nitroglycol, ammonium nitrate, and nitrocellulose, 8%) with a detonation velocity ranging from 4300 to 7500 m/s and generating high shattering capability. Sensitivity to initiation by cap or detonating cord is very good, and density, water resistance, and detonation pressure are high. Gelatine dynamites (Fig. 5.79) can be utilized as the principal explosive component where high density and energy are needed or as a primer for ANFO.

Fig. 5.79 Cartridges of gelatine dynamite (Image courtesy of Octavio de Lera)

ANFO

Dry blasting agent is a term used for components of an explosive that they themselves are not defined as explosives, but when mixed together they constituted a mixture that can explode. A dry blasting agent is a granular mixture of solid oxidizer, commonly ammonium nitrate, into which a liquid fuel or propellant is absorbed. Thus, the ammonium nitrate serves as the oxidizer and the fuel oil as the fuel. In surface mines, the most commonly used dry blasting agent is ANFO (Box 5.13: ANFO).

Box 5.13

ANFO

ANFO (ammonium nitrate and fuel oil) is formed by mixing ammonium nitrate (94%) with fuel oil (6%), a mix that gives maximum energy and velocity of detonation (around 3660 m/s). Even after mixing these two components, the final product remains fairly dry since the percentage of fuel oil in the mix is very small, and it is absorbed in the pores of the small granules (prills) of ammonium nitrate. Because of their insensitivity, ANFO cannot be detonated by a detonator, and it should be detonated by a primer of high explosive (e.g., one or two cartridges of dynamite with detonator).

ANFO is supplied basically as poured or packaged. Where the quantity of explosives is high (e.g., large open-pit mines), ANFO is supplied in separate component containers on a truck, mechanically mixed at the worksite, and poured into blastholes. Hence, it is also called bulk ANFO. Where the requirement of the mine is low, for example, in small quarries, ANFO is usually supplied in nylon bags. Figure 5.80 shows the ANFO loading operation using nylon bags. Poured ANFO proves more effective than the packaged form as it fills the entire cross section of the blasthole, whereas the package leaves a gap between the walls of the blasthole and external diameter of the package (Gokhale 2011). If a blasthole has a significant quantity of groundwater seeping into it, it cannot be charged by poured ANFO. In such cases, ANFO is premixed and packed into thick cylindrical plastic bags sealed at both the ends. Charging a blasthole with packaged ANFO is very tedious and time-consuming.

Advantages of ANFO are their safety, ease of loading, and low price. In the free-flowing form, they have a great advantage over cartridge explosives because they completely fill the borehole.

Regarding disadvantages, there are two disturbing aspects about the use of ANFO in large surface mines. The first is the quick evaporation of diesel oil; the second is the high solubility of ammonium nitrate in water. Where atmospheric temperature and humidity are high, it becomes essential to add extra fuel in the ANFO mix to take care of the degree of evaporation likely up to the time of detonation. In respect of solubility in water, if humidity of the atmosphere is high, the ammonium nitrate, being highly hygroscopic, absorbs a large quantity of water and becomes less effective in the process. In this context, it is worth noting that after about 9% water content, the ANFO mix becomes insensitive and fails to detonate. If ANFO mix contains aluminum as a sensitizer, such mixture is called ALANFO. It is particularly useful for blasting hard-rock masses, being 10–15% the most commonly used percentage of aluminum in ALANFO.

Fig. 5.80 Loading operation of ANFO nylon bags (Image courtesy of Octavio de Lera)

Slurries and Water Gels

The explosives that include more than 5% water by weight are called wet blasting agents. Slurry explosives, water gels, and emulsions fall within this category. Slurries are made from ammonium nitrate partly in aqueous solution. They are fluid, pumpable, and miscible with water. These types of explosives were invented to avoid ANFO explosives issues such as no water resistance, low density, and limited energy options. Thus, these substances are waterproof and are commonly the preferred selection in an environment where the blastholes stay wet. Slurry explosives are forthcoming in highly viscous paste (bulk slurry) as well as in cartridge form. They cost more than other commercial explosives such as ANFO. Bulk slurries can be pumped into a blasthole through tubes connected to a truck (Fig. 5.75). Slurry density ranges commonly from 1.10 to 1.25 g/cm^3.

Water gel explosives, a special form of slurry explosive, include meaningful quantity of water and split oxidizer and fuel elements, generating a mixture that is less sensitive than water-free nitroglycerin dynamites. Water gels are made up of oxidizing salts (e.g., ammonium nitrate, calcium nitrate, or sodium nitrate) and fuels (e.g., ethylene glycol, aluminum, or oil) dispersed in a continuous liquid phase. Physical sensitizers such as air, plastic, or glass bubbles can be also mixed with the gel. The density of most water gels ranges from 1.0 to 1.35 g/cm^3. In the last years, water gels and emulsions have almost completely replaced dynamite.

Emulsions

Emulsions (Fig. 5.81) are explosive materials that contain substantial amount of oxidizers dissolved in water droplets surrounded by a fuel that is unable of blending or mixing. The ratio of oxidizer to fuel in an emulsion is typically 9:1. An emulsifying agent (e.g., sodium oleate) stabilizes the water-in-oil emulsion against liquid separation. Dispersed

Fig. 5.81 Emulsion explosive

gas can be included into the emulsion matrix for density control ranging from 0.68 to 1.36 g/cm³. Thus, voids in the form of microballs or by chemical gassing of the composition make the emulsion more sensitive. Emulsion explosives show excellent water resistance, are moderately insensitive to temperature changes, have high energy, and develop very good efficiency and flexibility of use. The performance of the emulsion explosives makes them as superior products compared to the available slurry-based explosives.

Explosive Initiating Systems

In rock blasting, many additional items are required besides the main explosive. These components are commonly called accessories. The main items in a blast are a booster cartridge, a primer cartridge, initiation transmission line (ITL), and detonators. Of these, the primer and booster are used to amplify the energy released by the detonation of the detonator. The explosive cartridges are mainly formed by pentolite, although other explosives such as dynamites, water gel/slurries, or emulsions are also utilized in primer or booster cartridges. Since these types of explosives have been already explained, this section is devoted to aspects related to detonators, including blasting instruments such as testing or initiating instruments.

Fig. 5.82 Detonating cord (Image courtesy of Octavio de Lera)

Initiation Transmission Line

An initiating device located at a very great distance for the sake of safety always detonates a primer cartridge. It is therefore crucial to transmit the pulse through a line called «the initiation transmission line» (ITL). The more common used ITL in blasting is probably the detonating cord (Fig. 5.82), which transmits a detonation wave. It is made up of a plastic tube with 3–5 mm outside diameter and being usually filled with penthrite (10–15 g/m). Thus, the velocity of detonation is about 6500 m/s with very high

Fig. 5.83 Signal or shock tubes

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shock energy. Since ANFO needs a great initiating effect throughout its charge column, detonating cord can fulfill this requirement perfectly.

Some similar device is igniter cords, which are cord-like in appearance. Last option of initiation is a signal or shock tube (Fig. 5.83), which transmits a signal from the detonating cord to the delay detonator in the hole. Signal tube can be initiated by an electric detonator and transmits a low energy signal at ±2000 m/s from one point to another. This initiation system is not violent compared to detonating cord and is hence much safer to use. It is also recommended in zones where electric detonators are not desirable to be used.

Detonators

A detonator is commonly referred to as an initiating device since it begins the detonation procedure in a blasthole. They are compact devices that are manufactured to safely initiate and control the efficiency of larger explosive charges (Box 5.14: Detonators).

Box 5.14

Detonators

A detonator consists of a metal tube (Fig. 5.84), usually 5.5–7.5 mm in outer diameter and variable length depending upon whether it is instantaneous or delay type. It incorporates a primer explosive (e.g., lead azide) and a secondary explosive such as penthrite or pentolite. These explosives can be initiated by electrical or shock energy from an external source. With such sensitive explosives, detonators become sensitive and are more prone to accidental detonation. These characteristics make them the most dangerous explosive products in industrial applications. Thus, they must be stored, transported, handled, and used according to set procedures. There are three types of detonators based on the source of energy used for starting detonation in the detonator: electric, non-electric, and electronic. In turn, they can be instantaneous or with a delay element built into them. The delay element is in the form of a small tube filled with densely packed pyrotechnic material. Commonly used delays are either from short delay series or long delay series.

Electric Detonators

Electric detonators cause the initiation of detonation by an electric current passed through the detonators by electric wires. They have an outer aluminum, copper, or steel shell that contains primary and secondary explosives, insulation material, two wires, and a delay element if applicable. The current heats up a high resistance wire that ignites a fusehead, similar to a match. The resulting flash ignites a delay element, which, in turn, burns the primer charge that detonates the base charge or secondary explosive. The simplest and better way to connect electric detonators is in series because if one or more detonator connections are faulty, then the entire circuit will not fire. This eliminates the possibility of having explosive in the broken rock

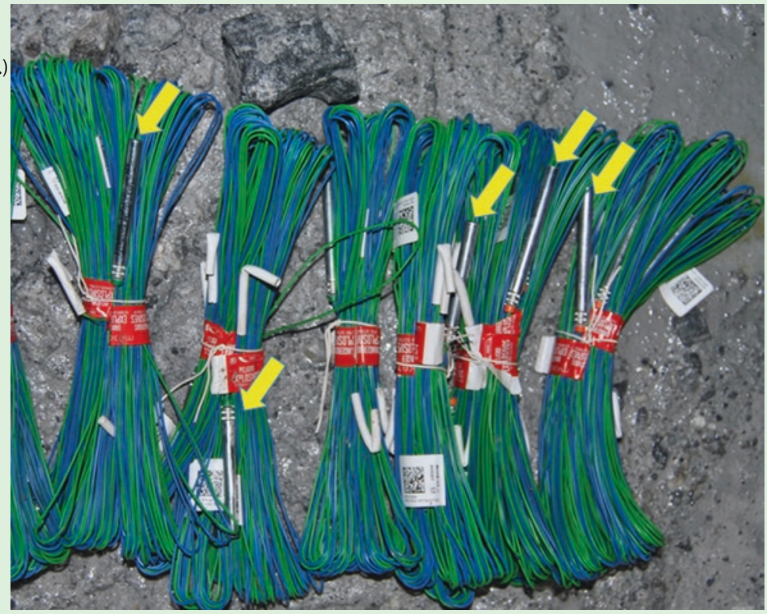

Fig. 5.84 Detonators (Image courtesy of Magnesitas de Rubián, S.A.)

after blasting. In a parallel circuit, each detonator is independent of the others. Moreover, connection in series allows the entire circuit to be tested for continuity and resistance.

The electric delay detonators are manufactured as two varieties, long/half-second delay detonators and short/millisecond delay detonators. Long delay or period detonators are available in several numbers, with a nominal half-second time interval between each delay. Short delays detonators present delay intervals much shorter, varying from 8 to 100 or more milliseconds. Anyway, delays available can differ from manufacturer to manufacturer.

Non-electric Detonators

Non-electric detonators are fired by detonating cord instead of electricity. A non-electric detonator consists of a plastic shell filled with primary explosive, secondary explosive, and a delay if applicable and a certain length of detonating cord. This system is frequently used for blasting a large number of holes. It is able of introducing delays of millisecond intervals between holes or rows of holes. The delay intervals also change depending on the manufacturer but always in milliseconds (e.g., between 15 and 700 ms or between 75 and 1000 ms). The use of these delays can produce advantages such as easy and safe to handle, better fragmentation, and reduced ground vibration. The system finds its applications in surface and underground metalliferous mines.

Electronic Detonators

The difference between electronic and electric detonators is the replacement of the pyrotechnic delay element by a microchip. Most electronic detonators consist of wires, a detonator shell that looks similar to electric and non-electric detonators, a microchip, a capacitor, and a primer charge/base charge similar to electric and non-electric detonators. At firing time, the blasting machine sends out a code to initiate the electronic timing devices within the detonators. Since electronic detonators utilize microchip technology to provide delays for blast designs, it allows for much greater accuracy in firing times. Thus, the negligible variation in the electronic delays means that the firing pattern will consistently be the same for each blast, resulting in uniform blast results (Banda and Rhodes 2005).

Each detonator has its own time reference, but the final delay time is determined through the interaction between the detonators and the computerized blasting machine before their firing. Shortest delay time is 1 ms, but detonators are extremely precise to the extent of 0.2 ms. This electronic initiation system is considered the safest among all the initiating systems. It can be tested in the field without causing actual detonation. Electronic detonators cannot be initiated by a conventional blasting unit nor can they be activated without entering proper security codes. However, electronic detonators are still susceptible to initiation by lightning, fire, and impact of sufficient strength. Therefore, as all other detonator types, they must be properly transported, stored, and handled as an explosive.

Blasting Instruments

Blasting instruments can be broadly classified into two main groups: testing instruments and initiating instruments. Regarding the testing instruments, every circuit must be completely checked prior initiating the blast. Besides this, the area of blasting must also be surveyed for extraneous current if electric detonators are utilized. For this purpose, two very commonly instruments are used: blaster's multimeter and blaster's ohmmeter. The first is utilized to measure voltage, resistance, and current in various parts of the blasting circuit. The second is sometimes preferred because it is more accurate than multimeters for measure of resistance, for instance, to ensure that the electric detonator circuit has continuity.

Initiating instruments produce an action that leads to detonation of the detonator in the main explosive. An initiating device naturally depends upon the ITL used for the blasting circuit. There are many different types of initiating devices. Initiators used for initiating the detonation of electric detonators, or exploders (◘ Fig. 5.85), have the capability of imparting electric current to the blast circuit. Another initiating device is the detonation wave initiator, in which a detonation cord transmits a detonation wave. Finally, electronic blast initiator is utilized to compose a computer program listing that controls the complete blast, including detonation sequence, delay intervals, etc. It includes also protection to overvoltages, electrostatic discharge, and unauthorized use as the detonator requires a specific coded signal to fire.

◘ **Fig. 5.85** Exploder (Image courtesy of Pedro Rodríguez)

5.6.3 Surface Blasting

Blast design is the most crucial step in drilling and blasting. First and foremost, blast design is an iterative process where important factors such as the required fragmentation, production, and muck pile shape are used as a starting point for determining optimal drillhole diameter, depth and inclination, subdrilling, explosive type, and detonation timing. Moreover, operating costs of both the mine and the processing plant are directly related to the fragmentation achieved during blasting (Bhandari 1997). The aim of a good blast design is to spread the explosives throughout the rock mass such that the rock breakage generates the desired result, the rock blasted is easily mobile by the excavation equipment, and the procedure originates minimal adverse environmental effects (e.g., flyrock, high air blast, and ground vibrations). Distribution here is considered a combination of blast pattern and explosive density. Blast modeling programs and other tools such as high-speed photography or computer software to calculate fragmentation distribution have significantly aided engineers in accurately simulating and analyzing different blast designs.

The most common blasting method in surface mining is bench blast, being the bench height the starting point for blast design (◘ Fig. 5.86). In bench blasting, parallel holes are blasted in each round in large numbers. It is of huge importance to have a proper delay between each row and even between individual holes in each row. The bottom charge from where the initiation normally starts requires well-packed explosives of higher blasting power than is needed in the column charge (a charge of explosives in a blasthole in the form of a long continuous unbroken column). Stemming materials are used to top off the blastholes to provide confinement.

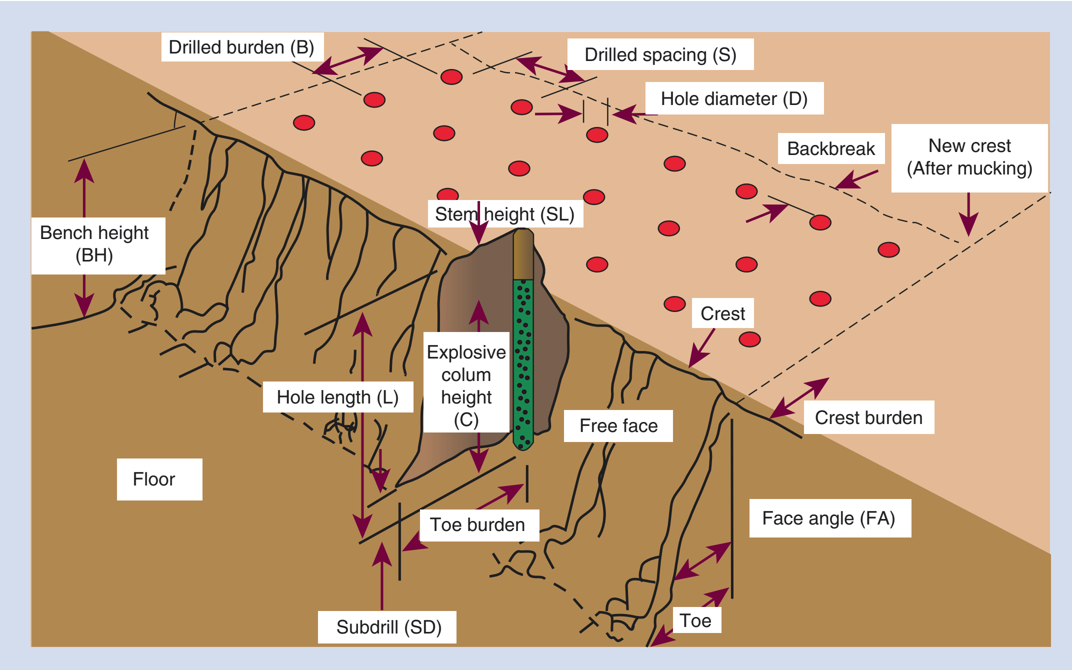

Fig. 5.86 Main blast pattern parameters in surface bench blasting

Blasthole Diameter

It corresponds to the cross-sectional width of the borehole (Fig. 5.86). The blasthole diameter is generally chosen in accordance to the depth of the excavation: shallow excavations commonly utilize smaller diameter holes than deeper operations. The selection of the hole diameter in the blast design is based on the geology of the blast site, primarily the jointing and bedding of the formation, which is the only factor that cannot be changed. The desired fragmentation and economics must also be considered. Large-diameter blastholes are less suitable in strong, massive rock where minimal broken rock movement is required or where it is essential to monitor blast vibrations. Larger blasthole diameter commonly reduces costs for drilling, primers, and initiators. However, smaller blasthole diameter gives better distribution of energy in the rock mass. Since blasthole diameter is directly related to bench height, a good rule of thumb is that bench height in meters is equal to blasthole diameter in millimeters divided by 15. In surface mining practice, the rate of drilling and rate of removal of the blasted rock must match; thus, the diameter of blastholes is loosely related to the capacity of the shovel bucket (Gokhale 2011) as matched in Table 5.5.

Table 5.5 Blasthole diameters based on shovel bucket capacities

Bucket capacity of the shovel (m^3)	Hole diameter range in mm
4.5	76–127
7.5	127–215
9.17	171–250
11.5	200–270
15.3	229–311
20	250–349
35	270–381
50	311–445

Free Faces

A free face is a rock surface exposed to air that generates room for expansion upon fragmentation. It is sometimes also termed open face. Forward displacement of blasted rock takes place if a blast shoots to a free face (Fig. 5.87). Free faces are necessary because some movement of

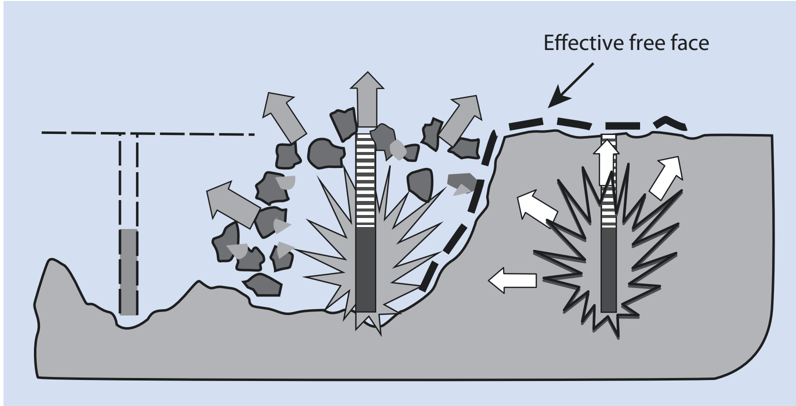

Fig. 5.87 Effective free face (Illustration courtesy of Atlas Copco)

the rock mass is crucial to enable for crack propagation. Moreover, increased movement assists crack propagation and improves fragmentation. In some cases, free faces can be limited to avoid dilution in the mineralization.

Blasthole Angle

Vertical blastholes are commonly utilized in surface open-pit metal mines since drilling accuracy is greater and angled blastholes are more difficult to set up and drill. Some drills even do not have an angled drilling capability. In free-face blasting, vertical front row blastholes often leave variable and excessive burdens between the top and bottom of the charge, causing hard and immovable toe; toe in bench blasting is the excessive burden measured at the floor level of the bench. It is common to prevent adequate breakage and movement of the toe using some angled blastholes in front rows. However, excessive blasthole angles can cause problems (Fig. 5.88).

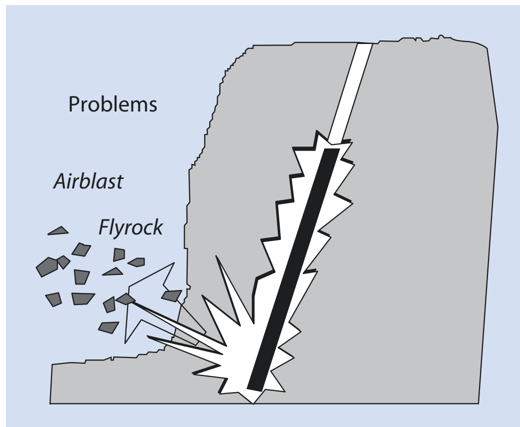

Fig. 5.88 Excessive blasthole angles cause problems (Illustration courtesy of Atlas Copco)

Spacing and Burden

Spacing and burden are related to blasthole diameter, depth, rock type, and charge length. Spacing is the distance between adjacent boreholes in a row (Fig. 5.86). In bench blasting, the distance is measured parallel to the free face and perpendicular to the burden, being burden the distance from the borehole and the nearest free face or the distance between boreholes measured perpendicular to the spacing (Fig. 5.86). Spacing can be somewhat dependent on the timing but is most often a function of the burden. The presumption of from 1 to 2 times the burden is a correct starting point for establishing the spacing of a blast to be initiated simultaneously in holes in the same row. With respect to burden, the proper burden dimension to utilize in any given individual blast can be calculated by taking into account hole diameter, relative rock density (Table 5.6, Gokhale 2011), and the explosive that will be incorporated in the blast. A burden too small can result in excessive air blast and flyrock. On the contrary, a burden too large can result in improper fragmentation, toe problems, and excessive ground vibrations. The assumption of 25–35 times the hole diameter can be a good approximation for establishing the burden dimension.

Subdrilling and Decking

Subdrilling (Fig. 5.86) is the procedure of drilling boreholes below floor level to assure breakage of rock to working elevation. Subdrilling is also the

Table 5.6 Dependence of burden on rock density and type of explosive

	Values of burden in terms of blasthole diameter D for rocks of different densities		
Type of explosive	**Low 2200 kg/m³**	**Medium 2700 kg/m³**	**High 3200 kg/m³**
ANFO	28 × D	25 × D	23 × D
Slurry dynamite	33 × D	30 × D	27 × D

Fig. 5.89 Stemming of a borehole on top

length of the explosive charge that lies beneath the designed bench floor level. Some operations range from 0.2 to 0.5 times the burden or 5–8 times the diameter of the hole. It is good practice to drill always a certain extra distance, especially in blasting massive rocks where there is no adequate horizontal bedding plane to maintain floor grade. The subdrill part is usually backfilled with drill cuttings or other stemming material. For its part, decking is the separation of the explosives column in a blasthole into two or more parts with stemming between them. This procedure is commonly utilized to decrease either the charge load per hole, the quantity of explosives detonated per delay or both. It should be 6 times the hole diameter for dry holes and 12 times the hole diameter for wet holes.

Stemming

Stemming is the inert material located in a borehole on top (Fig. 5.89) of or between separate charges of explosive material. It is utilized for confining explosives or to separate charges of explosives in the same borehole. Stemming improves fragmentation and rock displacement by reducing premature venting of high-pressure explosion gases to the atmosphere. Dry granular materials such as sized crushed stone or drill cuttings are used for stemming. Appropriate stemming chip size lies in the range of 10% of the blasthole diameter. Inadequate stemming creates more flyrock, surface overbreak, noise, and air blast (Fig. 5.90). Optimum stemming length depends mainly on blasthole diameter, stemming material, and surrounding rock properties. Stemming column is usually 0.5–1.3 times the burden, being a correct approach for its height the value of 0.7 times the burden. As a summary of all these concepts and associated values, three examples of real data are shown in Table 5.7.

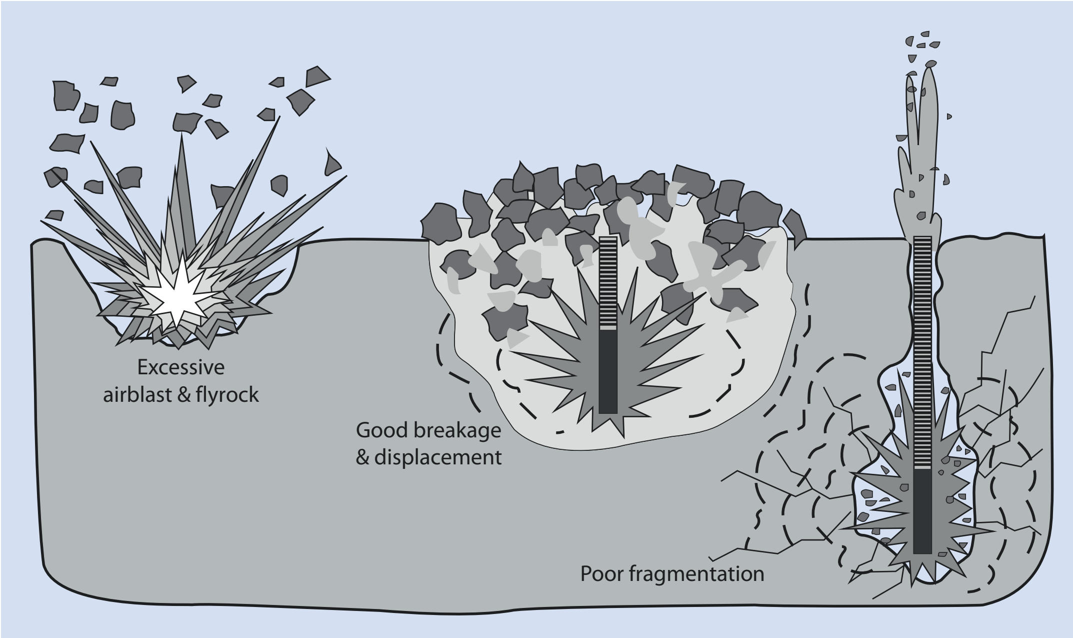

Fig. 5.90 Effect of correct and incorrect stemming (Illustration courtesy of Atlas Copco)

Table 5.7 Some real blast pattern data: A, iron ore formation; B, sulfide rock; C, copper-gold porphyry ore; D, saprolite waste; E, granodiorite waste

Parameter	A	B	C	D	E
Bench height (m)	15	10	30	15	10
Blasthole diameter (mm)	381	102	311	311	311
Burden (m)	7.9	5	9	8.4	8
Spacing (m)	7.9	5	9	9.7	10
Subdrilling (m)	1.8	1.2	–	2	2.5
Stemming (m)	5	3	6	–	6
Explosive density (g/cm^3)	1.28	0.85	–	–	–

Blasthole Pattern

If the hole diameter and the explosive type have been elected, the next step is to establish the configuration of the holes, the so-called blasthole pattern. To distribute the explosives throughout the rock mass, holes are drilled in the rock in a grid-like pattern (Fig. 5.8). As a rule, blasthole patterns depend on blasthole diameter, rock properties, explosive properties, bench height, and the results needed. Each blasthole is intended to break the rock beside the hole, toward the open excavation. If the holes are too far apart, the energy will not be sufficient to originate the desired breakage. If the holes are too close together, energy will be lost. Another important part of the blasthole pattern is the number of holes. It is largely dependent upon how much material requires to be extracted by each blast.

Different patterns can be chosen, including square, rectangular, or staggered (Fig. 5.91). Blast modeling results have shown that, in massive rocks, better rock breakage and productivity

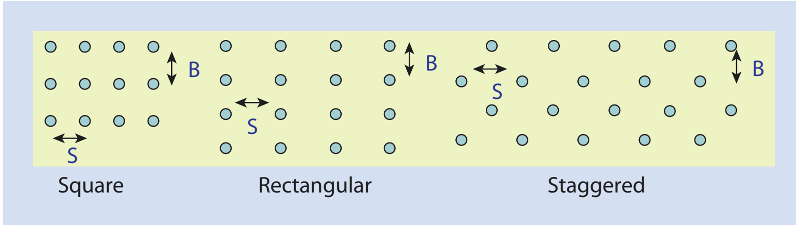

Fig. 5.91 Square, rectangular, and staggered blasthole patterns

are generated using staggered patterns than with either square or rectangular patterns. This is «because the hole in the row behind is blasting into more solid rock rather than a weakened pocket, and because holes tend to break at 45° to the free face, making a square end to a bench almost impossible» (Lusk and Worsey 2011).

Delay Times and Blast Timing

Delay blasting is the method of initiating explosive boreholes or rows of boreholes at predefined time intervals utilizing mainly delay detonators. The sequence in which blastholes are initiated and the time interval between successive detonations play a major role on global blast efficiency. This enables the blastholes closest to the open excavation to detonate and translate rock into the open space first. The blastholes behind the first holes then can translate rock horizontally into the new open space. Thus, the burden on each blasthole requires time to move after the detonation to generate an effective free face. Dependent blastholes then fire toward this new free face developed during the blast. Therefore, the first consideration to establish delay intervals is the availability of free faces. The efficiency of production blasts can only be optimized where charges detonate in a controlled sequence at suitable discrete but closely spaced, time intervals.

There are two main types of delay in a blast pattern. These are the hole-to-hole delay and the row-to-row delay. The optimum hole-to-hole delay is 4–5 ms per meter of burden for designing delay times needed for maximum rock breakage. The row-to-row delay to provide good movement and fragmentation is a minimum of 3 ms per meter of burden. Obviously, these values depend on many factors such as rock mass properties, blast geometry, explosive characteristics, initiating system, environmental constraints, or the desired result (fragmentation, muck pile displacement and profile, etc.).

Blast Design to Protect Pit Walls

If blast is not well designed, overbreak can contribute to pit wall instability. Therefore, it is important to optimize but not minimize overbreak, especially as blasts approach the designed wall of the pit. The successful application of overbreak control blasting techniques reduces not only the quantity of rock to be removed, but it lessens the hazard and cost of rockfalls. It can also reduce the need for pit wall support. Cushion blasting, postsplitting, and presplitting are the three more important blasting methods utilized to produce stable final walls.

Cushion blasting is the simplest and least expensive smooth wall blasting technique. It is also the most versatile and useful method of the three techniques mentioned. A cushion blast is a pit wall blast in which back-row blastholes contain lighter charges than the production blastholes and are drilled in a correspondingly small pattern. Cushion blastholes have generally the same diameter as the production blastholes in front of them. The charge weight for the cushion holes is commonly reduced by about 45%. Both burden distance and blasthole spacing are also reduced by about 25%. Cushion blastholes should detonate in a delayed sequence after the more heavily charged blastholes in front of them. This method is utilized without pre- or postsplitting where the rock is strong or only minor reductions in damage are needed or for forming pit walls with relatively short lives (Hagan and Bulow 2000).

Postsplitting is frequently used in conjunction with cushion blasting. It consists of drilling a row of parallel, closely spaced blastholes with a suitable burden to spacing ratio (about 1.25:1) along the final face. These blastholes are charged with light, well-distributed charge, which is fired after the production blastholes in front of them have detonated. This produces a sound, smooth face with minimal damage. Presplitting requires a row of parallel, closely spaced blastholes drilled along the design excavation limit. The blastholes are then charged lightly and detonated simultaneously before the blastholes in front of them. Firing of the presplit charges splits the rock along the designed final face producing an internal surface to which the later-firing blastholes in front of them can break. Presplitting rarely gives impressive results in closely fissured rock. Comparatively, postsplitting gives considerable reduction in damage in massive rocks, but the final face is rarely as sound as that produced by presplitting. In closely fissure rocks, however, the final face formed by postsplitting is sounder than that produced by presplitting. Because the optimum spacing of postsplit blastholes is larger than that for presplit blastholes, the cost of postsplitting is usually lower.

5.6.4 Underground Blasting

As in surface mining, a well-designed blasting process and right execution are crucial components for successful underground mining. Thus, bad blasting procedures can generate a very negative impact on the economics of underground mining (Holmberg et al. 2001). Development of tunnels, shafts, raises, stopes, caving, and other underground openings is performed by means of blasting rounds (Fig. 5.92). The design of underground blasting rounds can utilize two types of rounds: those with one free face and those with more than one free face. Single-face rounds are utilized in development openings (tunnels, shafts, raises) as well as in room-and-pillar, longwall, and shrinkage stoping methods of mining. Multiple-face rounds are indispensable for open stopes, sublevel caving, and large in diameter tunnels that use benching methods. Sometimes, the design of multi-face rounds is similar to that of surface blasting (Dowding and Aimone 1992).

Types of patterns of holes (Fig. 5.93) mainly vary in the configuration of breaking in holes, which are utilized to generate a first free face, toward which the blast is further directed. Otherwise, the rock will be projected outward into the openings, which can damage infrastructure. These patterns can be broadly classified as angled cuts or parallel cuts. If breaking in holes is put at an angle to the axis of the working face, the patterns of holes are known as angled cut. Wedge cut is an example of angled cuts and it is particularly suited to large-sized drifts, which have well laminated or fissured rocks. Blastholes are drilled at an angle to the face in a uniform

Fig. 5.92 Driller marking rock face (Image courtesy of North American Palladium Ltd.)

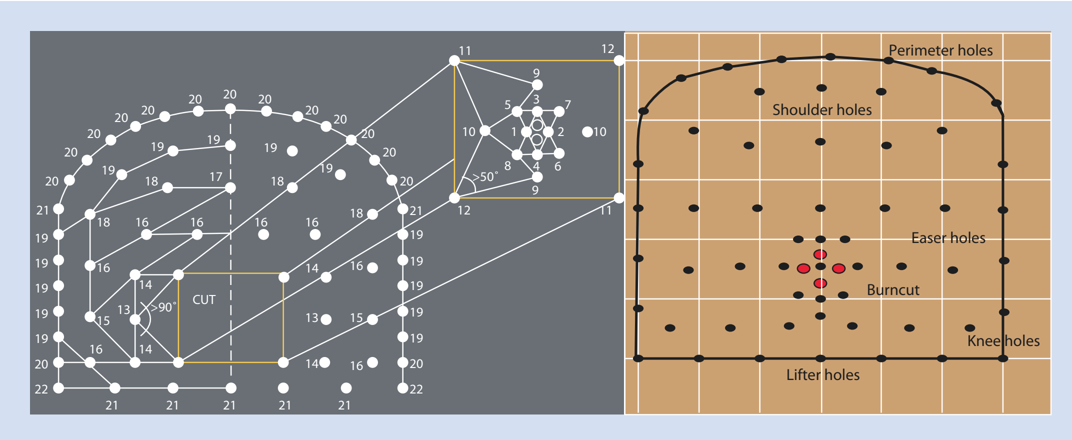

Fig. 5.93 Some examples of patterns of holes

wedge formation so that the axis of symmetry is at the center line of the face. The void into which rock broken by the blast can expand is generally achieved by a wedge cut. The cut displaces a wedge of rock out of the face in the initial blast, and this wedge is widened to the full width of the drift in subsequent blasts, each blast being fired with detonators of suitable delay time. Hole placement should be carefully preplanned, and the alignment of each hole should be accurately drilled. Other hole patterns of angled cuts in underground mining are diamond cut, drag cut, or fan cut.

Firing sequence for a typical parallel hole pattern includes contour or perimeter holes that are fired simultaneously with light explosives, and bottom holes, or lifters, which are fired last to shake up the muck pile. In burn cut, included in the group of parallel hole cuts, a series of parallel holes are drilled closely spaced at right angles to the face. One hole or more at the center of the face are uncharged, the so-called burn cut. When the shock waves are reflected at these empty holes, the rock is shattered and subsequently blown out by the escaping gases. Thus, «there is a specific geometrical relationship between the diameter of empty holes and the spacing between the empty and charged holes, in a given rock, which performs essential conditions of breakage» (Tatiya 2013). Since all holes are at right angles to the face, hole placement and alignment are easier than in other types of cuts. This method is particularly suitable for the use in massive rock such as granite or basalt.

5.6.5 Dangers of Blasting

Dangers of blasting procedures can arise from generation of harmful gases, throw of rocks in the air, pollution by dust, vibrations generated in the ground mass, and propagation of air shock waves. Air blast is an airborne shock wave that originates from the detonation of explosives. Regarding the ground vibration, where an explosive is detonated in a blasthole, a pressure wave is originated in the surrounding rock mass, and as the pressure wave travels from the borehole, it generates seismic waves by displacing particles. Flyrock and elevated air blast levels indicate inadequate confinement, whereas elevated ground vibrations suggest excess confinement. Excessive flyrock, air blast, and ground vibration all indicate inefficient utilization of explosive energy (Rostami and Hambley 2011). On the other hand, improper translation, stored, and handling may be also very harmful.

5.7 Grade Control

Initial production planning commonly depends on exploration information plus smaller amount of information coming from other sources (e.g., surface trenches). However, prior to mining, a program of grade control sampling is generally carried out to define the boundaries of ore and waste blocks and where possible ore and waste blasted separately. Thus, grade control is essential for most mining operations. The grade control or ore control process involves predictive delineation of the tons and grade of ore that will be recovered by mining.

Accurate grade control is essential to the economics of any mine. Mistakes at this step are expensive and irreparable and can be quantified in terms of cash flow losses and incremented operational costs every year (Rossi and Deutsch 2014). Thus, the correct knowledge of grade distribution and optimizing mining selectivity through grade control is crucial to attain the mine plan. It is essential that mill feed be kept as close as possible to that called for in the original design specification of the mill and concentrator. Thus, one of the main purposes of grade control is to ensure that material being fed to the mill is of economic grade as well as minimizing ore loss and dilution. Large fluctuations in grade can be minimized by blending ores from different benches, or parts thereof, or from different stopes. Therefore, a well-managed record database is important for effective grade control and blending. This allows for continuous feed through elimination of fluctuations resulting in homogenized feed grade.

Grade control requirements and practices are largely dependent of the commodity. First, the commodity price controls the implications of ore loss, and management has to justify the extra expenditures relating to selective extraction. Second, the increased mill performance due to lower dilution must justify any additional actions required during the mining processes. Last, the style of mineralization, often commodity-specific, dictates whether grade control is geared more toward ore/waste discrimination or it is focused on grade and stockpile control (e.g., Davis 1992; Wetherelt and Van der Wielen 2011).

Modern grade control has the aim of minimizing errors in the classification of material types in a mining process, but not only ore versus waste but also the allocation of different types of rocks based on grade, deleterious material content, physical properties, or mineralogy. Moreover, the resulting ore grade misclassification is responsible for severe reconciliation problems. It is worthy important to remind that true block grades are never known before to mine and therefore must be estimated. Thus, if the classification of ore type based on the true but unknown block grade is different from the ore type based on the estimated block grade, then the ore type of the block is misclassified.

Grade control is performed at the mine on a daily basis (Fig. 5.94). The potential of grade control for improved profits is large. For example, in a 10 million ton per year copper mine, better grade and control procedure that generate an enhancement in average grade from 0.41% Cu to 0.42% Cu increment gross annual income by about 5 millions of US$, considering copper official prices at LME in January 2016.

Although grade control procedures may differ widely, it usually consists of sampling and assaying to establish the amount and position of the mineralization to define the valuable ore areas. Grade control generally entails sampling and assaying of blasthole cuttings followed by estimation of ore control block model grades. Often, blasthole samples are not as useful as samples produced from exploration or RC drillholes, but the compara-

Fig. 5.94 Grade control prior blasting (Spain) (Image courtesy of Daytal Resources Spain, S.L.)

tively huge amount of blasthole samples forthcoming will minimize the influence of the error of a single blasthole sample. In some cases, grade control can also involve the sampling of truck or shovel loads to ensure that rock is assigned to the correct stockpile or waste dump (Annels 1991).

Grade control should always be seen as a complex process in which at least three basic aspects must be considered: (a) data collection and quality, (b) grade control model to determine ore and waste boundaries, and (c) operational procedures and constraints, including mining methods and mining practices (Rossi and Deutsch 2014). Since grade control depends on a large number of samples, the estimation data to define the grade model can be carried out applying classical methods such as inverse distance or nearest point methods or geostatistical techniques (kriging). Nowadays, grade control practices have evolved from paper-based recording methods to computerized three-dimensional modeling and geostatistical simulations.

5.7.1 Open-Pit Grade Control

In an open-pit operation, grade control involves sampling of blasthole cuttings produced by drills and classification of bench reserves into ore, low-grade, and waste material or into various metallurgical types. The final and irreversible decision as to what is ore and what is waste is generally made on a daily basis. Blasthole samples are obtained on closely spaced grids (◘ Fig. 5.95) according to blasting requirements. The use of blastholes can be contentious for different causes, including sampling quality, and disagreements of its grade distribution compared to the exploration drillhole grade distribution. Consequently, classical blasthole sampling has gained an extraordinarily poor reputation for the last five decades due to the introduction of many sources of bias in the procedure (Pitard 2008). Some of these biases are due to the type of drilling machine that is used and are nearly unsolvable. Others are due to the sampling tools used, often unsatisfactory. Furthermore, there is commonly an unsolvable time logistic problem in blasthole sampling: the miner wants that the ore grade control was carried out in 2 or 3 days at most, but not enough time is allowed for samplers, preparation facilities, laboratory, and resources department to perform an accurate job.

The amount of sampling (◘ Fig. 5.96) is constrained by both practical limitations and cost considerations, but random sampling errors can be large if sample volume is too small. There are different sampling methods to choose from, including different grid patterns and spacings, although all sampling methods incur errors. In open-pit operations, possibly the most typically utilized method to forecast in situ grades is the arithmetic average of the forthcoming blastholes. Thus, a block model is developed, commonly with the block size similar to the blasthole spacing, and the predicted block grade is the arithmetic average of the blastholes that fall within the

◘ **Fig. 5.95** Closely spaced blasthole samples (Image courtesy of Octavio de Lera)

Fig. 5.96 Grade control samples (Image courtesy of Alicia Bermejo)

block. Many times, «the blocks are relatively large with respect to the average distance between sample points, which is unjustifiable and a major source of inaccuracies because the data density is generally sufficient to justify much smaller blocks; thus, smaller blocks would lead to better definitions of ore and waste boundaries» (Rossi and Deutsch 2014).

As an example of the overall process, grade control at Skorpion mine in Namibia, a supergene zinc oxide deposit, includes the following steps (Gnoinsky 2007). Drilling is carried out considering drill burden and spacing based on the rock types, for example, 4 m × 4 m in ore and 6 m × 6 m in waste material. Samples and drill chips are collected over intervals of 2.5 m, 5 m, or 10 m. Large samples (>5 kg) are riffle split at the drill rig. For assays, routine XRF analysis of Zn, Ca, Fe, Mn, and Cu concentrations in pressed powder pellets are then performed. The quality control procedure specifies the use of quality control samples to track laboratory performance. At least 5% of the grade control samples are submitted for analysis and comprise quality control specimens, including matrix-matched certified reference material, internal geology blanks, laboratory replicates, and laboratory blank samples, among others. Survey borehole collar coordinates geology and assay results are then captured in a database. Block modeling is undertaken over appropriate levels (selective mining unit = 5 m × 5 m × 2.5 m) depending on continuity of geology and grade. Mining perimeters are delineated based on assay results, and grade and tonnage evaluation reports and loading plans outlining mining perimeters in the open-pit are then generated.

5.7.2 Underground Grade Control

In underground mining, mining methods are insufficiently flexible, and therefore there is no chance for ore and waste definition at the time of extraction. In these situations, grade control can be based on infill drilling and completed at the time of defining the stopes to be extracted. Any failures that can take place at this situation are not only irreparable but also cannot be balanced by other types of errors, as it is in some cases with resource calculations (Rossi and Deutsch 2014). Grade control can involve mapping and sampling of stope faces, sampling of tramcar loads or drawpoint muck piles, broken rock at a recently blasted face, jackhammer cuttings, or diamond drill cores. Samples are measured off along the stope at specified intervals and marked on the face. The process is laborious and includes to extract the marked sample by chipping an exact rectangle from the solid rock face and to ensure at the same time that all the rock fragments are collected. It is the geologist's job to guarantee that mining is closely following the mineralized zone and that overbreak during stoping is kept to a minimum (Annels 1991).

At Big Gossan Mine (skarn-type deposit), an open stope-paste backfill underground mine in Papua Indonesia, the major objective in the grade control drilling program is to identify the grade boundary in certain levels to guide the mine planning in preparing the stope shape as well as the stope access development (Haflil et al. 2013). This program is designed in a fanlike drilling pattern so that the drilling covers the stope and also 40 m above and 40 m below the targeted stope using a diamond drilling. The drilling design is usually from the footwall toward the hanging wall of the mineralization. Detailed logging is conducted to gather better knowledge of the formation, mineralization, and alteration boundary. Thus, ore/waste boundary is defined based on the chalcopyrite mineral content, and the ore type is classified based on skarn mineral content. Sampling interval is also determined using those boundaries as a guide; sampling is done continuously along 3 m intervals. Prior to splitting, crushing, and assaying of the core, the core is measured on its rock mechanical properties. Geotechnical logging includes specific gravity, RQD, and point load tests. Assay testing covers five elements (Cu, Au, Ag, Pb, Zn), and the assay data is systematically stored in a drilling database. Standard QA/QC sampling practices include duplicate samples, blank samples, and certified standard samples for every fifteenth sample.

A grade control block model or short-range block model is constructed for short-term and stope mining purposes. This block model is created on 2.5 m × 2.5 m × 2.5 m block from 5 m drill core composite lengths and includes data from the updated grade control drilling. This short-range block model is used as a guide in determining the metal tonnage a stope produced. Stope reconciliation (see next section) is conducted after all material from the stope has been mined out. Grade and tonnage reconciliation compares the grade and tonnage from the planned stope based on the short-range block model versus the grade and tonnage using the present stope shape. Determining the dilution is also a main part in the reconciliation process. By utilizing the short-range block model, it is possible to predict the expected grades and tons to be produced in a stope. Daily grade-to-mill and stope reconciliation are based on data produced from this model.

5.7.3 Grade Control and Reconciliation

Predictions of grades in grade control process have a number of common characteristics across all mineralization and mining types, from small, low-production-rate metalliferous underground mines to large world-class open-pits: (a) abundant geological data that can have only minimal relevance or cannot be used, (b) abundant sampling data that can be of relatively poor quality (e.g., have significant sampling errors), and (c) production pressures requiring fast interpretation of the data and rapid prediction of the ore blocks. For this reason, besides ore/waste rock discrimination and assigning metallurgical grades to material, grade control provides a basis for reconciliation of mill production figures, geostatistical models, and pit production tonnages and grades (Davis 1992) (Box 5.15: Reconciliation).

Box 5.15

Reconciliation

Reconciliation is the process of comparing predictions to actual production. Regular reconciliations will be required between the estimated mining grades, the grades indicated from stope/bench sampling, and those reported by the mill (Fig. 5.97). It is essential that this is undertaken so that modifications can be made to sampling practice or to the methods or parameters used to calculate grade, tonnage, or contained metal (Annels 1991). In fact, reconciliation will increasingly become the benchmark by which mining company performance is judged, based on comparing actual production with predictions (promises).

In a mining industry context, reconciliation equates to the comparison of an estimate (a mineral resource model, a mineral or ore reserve model, or grade control information) with a measurement (survey information or the official production, usually from the processing or treatment plant).

Reconciliation does not of itself generate errors, but it can identify the net impact of the errors in the process. It is not necessarily the determining test as to whether the mine is successful. A mine may be profitable even if it is based on a poor mineral resource, a poor ore reserve (which includes mine planning practices), or poor mining or processing practices (Shaw et al. 2006).

Therefore, the most useful concept of reconciliation is that

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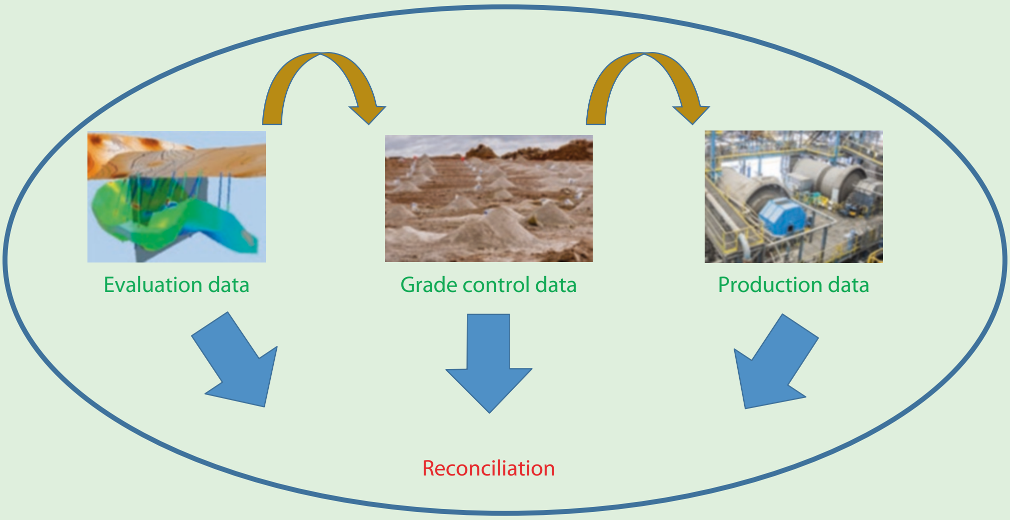

Fig. 5.97 Reconciliation process

of ore reserve (prediction) to grade control (prediction) to mining (production) to milling (production). The basic aims of reconciliation are (a) to measure performance of the operation against targets, (b) to confirm grade and tonnage estimation efficiency, (c) to ensure valuation of mineral assets is accurate, and (d) to provide key performance indicators, in particular for grade control predictions (Morley and Moller 2005). Thus, reconciliation of resource and reserve models, grade control models, mine production data, and plant tonnage and grade are one of the most vital functions in the mining cycle. Reconciliation can also highlight any issues in the reserve to production process and in the stockpiling systems. Minimizing the difference between planned versus actual production will improve business performance. Consequently, the implementation of a reconciliation system often generates a range of benefits such as lowering costs, improving efficiency, enhancing the accuracy of estimates, and saving capital.

In the simplest case, the shareholders want to see a comparison between the annual net revenue for the mine compared to the predictions made to them at the end of the feasibility study. Defining this question more tightly, they want to know there are comparisons of production against predictions for ore and metal produced over consistent volumes and time periods. Reconciliations should be consistently monitored over time. A successful predictive approach can deteriorate due to changes in geology, ore type, sampling procedures, grade control methods, mining methods, milling controls, etc. Lack of systematic reconciliation means that there are no controls to monitor the predictions, and this can result in wrong use of the resource and profit objectives not being met (Shaw et al. 2006). It is useful to know that the mill is receiving the predicted ore at a lower than expected grade, even while there is still uncertainty as to whether this is due to problems with the ore reserve (due to data, interpretation, or estimation), with the grade control (due to similar errors plus ore loss and dilution), with mining (due to deviations from the plan), or with milling (due to sampling errors or losses). Similarly, it is useful to know that production is exceeding predictions since this can mean the grade control process, the mine plan, and the revenues are all suboptimal.

In summary, a robust reconciliation system enables the total mining operation to be seen in context, major problems and sources of error to be identified, both underestimation and overestimation to be critically monitored, improvements to be tested and evaluated, and reporting to management and communication to shareholders to be clear and consistent. Reconciling from the resource through to delivery of a mineral product is the key to adding value during development of a mining project.

5.8 Questions

Short Questions

- What is mineral extraction?
- List the production operations that conform the production cycle.
- Define stripping ratio. What are the stripping ratio values in metal mines?
- What does dilution mean in mineral extraction?
- What are the most common surface monitoring techniques in geotechnical studies?
- Explain the in-pit crushing and conveying process.
- What is the objective of Lerchs and Grossmann algorithm?
- What is the main difference between open-pit mining and strip mining? How are they related?
- Explain the concepts of transition depth and crown pillar in the combination surface mining-underground mining.
- What is a shaft in underground mining?
- What are the more commonly used surface rock support methods?
- What does block caving mean in underground mining?
- Define the blasthole drilling procedure.
- What is ANFO?
- Explain why delay blasting is very used in mineral extraction.

Long Questions

- Describe in detail the room-and-pillar underground mining method.
- Discuss the grade control and reconciliation processes in mineral extraction.

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Mineral Processing

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Summary
The run-of-mine extracted from the ground requires further processing in order to make a marketable product. This preparation is called mineral processing. Thus, ores must go through a number of different operations to obtain the final products: comminution or size reduction, size separation, concentration or beneficiation, and dewatering. This chapter is devoted to explain all the processes and equipment involved in the operations cited above, with special emphasis in flotation method. This is certainly the most important and flexible mineral separation procedure. All the operations are described including up to eight different case studies of processing plants. At the end of the chapter, waste/tailings disposal heading provides a short review of the main methods of tailings disposal. It is essential to take into account the fact that tailings must be disposed of in specially engineered repositories capable of containing the fine-grained and often saturated tailings mass without the risk of geotechnical failure.

6.1 Introduction

The solid value-bearing material that is generated in the mine is termed «run-of-mine,» being a material particulate in nature and consisting of a heterogeneous assemblage of different mineral species. The run-of-mine extracted from the ground using the methods described in ▶ Chap. 5 requires further processing in order to make a marketable product. A large amount of material formed by ore minerals, gangue minerals, and host rock from the mine goes to the processing plant, being transformed by selectively concentrating the valuable components and rejecting the rest. Thus, any mineral as mined is not suitable for conversion to a final product because minerals are needed as highest state of purity as possible for their uses. Thus, ore requires preparation that is commonly carried out by physical methods. This preparation is called mineral processing. It is also termed mineral dressing, mineral beneficiation, mineral concentration, mineral engineering, mineral extraction, or simply minerallurgy.

In a broad sense, the term mineral processing is utilized to define the unit operations involved in recovering minerals or metals from ores. Under any circumstance and even at any intermediate stage, mineral processing should not alter the chemical composition of the minerals for the subsequent treatment. A wider concept of mineral processing includes chemical methods of treating minerals and therefore extends across the field of extractive metallurgy to the production of commercially pure metals. Basically, only the physical processing of ores is considered in this chapter.

The treatment of the ore is carried out in process plants (◘ Fig. 6.1) generally located at the mine site to reduce costs of long transportation. It is important to note that the grade of valuable minerals in host rocks is usually very low. Accordingly, large tonnages of materials have commonly been extracted to obtain the required amount of valuable mineral to make the process commercially viable. Other terms for a process plant are concentrators, if base metal ores are treated, or mills, since grinding in mills is a typical process in the plant.

Haldar (2013) described mineral processing «as the value-added processing of raw material (run-of-mine ore) to yield marketable intermediate products (e.g. copper concentrate) or finished products (e.g. silica sand) containing more than one valuable minerals and separation of gangue (tailing).» The run-of-mine components consist of the following:

1. Useful materials such as building and decorative stones (e.g., granite, marble, or slate); materials in this category are not classified as minerals.
2. Useful minerals such as calcite, fluorite, apatite, or barite; they are often referred to as industrial minerals.
3. Metalliferous minerals such as chalcopyrite, sphalerite, or galena; these minerals have not commonly any direct use and require further treatment by minerallurgical or metallurgical methods.
4. Precious metals such as gold, silver, or platinum (in native form).

The main goal of mineral processing is to reduce the amount of ore that must be translated to and processed by the smelter where the metal is obtained from the ore. It is carried out by utilizing relatively non-expensive, low-energy physical methods to split the useful minerals from the gangue. This enrichment procedure notably increments

Fig. 6.1 Aguas Teñidas processing plant (Spain) (Image courtesy of Matsa, a Mubadala & Trafigura Company)

the value of the ore to enable further transportation and smelting. Thus, the goal in mineral processing is to generate maximum value from a given raw material. The technologies to achieve this goal are classical, complementary, and well defined, being the field of mineral processing based on many fields of science and engineering. Nevertheless, the steps involved in mineral processing have to be founded not only on sound scientific and technological bases but on environmentally acceptable grounds as well.

The history of mineral processing is as old as the history of human society. As an example, the Egyptians understood that it would be easier to melt a material rich in gold particles than another that is poor. Consequently, endeavors were carried out to enhance the gold content by washing the lighter gangue minerals. In the words of Habashi (2014) «Mineral processing was an art until the 1920s when it started to become a science; about three decades later, it became a highly sophisticated science.»

An essential clarification in mineral processing terminology can be outlined by Kellerwessel (1991). He stated that «originally, the term primary resource processing referred mainly to techniques for dressing raw materials obtained from mines; such techniques are now classified as mechanical processing in contrast to metallurgical techniques where the value minerals are chemically altered, such as in the reduction of iron ore (iron oxide) to extract metallic iron, and also in contrast to conventional chemical processing.» This clarification is crucial since there is common confusion between mineral processing and metallurgical processing. The aim of the former is to recover minerals concentrating them, whereas the objective of the latter is to extract metals from ores. The end product of mineral processing is an ore

Fig. 6.2 Ore concentrate (Image courtesy of TEFSA)

concentrate (Fig. 6.2) that is then put through a metallurgical process. Pyrometallurgy, hydrometallurgy, or other chemical methods are used to refine the concentrate for extraction of metals in the purest form. In this sense, a brief description of hydrometallurgical processes is also included in this chapter because this method of metal extraction is commonly carried out at some metal mines as a mineral (metal) beneficiation process.

6.2 Basic Concepts

6.2.1 Concentrates and Penalties

Minerals are concentrated in mineral processing, and the resulting concentrate is commonly marketed to a particular smelter, depending of the type of concentrate. Most concentrates include various minerals, some of them including metals with economic value (e.g., copper, molybdenum, zinc, lead, nickel, gold, and silver) and less common metals such as antimony, arsenic, bismuth, cadmium, cerium, cobalt, gallium, indium, mercury, platinum, tellurium, tantalum, titanium, and vanadium. It is indispensable not only to split meaningful from gangue minerals but also to split meaningful minerals from each other. For instance, complex sulfide mineralization including economic quantities of copper, lead, and zinc commonly need separated concentrates of the minerals of each of these metals.

Concentrates are generally specific to certain smelters. Therefore, a lead smelter or a zinc smelter could not smelt a copper concentrate. Including copper in a lead or zinc concentrate is troubling for the smelting process and generates frequently a penalty charge (Box 6.1: Penalty). For this reason, lead and zinc concentrates need to be marketed as separate products, although there is also a market for mixed lead/zinc concentrate, usually termed as «bulk concentrate.» Concerning precious metals, silver above a minimum content will be paid (no payment is made if there is less than 30 g of silver per dry ton), but usually gold is not paid for. This is because the smelter recovers silver easily but cannot recover the gold at all. For gold, quantities below 1 g per dry ton of concentrate typically do not receive payment, because they are usually unreasonably expensive or difficult for smelters to recover. If the gold content is considerably high, silver-/gold-containing product can be marketed to a third-party smelting plant that specializes in processing of such a product. The provision of clean concentrates without associated metals is not always economically feasible.

Box 6.1

Penalty

These impurities are found in the gangue minerals, and the purpose of mineral processing is to reject them, as smelters often impose penalties according to their level. Metals misplaced into the wrong concentrate are rarely paid for by the specialist smelter and are sometimes penalized. A metal reported to the «wrong» concentrate can be difficult, or economically impossible, to recover and never achieves its potential valuation. For instance, lead is essentially irrecoverable in copper concentrates and is often penalized as an impurity by the copper smelter (Wills and Finch 2016).

The price paid by the smelter depends primarily on the market process of the metal, but penalties are introduced in the prices for constituents in the ore or concentrate that are detrimental to the smelting process. Thus, separation of penalty elements from the final saleable concentrate can be vital in order to avoid severe financial penalties from smelters. The agreed price for concentrates is typically based on a formula, which is the sum of value of the contained metals («payable metals») less the sum of deductions (treatment and refining charges) and penalties imposed. Thus, penalty elements reduce the overall grade and/or value of the concentrate for sale. Charges will vary, depending on the process used to smelt and refine the concentrate, but typically the penalty is a USD per ton for incremental percentages above a specified threshold. For instance, arsenic penalties in copper concentrates are generally in the range of US $2.50–5.00/dmt (dry metric ton) per 0.1% As over 0.2% As. Bismuth penalties in lead concentrates are about US $2/dmt for each 100 ppm above 300 ppm, antimony penalties are US $2/dm for each 0.1% above 0.2%, and arsenic penalties are US $2/dmt for each 0.1% above 0.2%. If too high, excess concentrations of some elements will result in the concentrate being rejected, usually because they either exceed environmental or safety limits, are unreasonably difficult (and therefore expensive) to treat, or the materials are expensive to dispose of (such as mercury).

Penalty elements encompass a large variety of elements based upon a smelter contract between a specific concentrator and its buyer, the smelter. The penalty charge is specific to particular elements that result in additional process activities being necessary to remove them from the final metal product or that need to be disposed of in an environmentally friendly manner. A penalty is usually specific for a particular deleterious element, and is related to the cost of its removal and/or disposal, or the extent to which it reduces the value of final metal product. Thus, penalties apply to those elements that are toxic and expensive to dispose of such as mercury. As an example, it is necessary to remove arsenopyrite from tin concentrates since it is difficult to remove the contained arsenic in smelting and produce a low-quality tin metal. Sphalerite rejection from copper concentrates is often of major concern because zinc is a penalty element in copper smelting. There are many other penalty elements, and if some elements are high enough, the concentrate cannot even be saleable. The most important impurities affecting prices for iron ore products are silica, alumina, phosphorous, sulfur, and loss on ignition impurities, which refers predominantly to moisture content. This is because of the unwanted effects they have on the properties of iron and therefore steel. In addition, alkalis such as lithium, sodium, and potassium may affect prices if they are above trace amounts, but this is less common. Penalties may also be applied for concentrates with excessive moisture. ▪ Table 6.1 shows the main penalties and their limits in lead, copper, and zinc concentrates.

For instance, a zinc penalty means that for each 1.0% units by which the final zinc assay exceeds 3.00%, the seller shall pay a penalty charge of US $3.00 per ton of concentrate. If it is assumed a gross metal value per ton of concentrate of US $1200, the net amount payable to the seller could be of the order of US $800, after all charges and penalties. Thus, only 67% of the value of the metal in concentrate is paid. It demonstrates that the net amount paid is often significantly lower than the original value considered. Penalty clauses are typically applied for the impurities harmful to smelting and refining processes. Therefore, it is evident that the terms agreed between the concentrator and smelter are of paramount importance in the economics of mining and milling operations. Such smelter contracts are usually rather complex.

Table 6.1 Main penalties and their limits in lead, copper, and zinc concentrates (Data courtesy of Sander de Leeuw)

Lead concentrate penalties
Arsenic: 0.3%
Antimony: 0.3%
Zinc: 6%
Cadmium: 0.02%
Bismuth: 0.07%
Copper concentrates penalties
Arsenic: 0.2%
Antimony: 0.1%
Lead: 0.5% (some smelters recover the Pb and do not penalize this)
Zinc: 3.5%
Bismuth: 0.02%
Mercury: 20 ppm
Zinc concentrates penalties
Magnesium oxide: 0.4%
Iron: 8%
Cobalt/nickel: 0.1%
Cadmium: 0.5%
Mercury: 300 ppm
Silica: 8%

6.2.2 Grade and Recovery

An essential characteristic of mineral processing separations is that they are never perfect: some of the valuable product is always in the waste stream, and some of the waste is always in the valuable stream. This is due to the combined effects of mineral inseparability and inefficiencies in the separation process. Because these inefficiencies cannot be avoided, a further objective of the separation process is to minimize the extent to which minerals are misplaced to the wrong stream. One of the most interesting and primary concepts in mineral processing to describe adequately the extent of the separation is the efficiency, grade and recovery being the two indicators commonly used to describe it. Used simultaneously, they are the most broadly agreed measurements of evaluating metallurgical performance. These values rely on the separation of meaningful minerals and their separation from associate gangue. Since the objective of mineral processing is to increment the value of the mineralization, the significance of the recovery-grade relationship is in establishing the best economic combination of recovery and grade that will generate the greatest financial return per ton of mineralization processed in the plant.

The grade of a concentrated stream is a measure of its quality. The ideal result would be that the valuable product stream should be of high quality and the tailings stream of low quality. Obviously, the grade of the concentrate will be maximum if the separation is perfect. It should be noted that this maximum could not be 100%. For example, the copper grade of a pure chalcopyrite concentrate would be 34.6%, the remaining 65.4% consisting of the other mineral constituents, iron and sulfur. A copper concentrate commonly includes chalcopyrite, probably another sulfide such as pyrite and other gangue minerals. Thus, the global grade of the concentrate could range from 20% to 30% copper.

The term recovery, commonly expressed in percentage, refers to the percentage of the valuable mineral reporting to the concentrate with reference to the amount in percentage of this material in the feed. For example, a recovery of 90% implies that 90% of the metal in the mineralization is retrieved in the concentrate and 10% is sent to the tailings. Recovery measures therefore how effectively the separation process has extracted the valuable mineral contained in the input stream. Although 100% recovery is possible in principle, in practice it can be achieved only if all the feed is diverted to the concentrate and no separation is made. Sometimes, obtaining the highest possible recovery is not necessarily the best approach in a concentration process. High recovery without

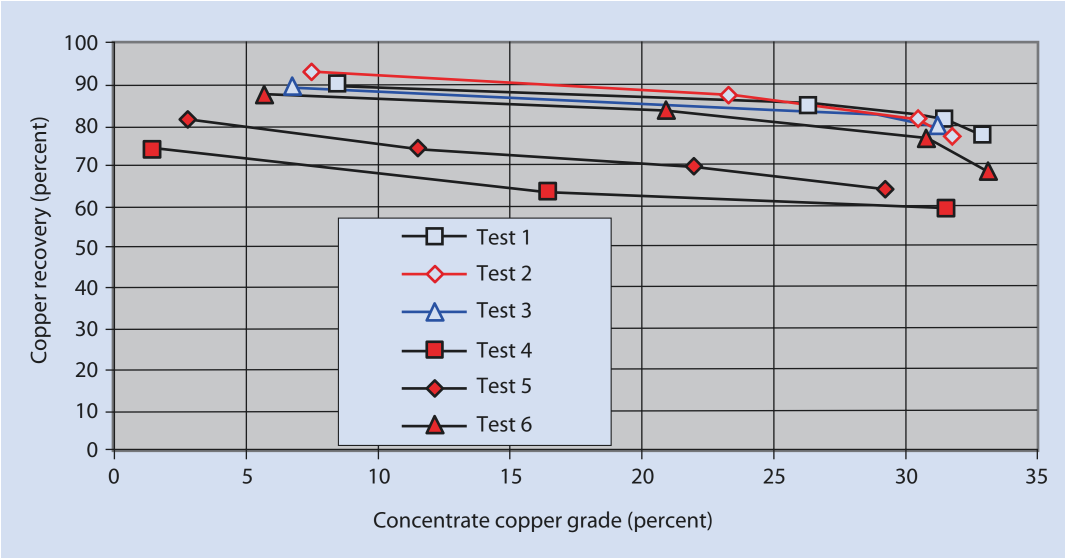


Fig. 6.3 Grade-recovery curves in copper concentrates

acceptable grade will lead to an unsalable product and is consequently unsatisfactory. Thus, economic evaluation of all potential technological alternatives must be carried out to obtain the highest possible recovery with reasonable grade (Fuerstenau and Han 2003).

Since there is roughly an inverse relationship between recovery and grade, the goal of achieving maximum recovery and maximum grade are always in conflict. Thus, the aim of mineral processing procedures is to keep the values of grade and recovery as high as possible and taking into account all factors involved. In practice, what is usually done in mineral separation is to pursue maximum recovery while achieving acceptable, not maximum, concentrate grade. In this sense, grade-recovery curves (Fig. 6.3) indicating the relationship between the grade of the concentrate that will be achieved for a given recovery of a mineral are commonly performed in mineral processing separation.

6.2.3 Net Smelter Return

Net smelter return is a measure of value of the ore. It is equal to the revenues derived from the sale of the products of the ore minus all off-property treatment and distribution charges. NSR method is commonly used to analyze the economic impact of the degree of concentration of enriched minerals in the light of processing costs and metal market prices. It can be a criterion for optimizing the extraction and beneficiation of ore according to the quality of the concentrates. More in detail, net smelter return concerns to the incomes awaited from the mill feed considering mill recoveries, transportation costs, processing charges, and other deductions at the smelter (Table 6.2).

Although not all items listed are applicable to each metal concentrate, the main factors required for the calculation of the NSR are (a) recovery factor of the metal, to know what proportion of the metal sent to the mill is actually sold; (b) concentrate grade, to establish the amount of metal contained in a ton of concentrate; (c) transport cost, from the mine site to the smelter; (d) payable metals, to establish the base quantity of metal that the smelter will use to determine payment; (e) treatment charges, to determine the cost of processing one ton of concentrate at the smelter; (f) penalties, the added cost of processing detrimental elements present in the concentrate; and (g) refining charges, to determine the cost of refining the metal recovered at the smelter.

Table 6.2 Example of NSR calculation parameters

Parameter name	Units	Parameter value
Copper price	USD/pound	3.21
Gold price	USD/troy ounce	1200
Recovery for copper	%	85.2
Recovery for gold	%	86.8
Copper concentrate grade	%	25
Copper concentrate moisture	%	10
Copper concentrate losses	% (weight)	0.25
Payable copper	%	96
Payable gold	%	93
Copper deduction	%	1
Gold deduction	g/dmt concentrate	0
Treatment cost	USD/dmt concentrate	80
Copper-refining charge	USD/payable lb of copper	0.08
Gold-refining charge	USD/payable oz of gold	5
Mercury penalty	USD/dmt concentrate	US $0.15 for every 1 ppm > 20 ppm
Arsenic penalty	USD/dmt concentrate	US $3.00 for every 0.1% > 0.2%
Freight, port, assays, marketing	USD/dmt concentrate	US $142

6.3 Steps in Mineral Processing

In mineral processing, ores must go through a number of different operations to obtain the final products. Thus, mineral processing involves four major steps or stages: (1) comminution or size reduction, (2) size separation, (3) concentration or beneficiation by taking advantage of physical properties, and (4) dewatering. First, the solid material must be prepared in an appropriate way. The most fundamental requisite is that the mineral to be extracted should be physically liberated from the gangue as discrete particles. If particles consist of both mineral and gangue, they need to be broken into smaller fragments until the mineral and gangue are physically liberated from each other, usually in the range between 10 and 200 µm. This liberation step is achieved by comminution.

Once a satisfactory degree of mineral liberation has been achieved, the material to be processed consists of two types of particles distinguished according to their mineralogical composition: valuable mineral and gangue. Mineral separation can be therefore carried out by the engineering of a separation environment in which particles of different physical properties undergo different physical forces and so move in different directions. Obviously, particles of different mineralogical composition will almost always possess different physical properties. The stream to which the bulk of the mineral reports is usually called the concentrate stream. The other stream (gangue minerals) is termed the tailings stream.

A deep understanding of the precise mineralogical composition of the ore is crucial if effective treatment needs to be performed, including not only the features of the meaningful and gangue minerals but also of the texture of the mineralization. Thus, the beneficiating of ores must consider the context of the composition of the mineralization with the objective to predict grinding and concentration needs, suitable concentrate grades, and possible difficulties of concentration (Baum et al. 2004). For example, a complex sulfide mineralization containing microscopic size particles of sphalerite within other sulfides displays a special challenge to the definition of the mineral processing treatment.

6.3.1 Size Reduction

The first step in mineral processing is the liberation of meaningful minerals between themselves and related gangue minerals at the coarsest possible particle size for economic considerations. This is because most ore minerals are usually finely dispersed and intimately linked to gangue minerals. In fact, the purpose of the size reduction or comminution process is threefold: (a) to liberate valuable minerals from the ore matrix, (b) to increase surface area for high reactivity in the further separation process, and (c) to facilitate the transport of ore particles between unit operations.

The degree of release is the percentage of a given mineral existing as liberated particles; they are particles containing only that mineral. Particles that contain both valuable and gangue minerals are known as locked or middling particles. A large proportion of the difficulties arisen in mineral separation are associated with the treatment of these particles. To avoid problems in comminution process, breaking of the particles should be preferably carried out at the material interlocking, that is, at grain boundaries. Comminution plays a major role in mineral processing since this operation is performed starting the treatment sequence, producing a very high impact on the effectiveness of the downstream processing stages. In other cases, comminution can also be required where a solid product must conform to the size specifications ruled by the market.

Table 6.3 Global energy consumption in a beneficiation plant

Process	Energy consumption (%)
Crushing	9
Grinding	38
Flotation	24
Dewatering	22
Other	7

Comminution commonly involves crushing and grinding. They are necessary for whatever further reduction in size may be required. Crushing and grinding is the most costly step in mineral processing because the power needed to liberate the minerals is very high. In this sense, grinding is commonly said to be the key to good mineral treatment and the major consumer of energy, accounting for up to 50% of a concentrator's energy consumption (Table 6.3). Obviously, the fineness of the grinding process is a crucial feature in mineral processing because fine grinding notably increment power costs and can generate very fine untreatable slime components that can be lost into the tailings. Thus, grinding therefore is a compromise between clean (high-grade) concentrates, operating expenditures, and losses of fine components (Wills and Finch 2016). For this reason, a great number of test studies are carried out in all mining projects to establish the best grind size for the mineralization.

6.3.2 Size Separation

In the second stage of mineral processing, the valuable mineral or minerals are subjected to size separation, which is the division of particles according to their size. This process is essential for further appropriate concentration process, and it can be carried out dry or wet. Separation processes include screening and classification. Both methods are distinct and differ in the size of the components to be separated. Thus, the coarser elements are split using screening techniques, whereas the particles that are considered too fine to be sorted

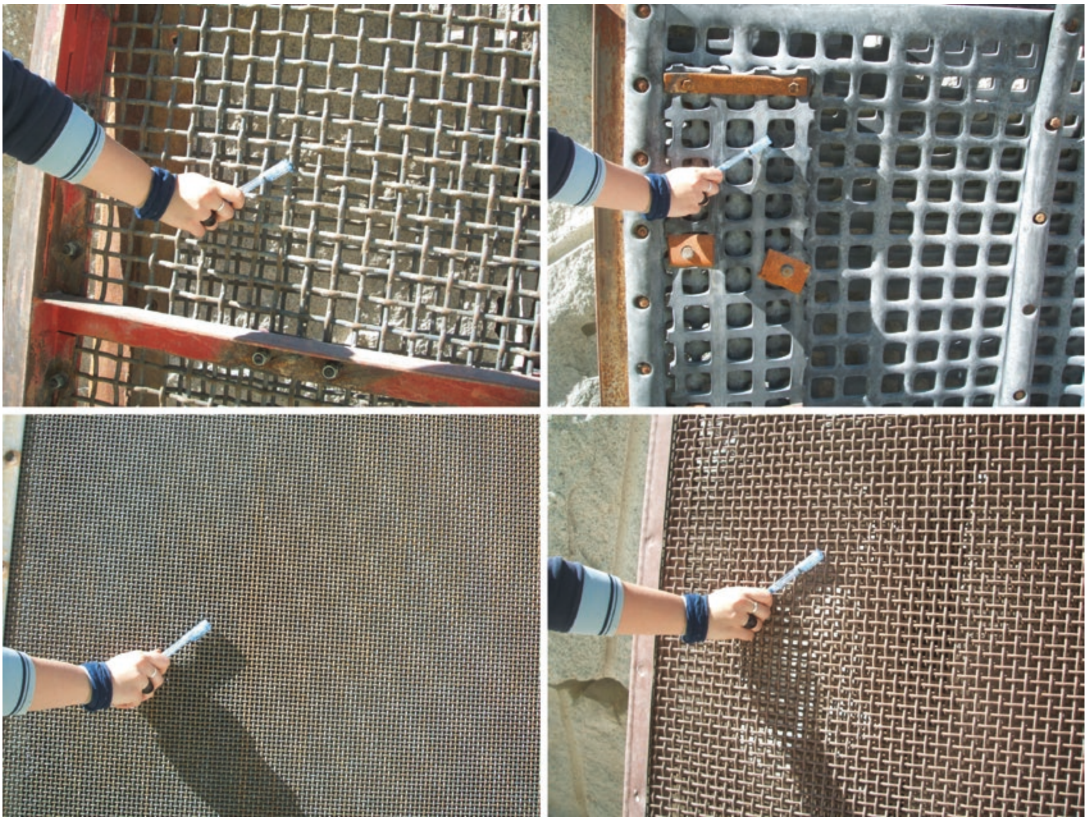


Fig. 6.4 Different types of screening surfaces and apertures

efficiently by screening are separated by classification methods. Screening, also termed mechanical classification, separates the different components utilizing the contrasts in particle size. The components are split utilizing a plastic or metallic screen with a perforated surface including a certain dimension aperture (Fig. 6.4). Materials are sent to the screen surface and the components that are smaller than the screen opening pass, while larger particles are targeted to a designated place.

The purpose of screening is therefore splitting the feed into two or more different products in size. Screening is a continuous process performed on a large scale, while sieving is performed on sieves on a small batch laboratory scale. Particles going through the screen form a product usually called undersize, minus, or lower product, while the ones not passing through the screen are known as oversize, plus, or upper product (Drzymala 2007). Since each screen usually provides two products, it is necessary to use additional screens to obtain more products. In this case, an intermediate product refers to the material passing through one screen and retained on a subsequent screen.

Separation using the velocity of the grains falling through a fluid is an essential technique because separation by screening is not effective for fine materials. This method of separation based on the settling rates caused by the variable size of the components in a fluid (commonly water) is termed classification. Thus, particles of the same shape and density but of different sizes will be separated using classification. In sedimentation classifiers, the feed is supplied from the upper part of the container, and particles fall down in water vertically or nearly vertically. Fine particles, those which settle slowly, are removed from the classifier together with water as overflow. The particles

Fig. 6.5 Hand sorting at conveyor belt (Image courtesy of Sumitomo Metal Mining Co., Ltd.)

settling rapidly are removed either as an underflow at the bottom or with the use of appropriate mechanical devices.

6.3.3 Concentration of Valuable Components

The third stage in mineral processing is the separation between valuable mineral or minerals and waste. The term separation here is synonymous with concentration. The methods of separation are commonly based on the difference of physical properties between meaningful and gangue minerals. The major physical methods utilized to concentrate mineralization are carried out by (1) optical and other properties, (2) differences in density, (3) surface properties, and (4) magnetic and electrostatic properties.

The most obvious physical property that is exploited in mineral beneficiation is that of appearance. Particles of different mineralogical composition can look different, so that they can be separated either by hand or by an automated system using a sensor. Separation based on appearance, color, texture, and radioactive properties is often called sorting. However, sorting methods are actually only of marginal importance compared to the rest of beneficiation methods. Hand sorting (Fig. 6.5) from conveyor belts has reduced in significance with the increased developing of mechanized techniques to treat large tonnages, but it is still utilized in countries where abundant cheap labor is available.

Since the valuable minerals are often significantly denser than the gangue minerals, this physical difference can be utilized to separate them. The various available techniques fall under the general title of gravity concentration. In one type of gravity concentration, the separation is carried out mainly by mechanical methods in water in which heavy minerals are the valuable mineral source. This separation process (a technology with its roots in antiquity), is based on the differential moves of mineral particles in water due to their different hydraulic features (separation of gold by density difference dates back to at least 3000 BC as depicted in writings from ancient Egypt). To define the adequacy of gravity separation process to a specific mineralization type, the separation of minerals relies mainly on a particle's settling rate in water.

If the density of the fluid medium that separates the valuable and gangue minerals lies between the densities of them, the separation becomes easier. Thus, in dense medium separation particles sink or float in a dense fluid or more commonly in a man-made dense suspension. This method, also called sink-float separation, is extensively utilized in many cases such as coal concentration, iron ore, and diamond treatment and in the preconcentration of metal mineralization (e.g., lead and zinc ores). Obviously,

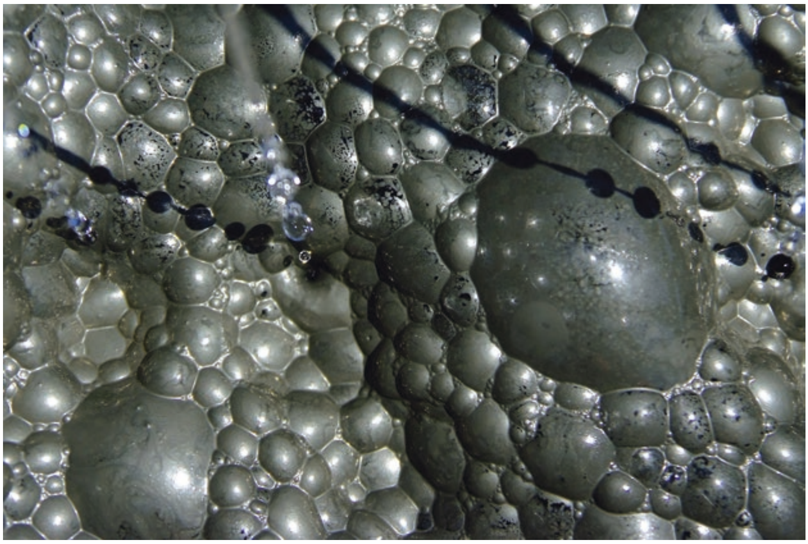


Fig. 6.6 Bubbles in froth flotation (Image courtesy of Anglo American plc.)

the fluid utilized for the separation process is determined by the specific gravity of the minerals and can be manufactured using many different components. The most widely used medium for metalliferous ores is ferrosilicon, an alloy of iron and silicon, because other classical medium such as heavy organic liquids are prohibited in industrial processes due to the toxicity.

The separation method that uses the different electrochemical surface features of fine-grained minerals is called froth flotation. It is certainly the most important method of concentration, not only for metallic minerals but also to separate different industrial minerals. Differences in the surface properties of particles control whether or not those particles will attach themselves to air bubbles (Fig. 6.6) within an agitated pulp. The particles that are so attached will rise to the surface of the medium and create a froth that can be removed as a concentrate. The rest of the particles remain in the pulp and flow out of the separator as the tailings stream.

If the magnetic properties of the valuable minerals are different from those of the gangue, they can be separated by the application of suitable magnetic fields in a magnetic separator. Consequently, magnetic concentration utilizes the different magnetic susceptibility of the minerals included in the mineralization being treated. This physical feature allows magnetic minerals to be split from nonmagnetic or less magnetic ones. Low-intensity magnetic separators can be utilized to beneficiate ferromagnetic minerals such as magnetite (Fe_3O_4), while high-intensity separators are applied to split paramagnetic minerals from their waste. Predictably, magnetic concentration is essential for processing iron ores and has also application in the processing of paramagnetic nonferrous minerals.

Electrostatic separation is based on the different ionization features between minerals subjected to an electric field. Thus, high-tension concentration depending on electrical conductivity characteristics can be utilized to separate conducting from nonconducting minerals. Electrostatic separation is very important because in theory this method depicts the universal concentrating method since most minerals display differences in their conductivity. Thus, it should be potential to concentrate almost any mineral by this process (Wills and Finch 2016). However, the method has fairly limited application, and its greatest utilization is in separating some of the minerals found in heavy sands from beach or stream placers.

In many cases, it is essential to combine different techniques to concentrate a mineralization economically. For instance, gravity methods are commonly utilized to exclude most of the gangue because they are relatively inexpensive. However, they do not present the necessary selectivity to generate the final clean concentrate. Consequently, gravity methods are generally combined with other methods (e.g., froth flotation) to obtain further upgrading in the concentrates.

Fig. 6.7 Semipermeable membrane for filtering (Image courtesy of TEFSA)

6.3.4 Dewatering

The previous mineral processing stages are commonly performed under wet conditions, utilizing water as a medium. Thus, the solids being processed are associated with large quantities of water or aqueous solutions. Since solids must be separated from water for metal production, the last process in mineral beneficiation is achieved by what are known as dewatering operations. They produce relatively dry concentrates at desired moisture content. The process is performed using thickeners and filters. The splitting of solids from liquid by gravity is easily carried out by continuous sedimentation procedures in thickeners. The problem is that the underflow still contains appreciable amount of water and further water removal is necessary. This is usually carried out by passing the slurry through a semipermeable membrane (Fig. 6.7) that is manufactured to retain the solids and allow the liquid to pass through (the membrane forms a screen). Once a «cake» is generated, permeability diminishes to stop the procedure. The filtering process is then finished where almost all the liquid has been extracted from the pulp and the filtered cake is removed from the filtering medium.

6.4 Particle Size and Size Distribution

Mineral processing methods are controlled by the particle behavior, which in turn change with its size. Thus, particle systems are essential in mineral processing because this engineering field works mostly with particles of different size, from run-of-mine to final concentrate. For this reason, size analysis is crucial to establish the efficiency of grinding and establish the degree of liberation of meaningful minerals between them as well as from the gangue minerals at different particles size. However, there are several problems connected with size control. The main issue is the fact that size reducing, as previously commented, is an expensive operation from an energy viewpoint, and final size distribution cannot be finer than what is needed for a perfect liberation process. Moreover, size analysis of the products in the separation stage is utilized to establish the optimum size of the feed process for maximum effectiveness. It also determines the range of sizes at which any losses are generating in the plant, so that they can be decreased. Quantification of the size properties of a particulate material is very difficult because the particles are generally so irregular that their size is no easy to define. With irregular particles, the quantitative description of particle size is always an approximation, and the basis chosen to represent size is largely a matter of convenience.

Unless a particle is spherical or cubic, determination of its size is never an exact process. Thus, regular shapes such as spheres, cubes, or tetrahedral can be described and quantified, but real particles very rarely fall into such categories, and they are most commonly described as irregular. One simple solution to this issue is to combine the effects of size and shape and to characterize particles in terms of an equivalent, simple shape (usually a sphere) with a given dimension. It is common to say that a particle behaves as though it were a sphere of diameter «d.» Although this is often a reasonable assumption, there can obviously be cases where it is not valid. In these instances, variations in particle shape would manifest themselves as apparent variations in size.

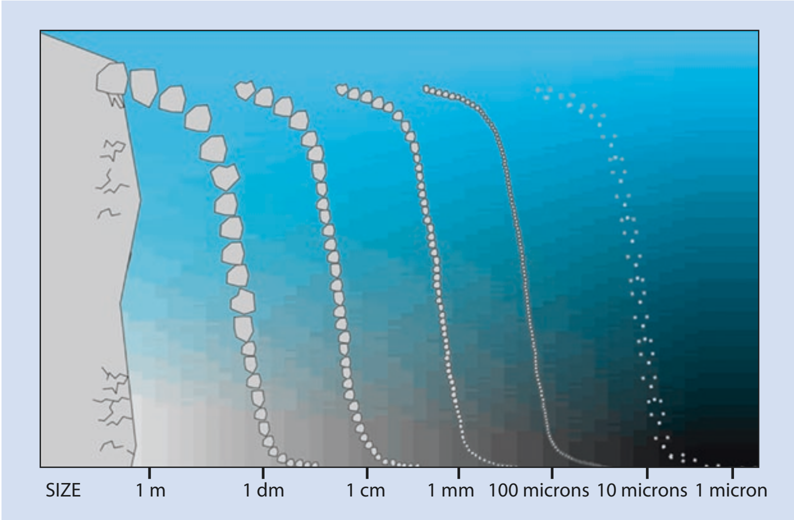


Fig. 6.8 Size particles in mineral processing, from 1 m to 1 µm (Illustration courtesy of Metso)

Because deviations from the spherical shape will have different effects on the response to distinct processes, size distribution estimates obtained for the same material but by diverse techniques cannot be expected to agree exactly, even in the absence of measurement error. Such discrepancies become especially important where more than one technique must be employed to span a broad range of sizes.

The size distribution of the particles must be controlled at various stages of a mineral processing plant for a number of reasons: (a) to allow undersized material to bypass the crushing or grinding system and to maintain oversized components for further size reduction, (b) to generate an optimum particle size material for effective operating in the downstream concentration circuits, and (c) to provide a product that meets particle size standards needed for the industry (Kelly and Spottiswood 1982). Moreover, depending on the type of mineralization, the meaningful mineral or the gangue mineral can be concentrated in particular size classes. In these cases, downstream processing exploits such phenomena. It is essential to note that the range of sizes in a single process stream is typically very large and can include particles that vary in diameter from 1 m to 1 µm (10^{-6} m) (Fig. 6.8).

Because of the large number of particles involved, it is neither practical nor useful to consider their sizes on an individual basis. Instead, statistical principles are employed. The best approach is the grouping of particles into classes based on their size. The entire population of particles can be split into a finite number of classes, each class being defined by a different set of class boundaries. The distribution of particles according to their size can then be indicated as the proportion of the particle population that is found in each size class. An important feature of the concept of particle classes is that it makes possible a significant simplification in the quantitative evaluation of the behavior of particulate material. This is based on the assumption that every particle in a given class behaves in the same way. Obviously, this assumption is not absolutely the truth but becomes more accurate as the range of sizes covered by the class is narrower.

Despite the obvious importance of particle size, the evaluation and even precise definition of particle size are far from simple tasks. There are many ways to characterize the size of particles, none of them being perfect. For instance, «particle size can be characterized by determining the size of hole or aperture the particle will just pass through (sieve size), or the time the particle takes to settle in a fluid such as water and express the particle size as the size of a sphere that has the same settling rate (Stoke's diameter)» (Gupta and Yan 2006). In selecting a sizing method, consideration should be given to matching the method to the particular application for which the size information is desired. Thus, if it is necessary to characterize the particles in a liquid suspension, it would try to use a method that evaluates the behavior of the particles in a liquid, for example, a sedimentation method.

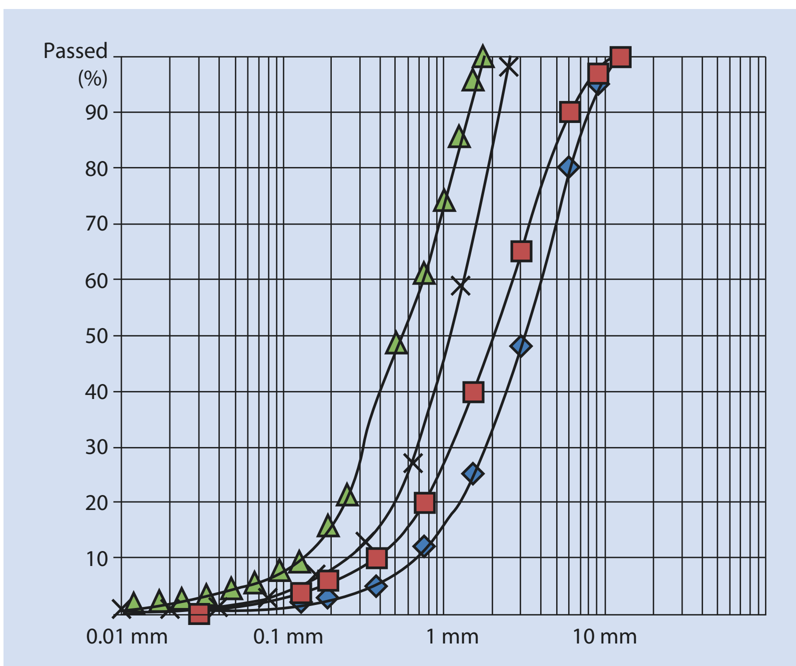
There is a broad range of methods of particle size analysis forthcoming in the market, being test sieving the most widely used. It covers a very wide range of particle sizes, the one of most industrial importance. Test sieving is used so much in size analysis that particles finer than approximately 75 µm are commonly mentioned to being in the «subsieve» range. However, modern sieving techniques enable sizing to be carried out down to about 5 µm (Wills and Finch 2016). Subsieving methods include mainly sedimentation, elutriation, microscopy, and laser diffraction.

Sieve analysis is carried out by passing a known weight of sample successively through a set of screens whose apertures correspond to the class boundaries and weighting the quantity of sampled retained on each sieve, thus calculating the percentage weight in each size fraction. A good degree of standardization has been established in the industry with regard to the sieve apertures used in measuring and describing size distributions. Test sieves are classified in accordance with nominal aperture size, being square apertures the most generally used. In respect to the presentation of data, there are different ways, but the most common method to represent graphically the results is plotting cumulative undersize (or oversize) against particle size (Fig. 6.9).

Sedimentation methods of analyzing particle sizes are based on the estimation of the rate of settling of the fine particles evenly dispersed in a fluid. For these methods, the particle size should be greater than 1 µm. This technique separates the particles according of the resistance to motion in a fluid. This resistance establishes the terminal velocity that the particle attains as it is enabled to fall in a fluid under the influence of gravity. The terminal velocity is derived from Stoke's law for spherical particles. Size distributions are measurable with these methods, and both large and small amounts of sized material are obtainable.

Microscopy methods or, more recently, laser measurements have been also utilized to calculate statistical sizes of particles. Microscopy methods are highly attractive since they involve direct observation of the particles and, through the combination of optical and electron microscopes, are virtually unlimited with respect to size. Consequently, they are extremely useful for qualitative and semiquantitative assessment of the average size and approximate range of sizes present in a distribution. However, they are not recommended for the quantitative evaluation of size distributions, especially for materials in which a broad range of sizes is present (Hogg 2003). It is essential to comment the measurement method utilized when quoting particle size because all of these methods do not indispensably offer the same results. Regarding laser measurements, different instruments relying on the diffraction of

Fig. 6.9 Sieving results plotted graphically



laser light by fine particles are available. In this method, particle size distributions are calculated by measuring the angular variation in intensity of light scattered as a laser beam passes through a dispersed particulate sample.

6.5 Ore Handling

According to Wills and Finch (2016), «ore handling is a key function in mining and mineral processing, which can account for 30–60% of the total delivered price of raw materials.» It covers mainly the processes of transportation between the mine and the mineral processing plant and between the various stages of treatment in the plant. Run-of-mine is transported from the mine to the beneficiation plant by different methods depending on the size of operation and the distance between the mine and the plant. In case of long-distance transportation (e.g., several kilometers or more), it is done commonly by trucks. If the distance is short (e.g., 100–200 m), the standard rubber belt conveyor with support rollers at the bottom of belt is the most broadly utilized method of handling bulk material. Belts today have capacities up to 40,000 t/h, and advances in control technology have enhanced the reliability of belt systems. Regarding the transportation inside the mineral processing plant, for example, between the different comminution equipment, the belt conveyor system is usually the suitable method.

Ore handling can also include ore storage and feeding. Ore storage is a continuous operation that runs 24 h a day and 7 days a week. The type and location of the material storage depends primarily on the feeding system. Grinding and concentration processes are most effective when working continuously. On the other hand, storage has also the advantage of enabling the mixing of mineralization with different grades with the objective to generate a continuous and similar feed to the mill.

Depending on the nature of the material treated, storage is accomplished in stockpiles, bins, or tanks, stockpile being one of the most classical methods of storage. Stockpiles are usually used for ore that has passed through the primary crushing stage and sometimes also for coarse materials (e.g., coal or iron mineralization). Several methods are used to constitute a stockpile obtaining shapes such as conical, elongated, or radial. Reclaiming from stockpiles can be performed using bottom tunnels, bucket wheel machines, front-end loaders, etc. It is important to note that intense segregation often occurs in stockpiles, being minimized by the suitable selection of reclaiming equipment.

Regarding the feeding process, feeders (Fig. 6.10) are essential if a uniform stream of mineralization is needed. This is because mineralization does

Fig. 6.10 Vibrating feeder in an aggregate quarry (Image courtesy of Marcelino Martínez)

not usually flow uniformly from a storage, except where some type of mechanism controls it. Feeding is basically a conveying process in which the travel distance is low and in which correct control of the rate of passage is needed. When main operations are disrupted by a storage sage, it is indispensable to include a feeder in the circuit. Feeders also reduce wear and tear, abrasion, and segregation. There are many types of feeders in the market such as apron, belt, chain, roller, rotary, revolving disk, drum, drag scraper, screw, vane, reciprocating plate, table, and vibrating feeders (Wills and Finch 2016). Factors like type of material to be handled or the storage method govern the type of feeder.

6.6 Comminution

Comminution is the global term used to describe the progressive reduction in size of run-of-mine ore. The main objective of comminution or particle size reduction is to maximize the liberation of the mineral from the host rock. It is a process whereby particulate materials are reduced to the product sizes required for downstream processing or end use. In fact, the comminution procedure starts during the mining process, being blasting of in situ material the first stage. Thus, the ore body is reduced from its natural size, which can be a kilometer or more in extent, down to run-of-mine material containing fragments measuring up to a meter or more. Comminution is also accomplished to make the recent excavated material easy to handle by excavators or scrapers to generate a run-of-mine material transportable by haul trucks or conveyors. Mosher (2011) suggests that the degree of separation carried out by comminution establishes the grade-recovery curve for a given concentration procedure as well as classically represents the largest type of mineral processing capital and operating expenditures.

6.6.1 Mechanisms of Fracture

Particle fragmentation is accomplished by the application of force. The manner in which force is applied to the material to be broken determines both the nature of the comminution device that is used and the size ranges of the particles produced. Thus, the method selected for applying the breakage forces depends on the size to which the material must be broken. For a particle to fracture, a stress high enough to overcome the fracture strength of the particle is needed. Breakage is achieved mainly by crushing, impact, and attrition processes, being the three modes of fracture (compressive, tensile, and shear) (Fig. 6.11) distinguished based on the rock mechanics and the type of loading. Obviously, the nature of the

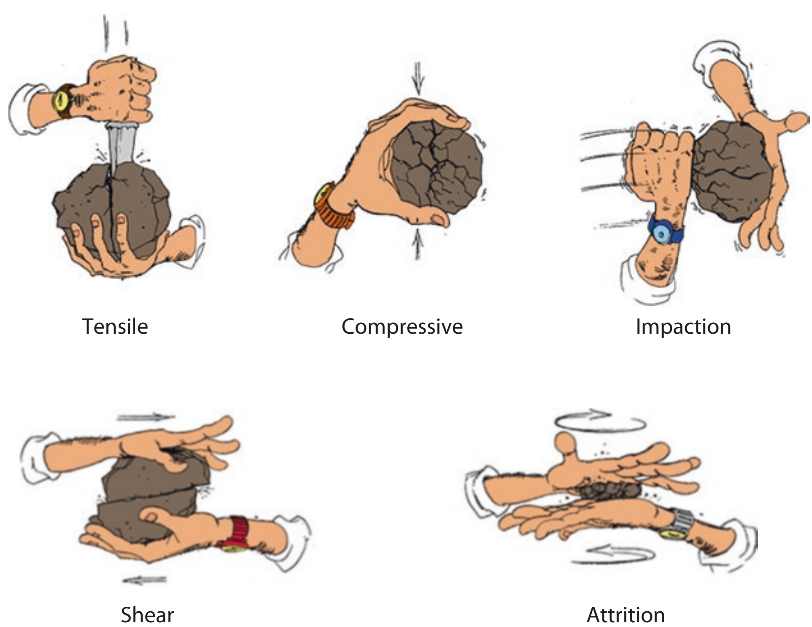


Fig. 6.11 Types of fracture and breakage (Image courtesy of Metso)

fracture products is different in each case. In practice, these fracture mechanisms do not take place in isolation, rather breakages require a combination of fracture mechanisms. However, comminution machines are usually constructed in such a way that one kind of breakage prevails. The mechanism that will predominate depends on the size of the components and the mechanical configuration of the comminution device used.

In breakage by compression, at least two crushing surfaces are required to apply compressive forces to a particle, either directly or indirectly through a bed of particles. The products of fracture are a small number of relatively large fragments resulting from the induced tensile fracture and a large number of small fragments that originate predominantly from the points of application of the compressive forces. The amount of fines produced can be reduced by minimizing the area of loading. This is often done in compressive crushing machines by using corrugated crushing surfaces.

In impact breakage, there is a rapid transfer of energy to the particle where a particle and a rigid surface or other particles undergo impact. If the energy absorbed is significantly greater than that required for simple fracture, the particle will shatter into a large number of intermediate-sized and very small fragments mainly by tensile failure. Thus, fracture by shatter takes place where the applied energy is in excess of that needed for fracture. The products of fracture in this situation are smaller than in the case with compressive fracture, and the production of fine particles is more extensive.

Regarding the breakage by attrition, the forces are applied to the surface of the particle by a process of abrasion between a particle and a breakage surface or other particles. The products of this mechanism are the original particle smoothed together with a mass of very fine particles. Attrition or abrasion is not strictly a breakage event but rather a surface phenomenon where shear stress causes a material to abrade off. In general, the procedures of size reduction during crushing and grinding are distinct. In crushing operations, the size reduction is carried out mainly by compression of the rock against a rigid surface while grinding includes principally abrasion of the rock by the grinding media. The results of a single particle breakage event are not fully predictable because of the extremely large number of variables that affect the outcome.

6.6.2 Energy for Size Reduction

Size reduction theory is devoted to the relationship between the quantity of energy put into a rock of known size and the particle size after the comminution process has taken place. It is very important to remember that the process remains inherently inefficient because 85% of the energy used in comminution is dissipated as heat, 12% is attributed to mechanical losses, and only 3% of the total energy input is used in size reduction of feed material (Alvarado et al. 1998). In this sense, higher use of blasting energy at its better effectiveness can produce a significant impact at lowering the comminution energy in the crushing and grinding circuit downstream (Murr et al. 2015). Moreover, «mine-to-Mill optimization in various operations over the years have shown significant benefits such as high mill throughput rates from reduced top size from mining through increased powder factor or blast energies» (Kanchibotla 2014). The coalition for Eco-Efficient Comminution (CEEC) has been lately carried out with assistance from different companies in the mining industry. The goal of CEEC is to promote awareness and know-how transfer principally to decrease energy consumptions in comminution.

Improved methods for mineral comminution such as SELFRAG technology are continuously being established, pursuing the objective of accomplishing the needed size reducing and mineral liberation at a lower energy consumption level than traditional technology enables. SELFRAG technology permits for monitored crushing process without contamination due to a combination of pulse power technology, physical (electrical) material discontinuities, and high voltage and mechanical engineering skills. Typical applications of this technology are mainly situated in industrial minerals (e.g., quartz) or metallic mineralization.

The selected product size also aids importantly to the energy intensity of comminution process. As the material size reduces, the energy needed to accomplish the product size increments greatly. This is because strength and therefore resistance to breakage of particles increases as the particles become smaller. On the other hand, the consumption of steel elements is another aspect of comminution that aids importantly to the energy consumption of the process.

Various theories have been developed to explain the relationship between energy input and the particle size obtained from a given feed size, but none of them is completely successful. The greatest issue is that most of the energy input to a crushing or grinding device is absorbed by the machine itself, and only a small fraction of the total energy is suitable for fracturing the rock (Wills and Finch 2016). Theoretical and empirical energy size reduction equations were proposed by Von Rittinger (1867), Kick (1885), and Bond (1952), known as the three theories of comminution. The oldest theory of Von Rittinger (1867) states that «the energy consumed in the size reduction is proportional to the area of the new surface produced.» The second theory (Kick 1885) stated «that the work required is proportional to the reduction in volume of the particles concerned.» Regarding the third and most used theory, Bond (1952) presented an equation based on the theory showing that the work input is proportional to the new crack tip length produced in particle breakage and equals the work represented by the product minus that represented by the feed (Box 6.2: Bond's theory).

Box 6.2

Bond's Theory

Bond's theory contains elements of both the Von Rittinger and Kick theories in that it assumes that the energy actually used in crushing and grinding is proportional to the length of the extension of the crack tips. Bond methodology is simple, and it does work for many circumstances to a reasonable degree of accuracy. In the design of grinding circuits in a mineral processing plant, the Bond method is widely used for a particular material in dimensioning mills, determining power/energy required, and evaluating performance. Its use as an industrial standard is very common, providing satisfactory results in all industrial applications. Despite having many advantages, this method has some drawbacks such as being tedious and time-consuming and also requiring a special standard mill (Saeidi et al. 2013). Bond's formula is useful because it indicates the energy input required to produce a desired degree of size reduction. The universally accepted formula of Bond is the following:

$$W = 10 \times \mathrm{Wi}\left(\frac{1}{\sqrt{P_{80}}} - \frac{1}{\sqrt{F_{80}}}\right)$$

where W is the energy required for comminution per ton of material broken and P_{80} and F_{80} are, respectively, the screen sizes in µm through which 80% of the material in the product and feed will pass. Wi is the Bond Work Index, which is a property of the material being broken. It is an expression of the resistance that a material has to undergo crushing or grinding or a measurement of how hard the ore is. Thus, each rock has a characteristic work index. For example, more energy is required to break quartz than calcite, so quartz will have a higher work index. Numerically, the work index is expressed as the kilowatt hours per ton (or per short ton in Bond's original publication) required to reduce the material from theoretically infinite feed size to 80% passing 100 µm. If the work index of the ore is known and the feed size and the desired product size of the ore are also known, then the power required for crushing the ore can be estimated.

Bond Work Index is determined experimentally by measurement of the extent of size reduction and the power absorbed in particle breakage under well-defined standard conditions. Determining Wi via the standard Bond method needs careful grinding cycles and screen procedures. However, these are tedious time-consuming procedures with potential errors in sieving steps. Considering the difficulties of the standard Bond method in the course of determining Wi, a number of simpler and faster alternative methods have been developed (e.g., Gharehgheshlagh 2016).

If the breakage characteristics of a material remain constant overall size ranges, then the calculated work index would be expected to remain constant since it expresses the resistance of material to breakage. However, for most naturally occurring raw materials, differences exist in the breakage characteristics depending on particle size, which can result in variations in the work index. Thus, calculations involving Bond Work Index are generally divided into steps with a different Wi determination for each size class. Laboratory tests for estimating work index for grinding circuits are Bond Ball Mill Grindability Test, Bond Low Energy Impact Test, and Bond Rod Mill Grindability Test. Although the Bond model remains the most widely used, it has a number of deficiencies. As a result, the conditions under which the index is measured are invariably not entirely applicable to the conditions that exist in operation plants. Consequently, the predictions of the Bond formula need to be modified by a variety of empirical correction factors. For this reason, other tests such as drop weight test, SPI and SGI tests, and SAGDesign test are designed for ore characterization.

6.6.3 Comminution Stages

As aforementioned, in mineral processing terminology, comminution in coarse size is termed crushing, whereas comminution in fine size is termed grinding. Thus, crushing is performed on large particles, while grinding deals with particles smaller than 50 mm. The crushing and grinding processes generate a group of particles with different degrees of liberation. Crushing decreases the particle size of run-of-mine mineralization to such a level that further grinding can be performed until the valuable components and gangue are significantly produced as separate elements. Any particle that overcomes a certain size needed for mineral beneficiation is returned to the crushing or grinding circuit. Accordingly, most comminution procedures in industrial applications are always closed circuit excepting primary crushing. A comminution circuit is termed closed circuit where it works in series including a size classifier, being recirculated to the comminution device the coarse fraction of the classifier.

Crushing

Crushing is commonly a dry operation, and it can be carried out in several stages (e.g., primary, secondary, and tertiary crushing), although the first stage of breakage is the only crushing step always present in modern comminution circuits. The reason for two or three stages of crushing is because of the limited reduction ratio that it is possible to achieve with a single crusher. The reduction ratio of a crushing stage can be usually defined as «the ratio of maximum particle size entering to maximum particle size leaving the crusher.» The nature of the crushing environment changes from stage to stage, particularly from the primary to secondary stage. In the primary crushing stage, pieces of run-of-mine mineralization can be as large as 1.5 m, being reduced to a range between 10 and 20 cm by using heavy-duty machines (Fig. 6.12). The emphasis at this stage is on the reduction of a very large material to more manageable sizes. The feed to the crusher is often delivered at irregular intervals.

The crushed ore normally goes to the grinding circuit via a belt conveyor after it leaves the primary crusher. In other cases, it can pass through a screen or other size classifier with oversized material circulated back into the crusher(s) for further reduction. If present, secondary crushing involves all procedures for treating the primary crusher materials from rock storage to the disposal of the final crusher product. In this stage, the size range of the

Fig. 6.12 Primary crushing in an underground mine (South Africa) (Image courtesy of Petra Diamonds)

feed material is more uniform, and the feed rate is more regular putting more emphasis on the optimization of the crushing efficiency. A third reduction step called tertiary crushing can be utilized if the ore is extremely hard or in particular cases where it is essential to diminish the generation of fines. However, tertiary crushing stage is often substituted by coarse grinding, especially if the ore tends to be slippery and tough. Tertiary crushers are basically of the same design as secondary crushers.

Primary crushers are commonly located adjacent to the surface or underground mine to ensure efficient transport of the ore, being jaw and gyratory type crushers the most used devices. They are defined by a wide input and a narrow discharge and can operate large tonnage of material. Jaw crushers generate a reduction ratio between 4:1 and 9:1, while gyratory crushers originate a somewhat larger range between 3:1 and 10:1. In general, gyratory crushers can generally process larger rock pieces and more tonnage per hour than jaw crushers.

Once the run-of-mine mineralization is reduced to smaller components by using the primary crushing units, the secondary machines are utilized to obtain a further decrease in size. These secondary units are generally much lighter and smaller than the heavy-duty and big rugged primary units. Since they use the primary crushed rock as feed, the maximum feed size will commonly be less than 15 cm in diameter. Secondary crushers are much easier to handle because most of the harmful components in the mineralization (e.g., tramp metal, wood, clay, among others) have yet been extracted from the circuit (Fig. 6.13). They work always with dry elements, and their objective is to decrease the mineralization to a size appropriate for further grinding. Examples of secondary crushers are cone crusher and impact crusher. Cone crushers generate reduction ratios ranging from 5:1 to 8:1. Very high reduction ratios, for example, from 20:1 to 40:1, can only be obtained utilizing impact crushers. At the end of the crushing stage, the ore is collected in stockpiles that have openings or drawpoints at the bottom. Ore will be drawn out from the piles to the grinding circuit as needed. Stockpiling crushed ore ensures that grinding circuit can be continuously fed, given that there can be variable operating

Fig. 6.13 Metal detection (Image courtesy of Antonio Durán)

Fig. 6.14 Different ball sizes

times or production rates in different parts of the mine and plant.

Grinding

The grinding stage takes crushed ore from the stockpile generated in the crushing process and reduces it to a finer size for beneficiation process (e.g., froth flotation). In this step, the components are decreased in size by a combination of impact and abrasion, either dry or in suspension in water. Most industrial grinding circuits are operated under wet conditions. The final ground size depends on the composition of the mineralization and is elected to maximize the recovery of the valuable minerals. Excessive grinding (overgrinding) should be avoided because its high cost and fine grinding usually make further upgrading difficult.

Grinding is carried out in rotating cylindrical steel vessels that include a charge of crushing components (the grinding medium). This charge is free to move within the vessel, decreasing the size of the particles. The mill shell rotates at a constant speed, and the ore is ground by the interaction of the ore particles with each other and/or with steel balls or rods that are added to the mill. In contrast to crushing, which takes place between fairly rigid surfaces, grinding is a random process because the level of grinding of a mineralization particle is based on the probability of the particle entering a zone between the grinding medium (e.g., balls) and the probability of some breakage event occurrence after entry.

The feed particles in tumbling mills range between 5 and 250 mm, and they are decreased in size to between 40 and 300 µm. The grinding medium can be steel rods, steel balls, or even rock itself. Ball sizes (Fig. 6.14) usually range from about 20 mm for fine grinding to 150 mm for coarse grinding. Although there are a lot of improvements in these machines, the energy effectiveness of tumbling mills is the subject of intense debate.

Obviously, it is essential that the ore is ground to the correct size for further separation of valuable minerals. Ore size is controlled by limiting the time the ore spends in each mill and by the use of a screen or other classifier. Undersized material will move to the next stage in the process, and oversized material will be recirculated back to the same mill for further grinding. Although to perform a certain particle size related to the degree of liberation is the main objective of grinding, it is occasionally utilized to increment mineral surface area. For instance, production of some industrial minerals such as talc involves size reduction to meet customer requirements. There are different variables that can be controlled to achieving this goal. Mill charge, feed rate, and pulp density, among others, can be controlled and set to the needed value to assure smooth operation and obtain output of requested specifications.

Regarding ultrafine grinding, the emergence of this technology is a solution for operating low-grade ores with complex mineralogy. This is because it is necessary to grind mineralization to as low as 5–7 µm to allow enough liberation for an effective separation. Ultrafine grinding is also usual for regrinding flotation concentrates and preparing feed for hydrometallurgical processes. Ultrafine grinding is even an obligation in certain industries (e.g., mica generated for the paint industry must be ground to below 10 µm). There are nowadays a broad range of ultrafine grinding machines such as stirred mills.

6.6.4 Comminution Equipment

The design needs of size reducing equipment change continuously as the particle size changes. Thus, the difference between devices is related mainly to the mechanical features of applying the force (compression and/or impact) to the various sizes of particles. Accordingly, the following machines and their main characteristics are numbered according to their crushing or grinding objectives because each of them works on a determined size range.

Jaw Crushers

A crusher consists of a crushing chamber in which compressive or impactive forces are brought to bear on the material to be broken. This material, which can be dry or have a low moisture content, is introduced through a feed opening and falls into the crushing chamber. Then, it is broken and falls out through a discharge opening. The breakage forces are applied through crushing surfaces, which can be either stationary or moving in a rigidly constrained path.

Primary crushers are rugged devices that operate dry run-of-mine feed rock as large as 1 m or even more. There are two principal types of primary crushers: jaw and gyratory crushers. Both crushers generate fracture by compression since this is the easiest procedure of applying a breakage force to big particles. A constraint that results from the crushing action in compressive crushers is a limit on the effective reduction achieved by a given device. In practice, it is not possible to design machines with the required mechanical strength that will be able to grip a particle and break it down into fragments all

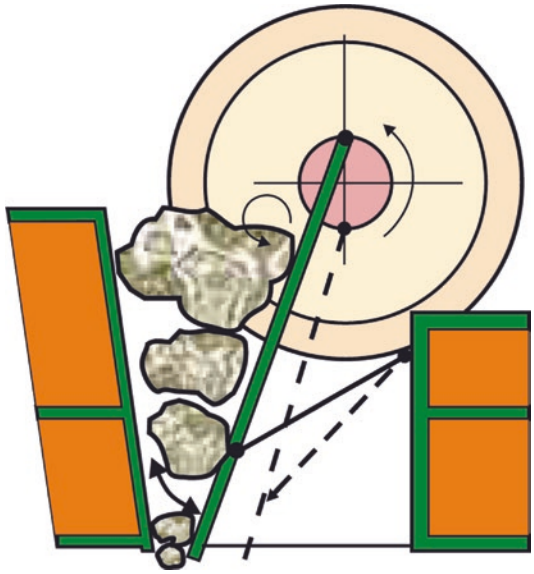


Fig. 6.15 Jaw crusher (Illustration courtesy of Metso)

smaller than about one-seventh of the original particle size. Thus, low reduction ratios are achieved in compression crushers. These values range in a jaw crusher between 4:1 and 7:1 with an average of 6:1.

In jaw crushers, the material (medium hard to very hard rock) is size reduced by dropping into a «V»-shaped space created between two surfaces (Fig. 6.15). One of them is stationary, whereas the second surface oscillates between two extreme positions. As the second surface moves toward the stationary surface, oversize particles are caught and broken. Fragments of breakage remain in the crushing chamber until they are sufficiently small to fall through the gap between the two surfaces. The gap of the equipment determines the size of the material that can be fed to jaw crushers. The crushing action in a jaw crusher is intermittent, and power is drawn only during half of the crushing cycle. The faces of the plates are made of hardened steel, and the surfaces of both plates could be plain or corrugated.

Jaw crushers are installed underground as well as on the surface (Fig. 6.16). Where utilized underground, they are usually disposed in open circuit. The material obtained is then reduced in size in crushers situated on the surface. Where the run-of-mine material is transported directly from the mine to the crusher, the feed to the primary crusher go through a magnet to extract steel components collected during the mining operation.

Fig. 6.16 Primary jaw crusher with a hammer on the surface

Gyratory Crushers

Gyratory crushers are the most common equipment for new crushing operations (Herbst et al. 2003). They are only used as primary crushers in surface-crushing plants, whereas jaw crushers are sometimes used also for secondary applications. During the last 20 years, gyratory crushers have not changed in size substantially, although their consumed horsepower has increment enabling these devices to operate higher throughputs (Gorain 2016). Gyratory crushers break material in a very similar way than jaw crushers. The crushing surfaces are curved and conical, and the crushing bowl provides the fixed surface. Located inside this bowl is the conical crushing head, which nutates about a pivot point. This action is provided by some form of eccentric drive applied to the base of the crushing head. The eccentric drive generates the opening and closing of the gap. The entire assembly can be visualized as a circular jaw crusher. The crushing process comprises reduction by compression between two confining faces and a subsequent freeing movement during which the material settles by gravity until it is caught and subjected to further compression and again released.

The feed material (from hard and abrasive to soft and sticky) that enters into the crushing chamber from the top is repeatedly crushed between the crushing elements as it falls by gravitation until it leaves the crushing chamber at the bottom. The particles are subjected to maximum breaking forces where they are on the side with the minimum gape. This equipment handles feed material as large as 1.5 m or more to produce discharged fragments as small as 125 mm. Thus, common reduction ratios can be about 10:1. Gyratory crushers (Fig. 6.17) tolerate different types of shapes of feed particles, including slabby rock, which are not possible to incorporate in jaw crushers due to the shape of the feed opening (Gupta and Yan 2006).

In deciding whether a jaw or a gyratory crusher should be elected, gyratory crushers are utilized more significantly than the capacity. For instance, if it is needed to crush rock of a determined maximum diameter, then a gyratory with the needed gape would have a capacity about three times that of a jaw crusher of the same gape. However, if a big gape is required but capacity is not important, then the jaw crusher will be possible for the selection since it is more economical due to its smaller size. Secondary considerations in the selection process include capital and maintenance costs and the type of rock being crushed. For the latter, jaw crushers work better with clay or plastic materials, while gyratory crushers are especially appropriate for hard, abrasive rocks. Moreover, gyratory crushers commonly generate a more cubic product than jaw crushers.

■ Fig. 6.17 Primary gyratory crusher (Image courtesy of Metso)

Cone Crushers

Secondary crushers are utilized to treat the primary crusher material, which is around 15 cm in diameter. This material is decreased in size up to between 0.5 and 2 cm in diameter so that it is appropriate for further grinding. Secondary crushers are comparatively lighter and smaller than primary crushers, operating commonly with dry clean feed exempted of harmful components. Cone crushers (■ Fig. 6.18) usually perform the bulk of secondary crushing of metalliferous ores. The cone crusher is a modified gyratory crusher, but the eccentric move of the inner crushing cone is similar to that of the gyratory crusher (■ Fig. 6.17). The essential differences are two: (a) it has a much shorter spindle with a large diameter crushing surface relative to its vertical dimension; and (b) the shorter spindle is not suspended, as in the gyratory, but is supported in a curved, universal bearing below the gyratory head or cone (Haldar 2013). The cone crusher has higher capacity than the gyratory crusher since it presents a greater head angle than in the gyratory crusher. Cone crushers were initially developed by Symons around 1920 and therefore are many times termed as «Symon cone crushers.»

■ Fig. 6.18 Cone crusher in an aggregates processing plant (Image courtesy of Benito Arnó e Hijos S.A.U.)

With the addition of semiautogenous milling circuits, cone crushers are increasingly being utilized for pebbles crushing. This design of big cone crushers puts emphasis on incrementing capability without important increasing operating costs along with lower maintenance and high readiness (Gorain 2016). Cone crushers work as tertiary crushers when installed in close circuit between the secondary unit and the ball mill to crush and overflow material of vibratory screening. Several specialized forms of cone crusher such as Gyradisc crusher or Rhodax crusher are developed in the market. The former is mainly utilized for generating finer material, which has found application in the quarrying industry due to its lower economic costs.

Roll Crushers

Roll crushers are designed with one or more cylindrical rolls located parallel to each other and rotating in opposite directions. The rotation of the rolls draws material into the breakage zone where particles are caught, either against the breaker plate or between two rolls, and so experience compressive forces. In single-roll crushers, the crushing chamber consists of one rotating roll and a stationary breaker plate; in double-roll crushers, two rolls rotate toward each other; three-roll and four-roll crushers consist of two stages of roll crushing built into a single machine. Unlike jaw and gyratory crushers, the crushing process in rolls is one single pressure. The roll surfaces play an important role in the process of catching an element and dragging it between the rolls. Thus, the rolls can be smoothed or toothed with pyramidal tooth. With smooth rolls the reduction ratio is about 3:1, whereas this ratio can be increased up to 7:1 where toothed rolls are used. Smooth surfaces are commonly used for fine crushing, while coarse crushing is generally carried out in rolls with tooth.

Roll crushers are mainly used in secondary or tertiary crushing applications. Although not widely used in the minerals industry, roll crushers can be effective in handling friable, frozen, and less abrasive feeds such as limestone, coal, gypsum, phosphate, and other soft ores (Wills and Finch 2016). One of the advantages of roll crushers is that they exert a more positive control on the top size of the crushed product than is possible with jaw and gyratory-type crushers. The main issue of roll crushers is that they have the highest capital cost of all crushers for a given throughput and reduction ratio.

The pressure practiced on the feed particles in classical roll crushers ranges from 10 to 30 MPa. If the crushing pressure increases hydraulically up to 150 MPa or more, the units are commonly termed high-pressure grinding rolls (HPGR) (◘ Fig. 6.19). This equipment was firstly used in the early 1980s in the limestone and cement industries as an energy-effective alternative to ball milling, demonstrating about 25% reduction in energy consumption for grinding limestone and cement clinker. It was introduced into the mining industry also in the 1980s to crush kimberlite mineralization in diamond operations, and it was recently incorporated in different hard rock metal mineralization such as iron ore, tin, copper, chrome, and gold (Erickson 2014).

Comminution in an HPGR is carried out almost absolutely by compression, being the result of a product with higher percentage of fines that can be obtained using a semiautogenous or autogenous mill. Thus, the principal use of HPGR is as a replacement for the classical semiautogenous mill in a grinding circuit. It produces important savings in energy cost and decreased grinding media consumption and operating cost.

Impact Crushers

In this class of crushers, the size reduction is performed by high-speed impact rather than compression. Impact crushers have a broader utilization in the quarrying industry than in the metal mining industry. Moreover, they are also encouraged in the quarry industry due to the improved product shape. There are two main impact crusher equipments: hammer mills (◘ Fig. 6.20) and impact mills. Particles in the product of an impact crusher tend to be more cubic in form than in a hammer mill. This factor can be significant where the crushed material is to be used for construction purposes. In hammer mills, which are falling into disuse in the last decades, a series of hammers is mounted on the rotor and operate at speeds between 2000 and 6000 rpm. Much of the comminution in a hammer mill is the result of attrition because the particles are given high velocities. This generates a much higher proportion of fines than with compressive crushers. The product size can be extremely fine; for example, talc can be reduced to a size of 2 μm.

Regarding the impact mill or impactor (◘ Fig. 6.21), this equipment is often used for coarse crushing. Impact crushers change mainly in the design of the impactors (they may be arranged onto a roll or as a series of hammers) and in the nature of the crushing chamber (plates or bars), being the bars mounted rigidly on a

Fig. 6.19 High-pressure grinding rolls (HPGR) (Image courtesy of Metso)

rotor. The feed material falls tangentially onto a rotor running at 250–500 rpm and is shattered by multiple blows from the impact bars. The particles undergo secondary breaking where they impact against the stationary breaker plates that line the crusher chamber. Large impact crushers will decrease 1 m top size run-of-mine mineralization to 20 mm; consequently high-reduction ratios (40:1) can be achieved. Since the impactor depends on high velocities for crushing, wear is greater than for jaw or gyratory crushers. Thus, major problems associated with operating impact crushers are the high wear rates of the liners or impellers, especially where handling abrasive feeds. Hence, these devices are not recommended for utilization in mineralization including over 15% silica (Woollacott and Eric 1994).

Tumbling Grinding Mills

The final comminution stage is commonly carried out in tumbling mills, cylindrical devices where the particle size is decreased combining impact and abrasion forces. The first difference between the mills is in the ratio diameter/length

Fig. 6.20 Hammer mill (Image courtesy of Octavio de Lera)

Fig. 6.21 Inside an impactor (Image courtesy of Benito Arnó e Hijos S.A.U.)

of the cylinder and the features of grinding media employed: steel rods, steel (Fig. 6.14) or ceramic balls, hard pebbles, or particles of the ore itself; thus, the mill is classified accordingly. The process is performed commonly in water although dry option can be also possible. In this case, it is employed only where the downstream processing is to be conducted necessarily on dry material (e.g., in cement industry). Water is added with the feed material to make a slurry with solid concentration of about 50% by volume. The mill load (grinding medium and the material to be ground) occupies between 45 and 50% of its volume. The grinding mill decreases feed particles of 5–20 mm to components with sizes ranging from 40 to 200 μm as needed for further beneficiation.

The tumbling action in a mill is brought by rotation of the mill on its axis, being the grinding operation continuous. The mill load is carried upward by the motion of the mill until the force of gravity causes the load to fall away from the mill shell and to tumble over itself. At low rotational speeds, the components of the grinding medium remain in contact with one another, and the tumbling action is referred as cascading (Fig. 6.22). In this case, the dominant size reduction mechanism is attrition; this motion generates fines and must be minimized as possible. At higher mill speeds, the components of the grinding medium begin to be ejected from the main body of the load, and the process is termed cataracting; this motion produces less amount of fines. At higher speeds still, the load in the mill begins to centrifuge. The speed at which the load just begins to centrifuge

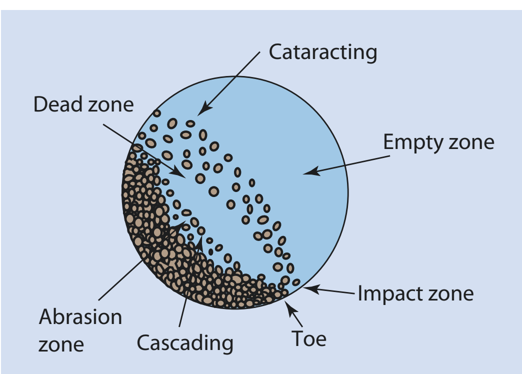


Fig. 6.22 Tumbling action

is known as the critical speed of the mill, being an essential parameter of the mill. In general, mills operate at a fixed speed between 60 and 92% of critical speed because beyond the critical speed the particles remain centrifuged at the wall and no grinding action is performed.

Rod Mills

Rod mills can be defined as fine crushers or coarse grinding units. Grinding is achieved as the rods roll over one another, nipping and crushing particles between them. Rod mills have the important property that they minimize overgrinding because the rods tend to be held apart by the largest particles so that they do not break small particles efficiently. The size feed is up to 50 mm, and the final product is as fine as 300 μ. Thus, reduction ratios are commonly between 15:1 and 20:1. Rod diameters (Fig. 6.23) vary between about 50 and 125 mm, and the length of the cylindrical shell is between 1.5 and 2.5 times its diameter. As in ball mills, steel grinding media are preferred because of their high density and relatively low cost; the higher the density of the grinding medium, the higher the grinding capacity of the mill. The major disadvantage in the use of steel grinding media is the cost associated with the steel consumption. This forms a significant part of the operation costs of a rod or ball mill operation.

Optimum grinding rates are obtained in rod mills with about 45% charge filling. Overcharging generates ineffective grinding and augmented liner and rod consumption. The rod consumption is usually between 0.1 and 1.0 kg of steel per ton of mineralization for wet grinding, being less for dry grinding. Rod mills commonly run at between 50 and 60% of the critical speed, so that the rods cascade rather than cataract. Since rod mills give a product of relatively narrow size range, they are appropriate to prepare the material that is fed in a gravity concentration circuit or to some flotation systems. These devices always run in open circuit because of their controlled size reduction.

Ball Mills

The ball mill uses steel balls as grinding medium (Figs. 6.15 and 6.24). The balls fall onto the ore and break it down while the mill rotates. The grinding material in the mill can be steel balls up

Fig. 6.23 Rods for a rod mill (Image courtesy of Daytal Resources Spain S.L.)

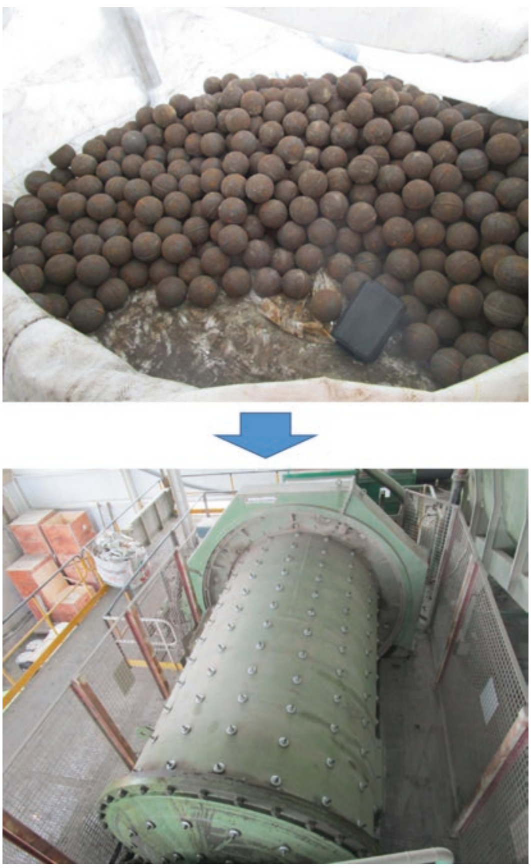


Fig. 6.24 Balls and a ball mill (Image courtesy of Daytal Resources Spain S.L.)

to 40% by volume. As grinding progresses, the steel balls are worn down, and so new balls will be added to keep the volume of balls constant. The so-called ball mill is limited to those having a length/diameter ratio of 2–1 and less. Where this ratio is between 3 and 5, the equipment is termed tube mills. This device is occasionally separated into different longitudinal compartments, each having a distinct charge type. In this case, the material can move through to the proceeding section, but the grinding medium cannot, which assures that the smaller fragments are attacked by the smaller grinding medium. Tube mills are commonly utilized in a dry manner to grind materials such as cement clinker, gypsum, and phosphate. Ball mill can be used for any grinding application in which the feed material is less than 20–25 mm in size.

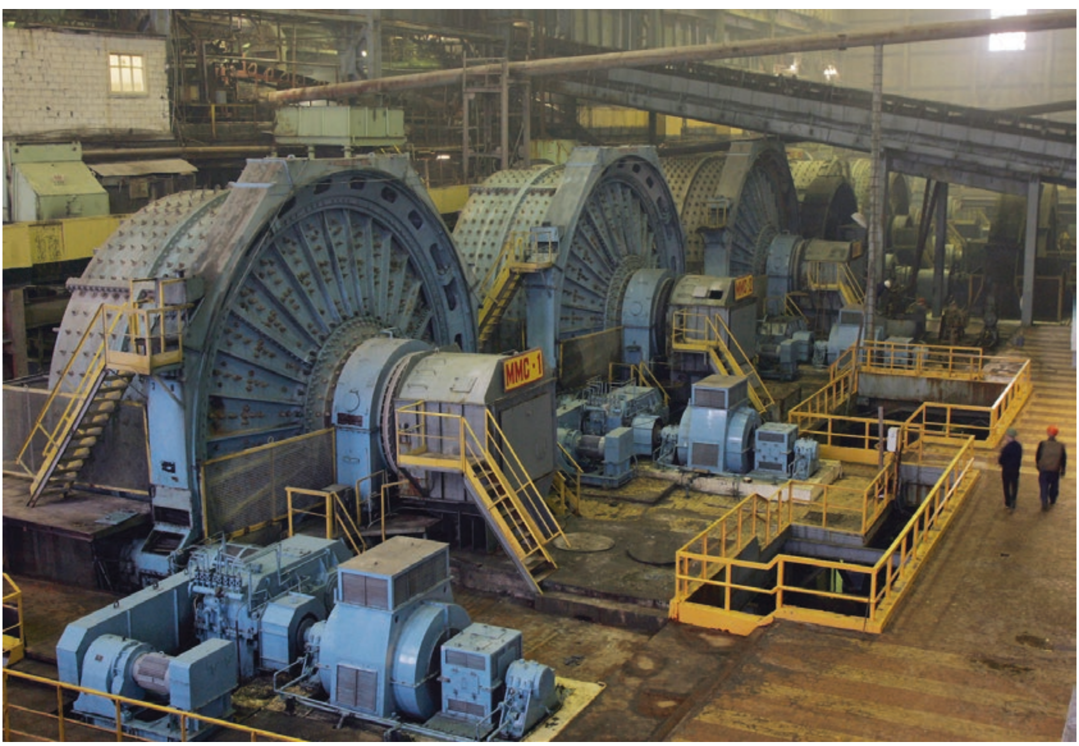
Ball mill is particularly suited for fine grinding for three reasons: (a) larger breakage forces can generally be brought to bear on individual particles in ball mills than in rod mills of the same diameter; (b) the presence of large particles in ball mills does not prevent the breakage of small particles in the way that it does in rod mills; and (c) a very large number of contact points between components of the medium can be engineered in a ball mill firstly by the use of a range of ball sizes and secondly by the use of small balls. However, one disadvantage in the use of small balls is that they exert weaker breakage forces than larger balls (Woollacott and Eric 1994). Different factors influence the effectiveness of ball mill grinding. The effectiveness also is based on the surface area of the grinding medium. Balls must be small, and the charge must be graded such that the biggest balls are just heavy enough to grind the largest and hardest particles in the feed. Primary grinding commonly requires a graded charge ranging from 5 to 10 cm diameter balls, while secondary grinding needs 2–5 cm.

Autogenous/Semiautogenous Mills

Since the advent of autogenous (AG) and semiautogenous (SAG) milling technologies in the late 1950s, they have established themselves as the current standard and are generally utilized in the industry up to now. Thus, the highest throughput grinding circuits in the mining industry use AG or SAG mills. These methods replaced the prior «conventional» comminution systems based on crushing ball mill or rod mill-ball mill installations at least in the precious and base metals industry (Gorain 2016). Therefore, AG and SAG mills have become a key element of comminution in mineral processing systems. It is clear that the main advantage of AG and SAG is that the high costs associated with the consumption of steel grinding media are partially or completely avoided.

The breakage mechanism in AG and SAG mills is basically similar to that found in other tumbling mills. In the same way that a ball mill grinds the ore using steel balls, an autogenous mill grinds due to self-grinding of the mineralization particles and any additional grinding medium such as balls or rods that are present. The tumbling drum works with a 25–40% volume filling of ore. Thus, metallic or manufactured grinding medium is not used. Therefore, there is little wear as the mineral itself carries out the grinding. AG or SAG mills are sometimes classified based on the aspect ratio of the mill shell design, which is the ratio of diameter to length. Common aspect ratios range between 3 or high aspect ratio mills to 0.3 or low aspect ratio mills. Bigger diameters are frequent in North America, while longer mills are more usual in Europe. AG/SAG mills can operate feed mineralization as big as 200 mm (the product of a primary crusher) and obtain a product of 0.1 mm in one stage.

One of the issues of AG mills (◘ Fig. 6.25) is that features of the mineralization as hardness and abrasiveness can change generating an inconsistent grinding behavior. This is because the hardness,



◘ **Fig. 6.25** AG mills (Image courtesy of Alrosa) abrasive, and fracture properties of the large lumps, which are important in determining the viability of autogenous milling, can change significantly in an ore body. Thus, the competency of the medium is a crucial matter in autogenous milling. The larger lumps of ore must not shatter too easily and so fill the mill with small pebbles that are incapable of performing any useful grinding function. It will be appreciated that not every ore will provide a competent medium. Consequently, a comprehensive test program is needed previous to autogenous milling selection for any given application.

Autogenous mills are not very efficient in producing a fine product because the breakage forces applied by the grinding media are much less than those typical of ball mills. The reason for this lies in difference in the relative densities of the grinding media: 7.5 for steel and 2.7–3.3 for media derived from ores. A good example of the opposite is the AG milling in grinding iron ores since the specific gravity of ore mineralization is about 4 versus 2.7 for high silicate ores. For this reason, autogenous mills are often used as primary mills followed by secondary ball or pebble mills. Autogenous mills are operated at higher speeds than is usual in other tumbling mills; speeds of up to 92% of critical speed are not uncommon. At such speeds, the primary tumbling action is cataracting, but the absence of steel media simplifies the problems of this mechanism.

The addition of steel grinding balls rectifies the problems associated with fully autogenous milling. The equipment is then called semiautogenous mill grinding, and the total quantity of balls incorporated in these devices are in the range of 5–15% of the volume. The addition of a small ball charge changes the nature of the mill performance considerably, although it increases obviously operating costs for ball and power. From an economic viewpoint, SAG mills are less expensive to build in terms of unit capital cost per metric ton of throughput than AG mills but more expensive to operate because of the cited increased grinding media and line costs. Many of the current mineral processing installations use SAG mills as primary grinding match with ball mills. SAG mills are usually applied in different industries such as gold, copper, platinum, zinc, silver, and nickel (Haldar 2013).

Under certain conditions, AG and SAG mills can replace the final two stages of crushing (secondary and tertiary) as well as rod milling on the traditional circuit. This generates advantages such as lower capital expenditure, capability to operate a broad range of ore type, comparatively simple flowsheets, large size of suitable equipment, lower manpower needs, and decreased steel consumption (Wills and Finch 2016). Nevertheless, this cannot proceed in the future since new devices such as high-pressure grinding rolls or ultrafine grinding utilizing stirred milling provide alternative possibilities. Comparing SAG and HPGR for primary grinding, the capital costs of HPGR are commonly higher than for the equivalent SAG-based system and for highly competent ores, and where power and grinding media costs are high, HPGR can offer substantial operating cost benefits because SAG mills are less energy efficient in handling harder and abrasive ores (Morley and Staples 2010).

Pebble Mills

Tube mills including only one compartment and a charge of hard, screened mineralization particles as the grinding medium are called «pebble mills.» The addition of pebble crushing is the most common variant to closed-circuit AG/SAG milling. The potential efficiency benefits, both in terms of grinding efficiency and in capital efficiency through incremental throughput, has long been recognized. However, the challenges of metal elimination were perceived to be a substantial obstacle (Mosher 2011). Even after steel ball removal had proved to be reliable, pebble crushing installations were still scrutinized at the design stage because of the additional cost and circuit complexity. However, a pebble crushing circuit is almost an imperative for efficient circuit operation in certain ore types. Examples of these ores are those with chert, andesite, or other hard component that develops a critically sized material that constrains milling rates.

In a size reduction circuit that uses pebble mills, large rocks (80–120 mm) are separated from the ore at an early stage in the crushing plant or by the use of large ports in the discharge end of a primary autogenous mill. These large rocks are stockpiled as pebbles and used as grinding medium in a pebble mill. Therefore, as the grinding medium in a pebble mill is derived from the ore itself, this equipment can be regarded as autogenous. However, the media in pebble mills are elected more carefully than in conventional autogenous milling, and a significant portion of the size reduction of the mineralization is carried out in a previous size reduction operation.

Pebble mills are used in a similar way to ball mills. The pebbles are charged periodically to maintain an optimum load volume, although the difference in density and hardness means that the replenishment rates are higher than for ball mills. Pebble mills have the advantage over ball mills where iron contamination needs to be avoided. Since the weight of pebbles per unit volume is about 35–55% of that of steel balls and as the power input is proportional to charge content, the power draw and capacity of pebble mills are consequently lower. Thus, a pebble mill would be much bigger than a ball mill for a certain feed rate, generating higher capital cost.

Stirred Mills

Starting about two decades ago, stirred milling has become more prevalent in milling operations, corresponding to the increase in processing of more complex fine-grained ores demanding liberation grinds of 10 μm (Ellis and Gao 2002). Thus, the stirred mill is a fine-grinding device capable of producing particle sizes less than 1 μm. Stirred milling devices have solidly established themselves in ultrafine grinding systems as energy-efficient alternatives to tumbling ball mills. Stirred mills differ from tumbling mills in how grinding energy is transferred to the material being ground. Tumbling mills use both impact and shear (abrasion/attrition) energy in roughly equal measure, whereas stirred mills use predominantly shear energy (Radziszewski 2015). Since shear is more effective than the impact for fine grinding, stirred mills are more energy effective than tumbling ones where the product is less than about 100 μm. While early applications were in regrinding, stirred mills have drawn attention for utilization in primary grinding circuits (Wills and Finch 2016).

Stirred mills can be broadly separated into two categories: gravity-induced mills and fluidized charge mills. In gravity-induced stirred mills, the screw rotates slowly such that the ball charge and slurry are settled under gravity. In contrast, the fluidized mill type utilizes high-rotational velocities of either impellers or disks to produce the suspension and total mixing of the grinding media and slurry particles. Thus, the fluidization compels the slurry particles and grinding media to remain in contact with each other, and the resulting relative motion induces size reduction of the slurry particles by abrasion and attrition grinding (Ntsele and Allen 2012).

6.6.5 Size Reduction Circuits

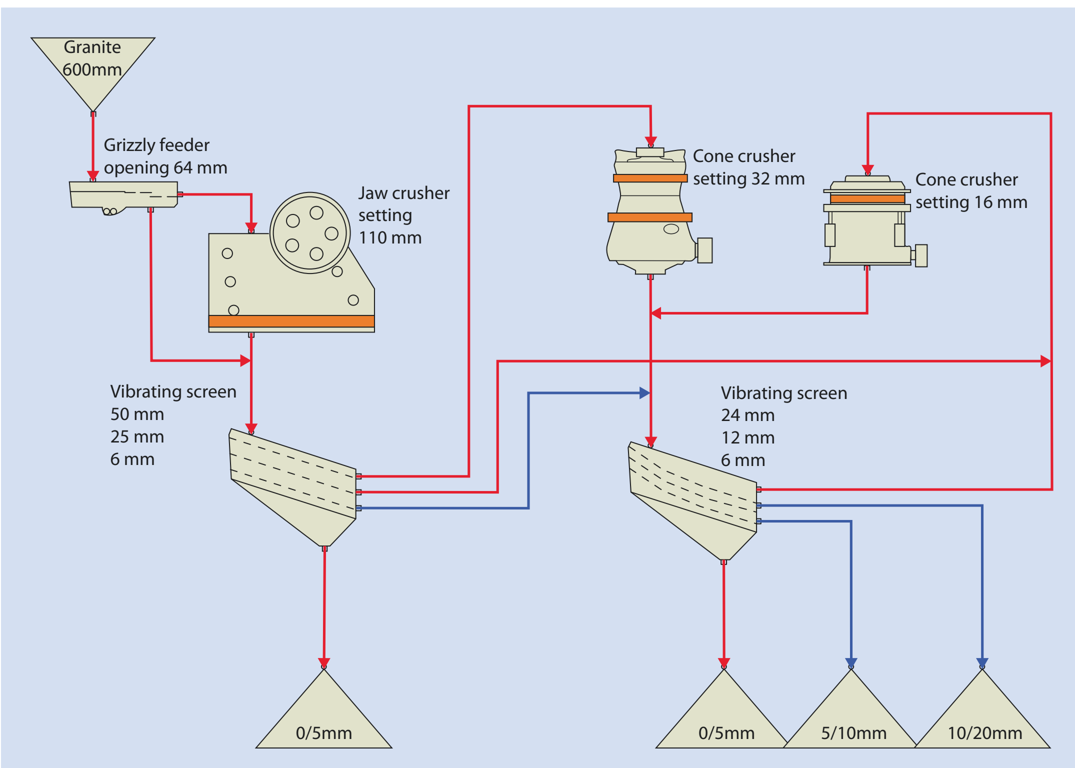
Three main criteria establish the manner in which a size reduction circuit is designed: (a) requirements to maintain the desired production rate,

Fig. 6.26 Mill control room (Image courtesy of North American Palladium Ltd.)

(b) to produce a crushed or milled product of the desired size, and (c) to minimize costs because the circuit configuration and selection of the equipment represents a critical cost saving. Thus, the layout of comminution installations in mining operations is an essential factor in meeting production requests while maintaining capital and operational costs to a minimum. Obviously, all processes of comminution are controlled from a control room (◘ Fig. 6.26). In fact, the number of factors that influence the selection of comminution circuits is extensive depending upon the nature of the project, whether it is greenfield plant or an expansion, as well as on a thorough understanding of the ore characteristics and scoping of test work at each of the study (Barratt and Sherman 2002). Circuit design has progressed with larger crushers utilizing more horsepower and velocity to operate higher throughputs at a deceased cost.

Processing installations can utilize several strategies to carry out comminution largely based on the ore types. The most common plans involve the use of single-stage crushing and further autogenous (AG) or semiautogenous grinding mills (SAG) and multiple-stage crushing followed by ball mills and/or rod mills. Single-stage crushing is commonly utilized where the final product size of the mineralization is generally between 90 and 200 mm. Secondary-stage crushing includes crushing the product to its required size (e.g., 15–35 mm) utilizing two types of crushers, normally a jaw and cone crusher. Regarding tertiary stage crushing (◘ Fig. 6.27), it involves crushing the product to its needed size (e.g., 7–15 mm) applying two or more types of crushers. If utilizing cone crushers for both the secondary and tertiary circuits, it is usual to operate with different head arrangements on each cone crusher.

The appropriate selection of equipment in circuit design is determined by feed size, ore type, tonnage, and final product size. The fact that the inherent efficiency of some devices is higher than others causes comminution circuit designers to select equipment that produces a favorable overall efficiency. In this sense, a key aspect of achieving high overall efficiency is to remove product size material as soon as possible after it is created. This is because the material that it is already finished takes up energy and interferes with the breakage of coarse particles. Thus, effectiveness size separation



◘ **Fig. 6.27** Tertiary stage crushing (Illustration courtesy of Metso)

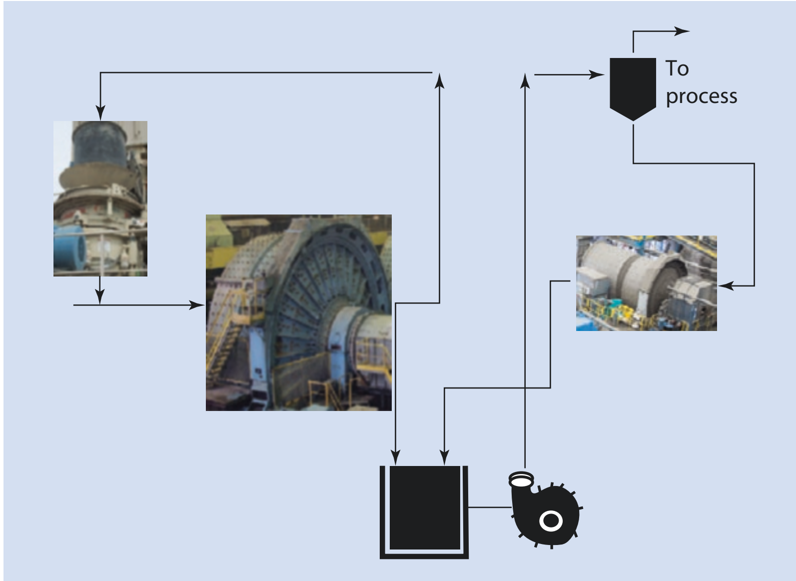


Fig. 6.28 Crusher + autogenous mill + ball mill circuit

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utilizing screens, hydrocyclones, and other classifiers is an essential part of the circuit design. In general, the ideal comminution circuit must be easy to operate and maintain, is power efficient, and has a low or no steel media consumption.

Regarding the grinding circuit (Fig. 6.28), since its main goal is to decrease the ore particle to a level that allows effectively separation between gangue and valuable minerals in the further processing step, a perfect trade-off between quality and quantity is crucial. As commented previously, it is also essential to develop this process at the lowest suitable energy and grinding medium consumption. The appropriate control of grinding process must manage the ore and water feed rate and the mill speed. Other features that increment complexity to this control issue are ore size and the quantity of circulating load. For this purpose, circuit simulation is becoming increasingly critical for optimization, design, and control. It is the ability to model the behavior of individual pieces of equipment and then to combine these models in such a way that they quantitatively predict the performance of circuits and ultimately entire plants.

Therefore, process control is an essential component of any comminution system. Virtually all plants built today have a sophisticated digital control system that enables all basic control functions. In addition, most new plants adopt advanced process control applications to deal with the multivariate nature of process optimization in real time. The complexity of the control strategy depends in part of the complexity of the process and in part on the nature of the disturbances. Disturbances are temporal changes in ore characteristics in the feed to a comminution process because the operator or control system can do nothing to modulate them. These disturbances will arise in ore hardness (e.g., different ore types and genesis modes), feed size (e.g., blasting practices and stockpile segregation), and liberation requirements (e.g., requiring a change in grind) (Herbst et al. 2003). Although the comminution process can well be stable to these fluctuations without intervention, control systems are normally required to ensure stability and to enhance the overall economic performance of the process. Instrumentation in process control includes ore level detectors, oil flow sensors, power measurement devices, belt scales, variable speed belt drives and feeders, particle size measurement devices, and many others (Flintoff et al. 2014).

6.7 Size Separation

Two distinctly different types of separation techniques based on size can be recognized: screening and classification. Election between screening and classification is determined by the fact that finer separations require large areas of screening surface and thus can be expensive compared to the classification for high-throughput applications. Unfortunately, ideal separations are never achieved because some undersize particles will report to the

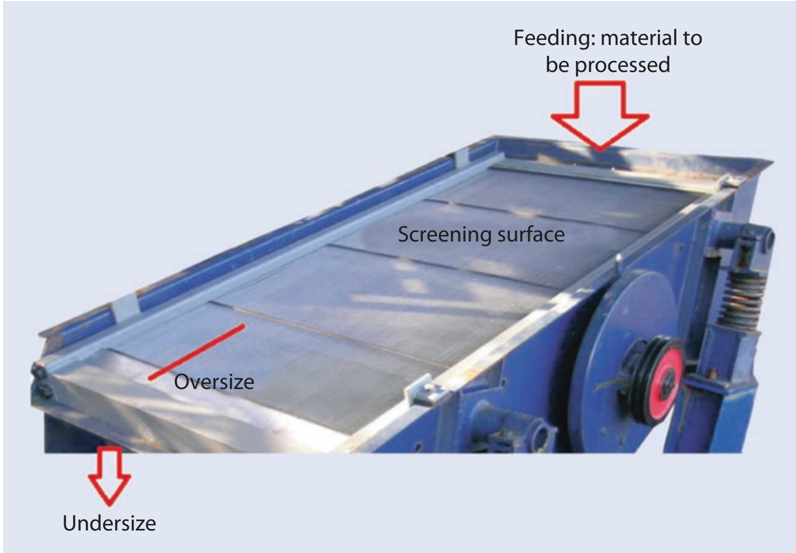


Fig. 6.29 Screening

oversize stream, and some oversize particles will report to the undersize stream, being this feature a measure of inefficiency in the separation procedure.

6.7.1 Screening

Screening, also called mechanical classification, is one of the oldest unit operations, and it is used in many industries worldwide. Industrial screening is widely applied to size separation from 300 mm down to 40 µm, but the effectiveness reduces quickly with fineness. Separations of dry particles by using screens are commonly attempted down to about 75 µm. As a general rule, classification is unchangingly the single choice for the separation of particles smaller than about 0.1 mm.

According to Wills and Finch (2016), there are a wide range of screening objectives in the minerals industry: (a) sizing, to separate particles by size; (b) scalping, to remove the coarsest size fractions in the feed material; (c) grading, to prepare a number of products within specified size ranges; (d) media recovery, for washing magnetic media from ore in dense medium circuits; (e) dewatering, to drain free moisture from a wet sand slurry; (f) de-slimming or de-dusting, to remove fine material, generally below 0.5 mm, from a wet or dry feed; and (g) trash removal, to remove coarse wood fibers or tramp material from a slurry stream.

In this section, the main objectives of screening are the first two goals, whereas other purposes such as dewatering will be covered in other headings of this chapter.

Screening uses a geometrical pattern for size control: the screen. In its simplest version, a screen is a perforated and rigid surface that is uniformly perforated with apertures, which are normally all the same size. In screening, the material is sent to the screen surface so that material finer than the apertures falls through the screen (they are called undersize, minus, or lower product) and the oversize is sent to the discharge zone (termed oversize, plus, or upper product) (Fig. 6.29). The desired separation size is called the «cut size.» The effectiveness of screening is established by the amount of separated material into size fractions above and/or below the aperture size. The screens are used in all types of crushing devices at feed and discharge stages. Each screen provides two products; to obtain more products, it is necessary to use additional screens.

Whether a particle passes through the apertures of the screening surface to the undersize product will depend mainly on its size and shape, although other aspects such as screen dimensions and loading, the nature of the relative movement between particles and screening surface, the percentage of open area or degree of perforation, and the screen angle can also influence on the separation process (Woollacott and Eric 1994). Regarding the particle shape, large needlelike particles can be able to pass to the undersize if correctly oriented, even though according to their length, they should report to the oversize. Similarly, fairly small platelike particles can tend to bridge apertures and be misplaced to the oversize product. In general, screen efficiency must always be matched with capacity. In this sense, it is commonly possible by the utilization of a low feed rate and a very long screening time to effect a near complete concentration.

Screen angle or slope of screen deck also affects considerably the separation process because the slope of the screening surface influences the angle at which particles are presented to the screen apertures. Some screens apply this effect to obtain separations definitely finer than the screen aperture. For example, banana screens incorporate a variable-angle slope that allows for increased throughput. In general, where the discharge end of a screen deck is inclined down from the horizontal, the material cascades more rapidly down the slope and either passes through an opening or over the screen surface in accord with some probability. Hence, the capacity of a screen must increase as deck slope increase. Efficiency will remain constant or increase up to a critical slope and then reduce as slope increases. In crushing plants, screen decks are usually operated at angles ranging from 20° to 30° below the horizontal (Mular 2003). Vibrating is another factor that affects the screen performance. Screens are commonly vibrated by circular or elliptical motion with the objective to throw particles off the screening surface so that they can again be presented to the screen and to convey the particles along the screen. In view of all explained about slope of the screen and vibration, inclined vibrating screens are by far the most popular device in mineral processing plants.

There are many types of screening surface available. The size and shape of the apertures, the proportion of open area, the material properties of the screening surface, and the flexibility of the screen surface can be essential to the efficiency of a screen machine. For a long time (Taggart 1945), screening surface can be mainly placed into one of three general categories: woven wire screen, perforated screen plate, and profile wire or bar. Woven wire screen is the most used, although perforated plate is often employed for very intensive use and/or coarse sizes. Plates are more expensive, but they resist wear and have a long life, less blinding, higher efficiency, and a high degree of accuracy in sizing. Choice depends on the size range involved, the nature of the application, the desired capacity, and the corresponding efficiency of the screen. Screening surfaces are usually manufactured from steel, rubber, or polyurethane (Fig. 6.30).

There are three main constraints that limit the application of screening in industrial-scale size separations; these constraints represent all problems taking place in the separation of small particles. First, the screens tend to become blinded by particles that get stuck in the apertures, increasing this tendency rapidly as the size of the apertures decreases. It becomes a major problem at an aperture size below about 0.5 mm. Second, every particle must be presented to an aperture at least once to achieve a reasonable separation. Near-size particles, those that are only slightly smaller than the screen

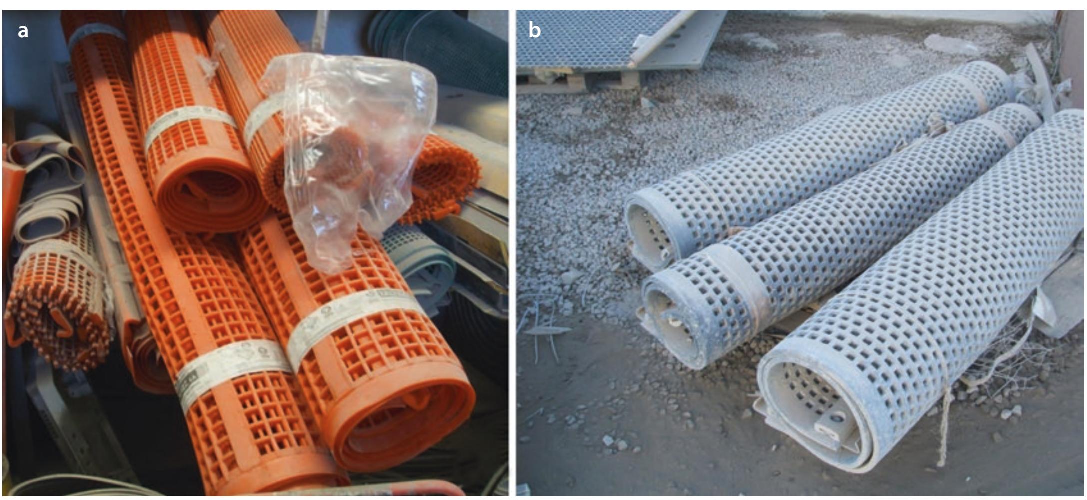


Fig. 6.30 Screening surfaces of polyurethane **a** and rubber **b**

aperture, need to be presented many times before they fall through. Thus, to screen very fine particles efficiently requires repeated presentations of a very large number of particles to a limited number of apertures. This is because the number of apertures is limited and the number of particles per unit mass increases very rapidly as the particle size decreases. Consequently, separation rates decrease very significantly as the separation size becomes smaller.

The third constraint is related to the screening surface and the number of apertures. With decreasing cut size, a reduced proportion of the screening surface is occupied by apertures, and the percentage of open area of the screen surface reduces. The reason for this is that the rigidity and mechanical integrity of a surface demand a certain minimum amount of material between the apertures. Therefore, as the number of apertures increases and their size reduces, the proportion of the open surface decreases as well (Woollacott and Eric 1994).

Screening Equipment

Grizzlies

Grizzlies (Fig. 6.31) are utilized for rough screening of coarse to very coarse rocks. They are most commonly utilized in crushing circuits, being usually incorporated for sizing the feed to the primary crusher. A grizzly is basically a set of steel rails, bars, or rods placed in a parallel manner at a determined separation (e.g., 10–200 mm). The rail grizzlies can be designed in a horizontal flat plane, but they are commonly inclined in the direction of the material flow to aid transport of ore across the screen. The inclination often ranges from 30° to 40°. Thus, coarse fragments slide on the inclined surface of the bars, and particles finer than the spacing between the bars fall through. Grizzlies are static, but they can be vibrated to improve the performance, for instance, where sticky materials are present.

Trommels

A trommel is a horizontal or slightly tilted, in the direction of the material flow, rotating cylindrical screen (Fig. 6.32). This configuration is very ancient and comprises a cylindrical screen commonly rotating at between 35% and 45% critical velocity. They can operate fragments from 6 to 55 mm, and even smaller sizes can be incorporated under wet conditions. A trommel can separate different products by utilizing a series of screen with the coarsest to finest apertures. It can operate both dry (e.g., in construction and demolition materials) and wet feed material. Trommels are cheap and most useful for soil washing of coal and iron ore industry at higher-end applications. Screening of aggregates and road material is also one of the most classical applications of this device, but this utilization is clearly diminished because of its lower capacities and quick wear. In this case, trommels are usually replaced by vibrating screens.

Fig. 6.31 Grizzly for rough screening

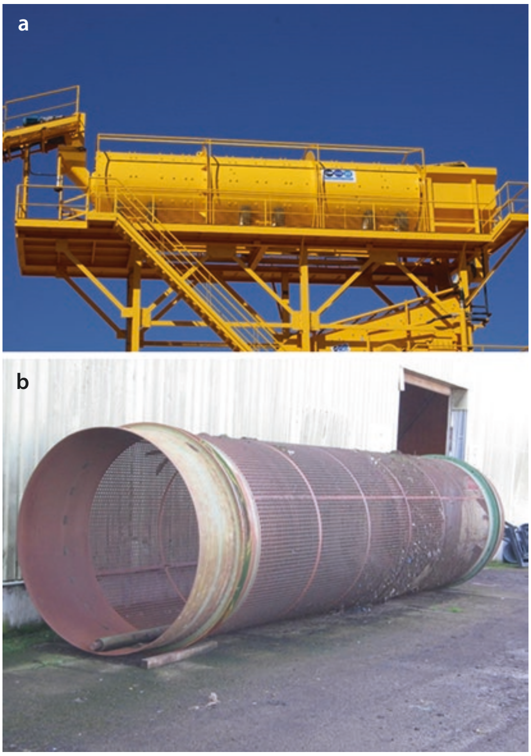


Fig. 6.32 Trommel **a** and the screen inside the device **b**

Vibrating Screens

Vibrating screens are undoubtedly the most famous screening devices for mineral processing applications. In these machines, the perforated surface is set into a frame that is agitated vigorously. The movement of the frame can be designed to facilitate the translation of the material bed along the screen. They perform size separations from 250 mm down to 80 µm. One, two, or three screening surfaces can be set into a single frame in single-, double-, or triple-deck configurations. The rapid relative movement engineered between the particles and the screening surface facilitates rapid rates of screening. Sometimes, water is used to enhance the separation process.

The ever-growing need for obtaining a clean product, for instance, in aggregate production for civil construction, justifies the great importance placed on washing equipment and processes in crushing and screening plants. The main purpose of washing is the removal of undesirable material such as clay, extremely fine particles, soft stone, roots, organic matter, etc. In these cases, washing is usually accomplished in coarse material through the direct washing on vibrating screens. Washing is performed by applying water jets through spray nozzles directed as a water curtain and under pressure at the material being classified, aiming to remove the impure particles adhering to the material. The nozzles are commonly manufactured in polyurethane because it is economical and abrasion and corrosion resistant. They are installed in metallic pipes placed cross-sectionally in relation to the material flow (Fig. 6.33).

Mogensen sizer is a classical vibrating screen, being a probability device useful in the separation of particles in the size range of 5–0.1 mm. It

Fig. 6.33 Polyurethane nozzles placed cross-sectionally to the material flow (Image courtesy of SAMCA)

exploits the principle that particles smaller than the aperture statistically need a sufficient number of presentations to the screen in order to pass. On the other hand, Banana or multislope or multiangle concept screening machine is a major innovation in screening technology. It is a nonconventional vibrating screen capable of achieving exceptional throughput per screening area. The screen incorporates as many as six inclinations varying from 45° to 10°. Banana screens are often used in high-tonnage sizing applications where both effectiveness and capacity are required. The capacity of these devices is around three or four times that of classical vibrating screen (Meinel 1998).

Stack Sizer Screen

The stack sizer screen has changed the concept of effectiveness, fine particle wet screening. It enables high-separation effectiveness and high-tonnage capacity utilizing conventional screens (Clark 2007). Stack sizer is formed by up to five independent screen decks located one above the other and working. Its utilization together with urethane screen surfaces as fine as 75 μm has produced fine wet screening a reality in mineral processing operations (Valine et al. 2009). There are now over 400 installations of this type around the globe in grinding circuits, as an alternative to hydrocylones, for both metal and nonmetal beneficiation applications (Gorain 2016) (◘ Fig. 6.34).

◘ **Fig. 6.34** Stack sizer screen (Image courtesy of Daytal Resources Spain S.L.)

6.7.2 Classification

Classification separates mineral particles into several products based on the speed with which the particles fall through a fluid medium (Heiskanen 1993). Classification techniques take into account that particles of the same density but of different sizes settle in a fluid at different rates. These differences in the settling rates of particles can be exploited in order to effect a separation between them. As aforementioned, classification is commonly achieved in mineral processing at particle sizes that are too fine for sorting effectively by screening methods.

The devices used in classification processes are called classifiers. They are matched to the grinding machines in close circuit for operating over- and undersize particles accordingly. Classifiers are nearly always used to enhance and close the final stage of grinding and so strongly influence the performance of these circuits. The benefits of classification can include improved comminution efficiency and product quality and greater control of the circulating load to avoid overloading the circuit. This improvement in efficiency of the grinding circuit is also seen as a decrease in energy consumption. In dry classification, the fluid used is air, whereas in wet classification, which is far more common, the fluid is water. In mineral processing operations, the majority of classifications are carried out in water environment because of increased efficiency.

However, since ores are heterogeneous by definition, the influence of particle density on settling rates is very important. Ore particles have a variety of constituents with different densities. Because a classifier separates particles on the basis of differences in settling rates, the cut size that describes the separation in classifiers is not a function of particle size alone. It is essential to appreciate the relative influence of density on separation in a classifier. According to Woollacott and Eric (1994), this influence becomes less significant as the size of separation being sought becomes smaller. The particle shape also influences on the classification process. Thus, flatter and more angular particles give rise to higher drag forces than spherical particles of the same mass. As a result, such particles will settle more slowly in the fluid. Furthermore, particle-settling rates decrease as the concentration of solids in a fluid increases because the number of inter-particle collision is higher. In these situations, the influence of particle density on settling rate is accentuated.

Classification Equipment

Gravitational Classifiers

Classifiers can be grouped into two categories: gravitational classifiers and centrifugal classifiers. Two types of gravitational classifiers can be defined according to the way in which the particles settle at different rates: hydraulic classifiers and sedimentation classifiers. Hydraulic classifiers (◘ Fig. 6.35) are characterized by the utilization of additional water to that of the feed pulp, entered so that its direction of flow opposes that of the settling particles. In these devices, the slurried particles are introduced to a separating chamber that has a base consisting of several pockets. The particles are separated by contrast between the velocity of the particles and the velocity of water. Thus, each pocket collects particles that have settling rates greater than the rising rates of the injected water. The arrangement and design of the pocket and subsequent pockets are such that they recover progressively smaller particles because the rising currents are graded from a relatively high velocity in the first pocket to a relatively low velocity in the last pocket. The finest particles flow over the discharge weir. Hydraulic classifiers can be free-settling or hindered-settling types. The free-settling methods, however, are seldom utilized; they are easy and have high capacities but are clearly inefficient in sizing and sorting (Wills and Finch 2016).

Sedimentation classifiers include two groups: nonmechanical classifiers and mechanical classifiers. In nonmechanical sedimentation classifiers (◘ Fig. 6.36), the main goal is only to separate the solids from the liquids. Consequently, they are sometimes utilized as dewatering devices in small-scale processing plants. Nonmechanical classifiers are not suitable for fine classification or if a high separation efficiency is required. Mechanical classifiers are mainly used in closed-circuit grinding treatments and in the classification of materials from ore-washing plants. Their design includes a settling tank and a mechanism to remove the settle solids from the bottom of the tank. The different classifier designs are based mainly on the mode of extracting the underflow and the overflow slurries. Immersed spiral or rakes are commonly utilized for underflow slurries, and an open launder carries the overflow. The rake design includes rakes actuated by an eccentric motion, which causes them to dip into the settled material and to move it up the incline for

◘ **Fig. 6.35** Hydraulic classifier (Image courtesy of SAMCA)

Fig. 6.36 Nonmechanical sedimentation classifier (Image courtesy of TEFSA)

Fig. 6.37 Spiral classifier (Image courtesy of Alrosa)

a short distance. In spiral classifiers (Fig. 6.37), a continuously revolving spiral moves the sands up the slope. Agitation in the pool is less than in the rake classifier, which is essential in separations of very fine material. Wet classification with spiral classifiers using separation by gravity covers the size range of 100–1000 μm.

Centrifugal Classifiers

The separating mechanism in centrifugal classifiers has elements of both hydraulic and sedimentation classifications. In these devices, rapid settling and classification is carried out by incrementing the force acting on the particles by substituting the gravitational force by centrifugal forces. Different types of devices using this principle are operated for the purpose, being the hydrocyclone the simplest. This machine has grown into one of the most interesting and broadly utilized classifiers in the minerals industry. It is usually applied in closed circuit within grinding systems and is utilized to return coarse material back to the ball or rod mill for further grinding. The main advantages of hydrocyclone is that it has big capacities in comparison to their size and can split at finer sizes than most other screening and classification devices. Hydrocyclone units can be installed on simple supports as single units or in clusters.

A typical hydrocyclone is formed by a conically shaped vessel, open at its apex, or underflow, joined to a cylindrical section, which has a tangential feed inlet (Fig. 6.38). Particles entrained in the fluid under pressure through the tangential entry experience a strong centrifugal force as a result of the spiraling motion of the carried fluid. This generates a vortex in the cyclone, with a low-pressure zone

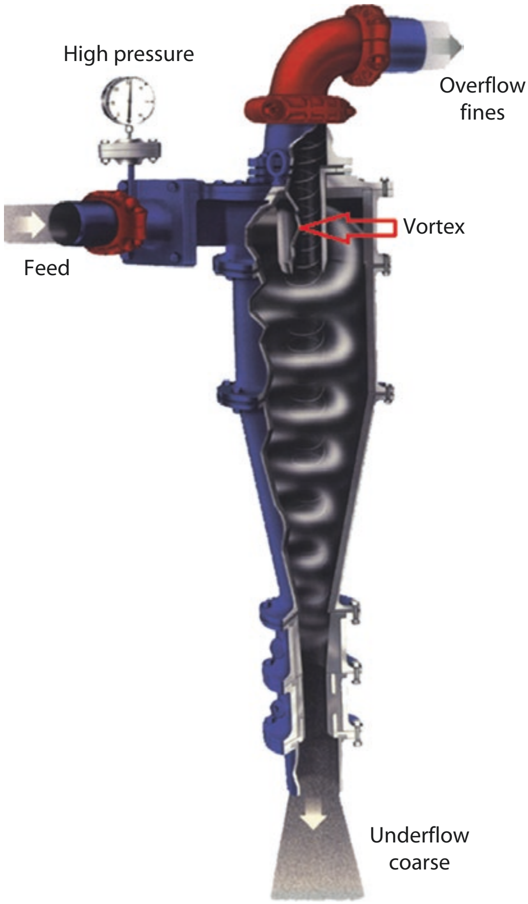


Fig. 6.38 Hydrocyclone (Illustration courtesy of Metso)

Fig. 6.39 Hydrocyclone (Image courtesy Daytal Resources Spain S.L.)

along the vertical axis. As a result, the particles settle through the fluid toward the outside of the cyclone. The largest particles settle more quickly and migrate to the apex opening. Due to the action of the drag force, smaller particles may not settle quickly enough to reach the envelope and move toward the zone of low pressure, being carried upward to the overflow. In this sense, the hydrocyclone works as a centrifugal sedimentation classifier.

Hydrocyclones (Fig. 6.39) are forthcoming in a wide range of sizes, in accordance to their applications. They range from 2.5 m in diameter down to 10 mm, which corresponds to cut sizes of 300 μm down to 1.5 μm, respectively. In this sense, these devices cannot be utilized where coarse separations are required. The inefficiency of the hydrocyclone separation needs the utilization high-circulating loads to diminish the mill residence time. In this sense, new and more efficient designs such as Cavex recyclone has been developed. Cavex hydrocyclone is a double classification unit in one stage, which seems to increase the sharpness of separation by reducing the bypass of fines to the underflow (Gorain 2016). Another major benefit of this hydrocyclone is the reduced amount of turbulence. This is due to the laminar spiral inlet geometry, which offers a natural flow path into the hydrocyclone. There are no sharp edges or square corners to slow down the feed stream. It blends smoothly with the slurry that is rotating inside the chamber. With less turbulence, it reduces wear and extends the life span of the product.

6.8 Mineral Beneficiation

Although there are obviously exceptions, the justification for processing most ores is to obtain the valuable minerals in a more concentrated form.

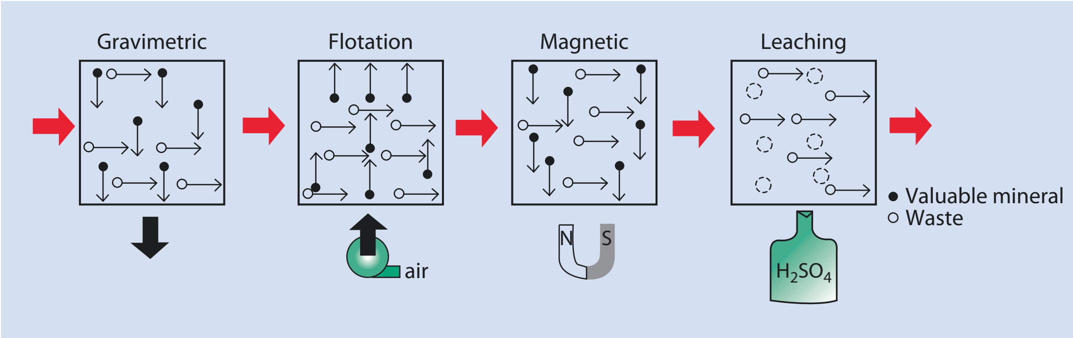


Fig. 6.40 Main beneficiation methods (Illustration courtesy of Metso)

The main goal of any mineral concentration procedure is to separate the material being treated into principally two process streams. Thus, the minerals of interest are conducted into one stream, generally termed the «concentrate stream,» while the gangue components present are conducted into a second stream, which is commonly termed the «tailings stream.» At the same time, as little as possible of the minerals of interest should be lost to the tailings stream (Woollacott and Eric 1994).

The beneficiation method selected for a particular concentration separation depends on the nature of the mineralization, the characteristics of the minerals to be separated, and the differences between those properties (Fig. 6.40). If more than one separation method is available, obviously the selection must be based mainly on economics. A wide range of processes and equipment has been developed for concentration separations. Sorting by hand and gravity concentration have been used for many centuries. By the end of the nineteenth century, these processes had been improved, and crude magnetic and electrostatic separation machines introduced. However, it was the development of flotation early in the twentieth century that provided a dramatic technological change; a wider range of minerals and lower-grade ores could be concentrated, and higher concentrate grades could be obtained. On the other hand, technological innovations in the hydrometallurgical industry in the last 50 years have consisted largely of changes that allowed companies to exploit lower-grade ores and to continually reduce the cost of metal production (Lakshmanan et al. 2016). Based on the above, the following beneficiation methods can be considered: (a) ore sorting, (b) gravity concentration, (c) dense medium separation, (d) flotation, (e) magnetic and electrostatic separation, and (f) hydrometallurgy. Although hydrometallurgy is a chemical, not a physical process to concentrate a mineral, it is considered here because its use is being increased, especially for the beneficiation of copper ores. This is the case of the so-called solvent extraction electrowinning (SX/EW).

6.8.1 Ore Sorting

Ore sorting exploits differences in the physical appearance of particles to effect a separation. The term physical appearance is used very broadly here and includes optical properties such as color and texture as well as nonoptical properties such as intensity of emitted radiation. Ore sorting is commonly undertaken after primary or secondary crushing where sufficient liberation is achieved. Applications indicate that many mines have about 30 wt% barren waste separated in the size ranging from 10 to 100 mm, which enables material to be diverted without meaningful loss of value. Therefore, preconcentration by sorting is really a procedure of enhancing the sustainability of mineral processing installations by decreasing specific materials handling requests, minimizing energy and water consumption in grinding and concentration, and obtaining more benign tailings disposal (Cutmore and Eberhardt 2002). Thus, the interest of mineralization sorting in enhancing valuation of marginal deposits is ever more being realized by the mining industry (Lessard et al. 2014).

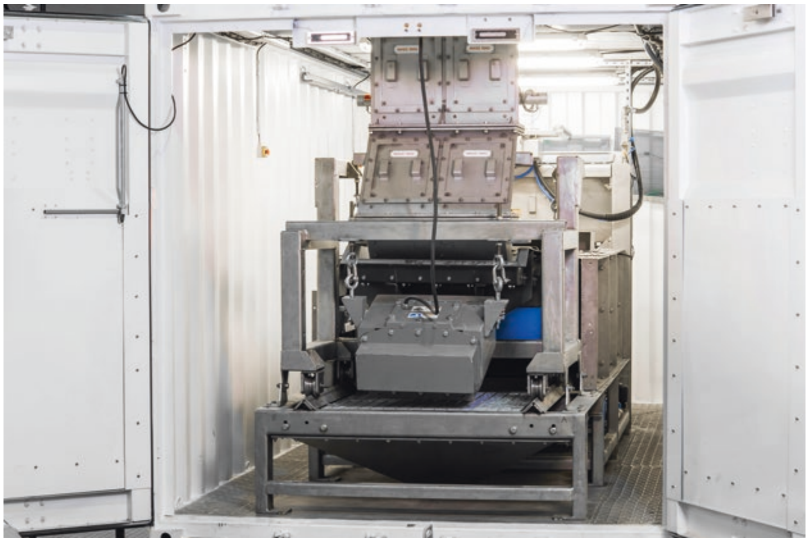


Fig. 6.41 X-ray transmission diamond sorter (Image courtesy of De Beers Group)

Ore sorting has the longest history of all mineral extraction techniques because the separation of ore from waste in mined ground during the Bronze and Iron Age would have been done by hand. In fact, the method is obviously the simplest. This principle has long been applied in handpicking (Fig. 6.6), in which the material to be sorted is placed on slowly moving belts that pass between rows of pickers. The separation desired is carried out by visually inspecting each lump of material and then picking out either the ore or the waste material. Clearly, only fair-sized fragments larger than 50 or 60 mm can be processed in this way. The practice of handpicking has declined primarily because of the cost of labor and a decrease in the grade of the ores being mined.

The advent of high-speed electronics has made possible the automation of the ore sorting process. New technologies were developed in electronic sorters based on the physical properties of the ore. Thus, «the term sensor-based ore sorting is introduced as an umbrella term for all applications where particles are singularly detected by a sensor technique and ejected by an amplified mechanical, hydraulic, or pneumatic process» (Wotruba and Harbeck 2010). The working principle for an electronic sorter is to determine the value of the mineral and then eject the particles that have this value lower than a given threshold value as gangue or concentrate. Depending upon the type of ore, various properties are used in sensor-based ore sorting. According to Bamber (2008), different methods can be outlined based on the following properties along with applications: (a) photometric (coal, sulfides, phosphates, oxides), (b) radiometric (uranium, gold), (c) conductivity (metal sulfides, native metals), (d) fluorescence (metal sulfides, limestone, iron ore), (e) X-ray luminescence (diamonds), (f) X-ray transmission (diamonds and coal) (Fig. 6.41), (g) electrostatic (salts, halite, sylvite), and (h) magnetic (iron ore, andalusite, quartz, kimberlites). Nowadays, the main types of automated ore sorters, excepting those used in diamond industry, are color or conductivity sorters. Matching two sensors has a future such as optical/near infrared, optical/inductive, or X-ray transmission/inductive (Arvidson and Wotruba 2014).

Concentration of valuable minerals using sensor-based devices is an essential procedure for the processing of certain minerals (e.g., diamonds) (Box 6.3: Udachny Diamond (Kimberlite) Processing Plant). Based on the particle size of the mineralization being separated, these machines can work at throughput rates up to 200 t/h per device. Sorting machines operate most effectively if the size of the biggest particles is no greater than two or three times the size of the smallest component, including also the presence of several machines in the plant operating diverse size fractions. Sensor-based ore sorting devices inspect particles to determine the value of some property using contactless and real-time measurements that obtain both location and material properties. Data are processed, and the information (e.g., visible light reflectance) creates a basis for ejection or retention of those components that fulfill one criterion (e.g., light versus dark particles). In this sense, the ejection can be selected either for valuable or gangue minerals. Manouchehri (2003) affirms that a sensor-based device must include several components as the essential requests: (a) method(s) for feeding, (b) method(s) for sensing (e.g., particle examination), (c) method(s) for utilizing the information achieved from sensing zone, and (d) method(s) for separating one mineral from another (e.g., air jet, water jet, and mechanical separating systems). The ultimate objective of sensor-based ore sorting is to minimize processing costs and decreasing the environmental footprint of an operation.

Box 6.3

Udachny Diamond (Kimberlite) Processing Plant (Udachny, Republic of Yakutia, Russia): Courtesy of Alrosa

The major part of the diamond reserves (about 80%) of Russia and almost half of the world's proved diamond resources are located on the territory of the Republic of Sakha (Yakutia) in Russia. Despite the fact that the greater part of deposits lies in the extreme environment of the Far North and is characterized by complicated mining and operating conditions, the diamond grade of ores contained in Yakutia's kimberlite pipes is normally higher, while their quality is comparable with that of other deposits in the world, which makes their development economically efficient. Almost all the Yakutia diamond reserves (93.1%) are concentrated in the kimberlite pipes.

The Yakutian kimberlite province on the Siberian craton is comprised of 20 kimberlite fields. The 15 northern fields do not host economic pipes, whereas some of the 5 southern fields contain kimberlites that have been mined (e.g., Udachnaya). The Udachnaya pipe is the largest source of diamonds in the Russian Federation in terms of value of diamonds recovered. The 353–367 Ma Udachnaya pipe is emplaced through the lower to upper Cambrian sedimentary rocks (limestones, dolomites, argillites, sandstones, and conglomerates) over 2 km thick, being located on the SW flank of the Cambrian Daldyn-Markha carbonate bank. The Udachnaya pipe is currently mined to a depth of more than 630 m, which makes it one of the ten deepest open-pit mines in the world. The Udachny diamond mine, discovered in 1955, is located just outside the Arctic Circle and is the largest known diamond deposit in Russia. In 2011, Udachny diamond mine has shifted from open-pit mining to underground mining.

Udachny processing plant has a capacity of 12 million tons of ore per year and is the largest of their kind in ALROSA and worldwide. The main stages of mineral processing in Udachny processing plant are the following: (a) ore preparation, (b) concentration, and (c) final recovery.

Ore Preparation

Ore preparation includes comminution and liberation. In crushing section, first there is a coarse reduction in jaw crushers in which lumps of ore of up to 1.5 m in size are reduced to 0.5 m and smaller. Then, head belt conveyor carries the reduced ore from the crusher to the feeders of the wet autogenous mills (AG). Thus, the diamond-bearing ore is fed into AG drum mills for wet milling. In AG mills, in a sparing mode, the ore is finely ground to particles sized 0.2 mm or smaller. The slurry of finely ground ore is then sent into spiral classifiers, separating the material by density. Later, the ore is screened into several fractions, and each fraction is processed separately.

Concentration

The concentration process uses three methods:

1. X-ray luminescence separation: the emitted light is detected by an X-ray recovery unit, which separates the diamond-containing material from the process stream; this concentrate is sent on to the refinement section.
2. Gravity concentration (jig separation): diamonds are heavy minerals, and they can be concentrated in heavy fractions using a jigging machine on a pulsating water bed; the concentrated material is then further refined.
3. Froth flotation: diamonds are hydrophobic, that is, they repel water; so, slurry of the finely ground ore, water, and collector chemicals are fed into a froth flotation cell; here, small diamonds get attached to air bubbles, which rise to the surface to form froth, and this froth is removed, and the concentrated mineral is further refined. The process technology minimizes breakage or other damage to the diamonds to be recovered.

In the process, midsized material is processed in jigging machines and dense media separation (DMS) units. Pulsating water jets separate the diamonds from the slurry. Smaller-sized

◘ **Fig. 6.42** X-ray fluorescent (XRF) separation site (Image courtesy of Alrosa)

material, together with water-soluble flotation agents, goes into a froth flotation cell where small-size diamonds stick to froth bubbles and are carried to the final concentration stage. On the other hand, large-size material is processed by X-ray fluorescent separation. The diamond-bearing concentrate is X-rayed while moving along the trough of the separator. After detecting a flash caused by the fluorescence, a special tool cuts precious gems away from the waste rock (◘ Fig. 6.42).

Final Recovery

In this concluding part of the recovery process, diamonds are (a) recovered from the separation concentrates, (b) cleaned by treating chemicals, (c) sieved, (d) hand-picked, (e) sorted, and (f) weighed and put in containers.

A typical example of sensor-based ore sorting is the recovery of diamonds because diamonds fluoresce where irradiated by X-rays (◘ Box 6.4: Saxendrift Diamond (Alluvial) Processing Plant). In this application, X-ray sorters are utilized in almost all diamond processing plants for the final steps of recovery after the mineralization has been concentrated by dense medium separation (see ► Sect. 8.3). The method replaces grease separation (◘ Fig. 6.43), which is actually applied specifically where the diamonds luminesce weakly or to audit the X-ray tailings. The process is commonly multistage to assure effectively the rejection of gangue components with very high diamond recoveries.

◘ **Fig. 6.43** Grease table used for secondary recovery (Image courtesy of Petra Diamonds)

Box 6.4

Saxendrift Diamond (Alluvial) Processing Plant (Douglas, South Africa): Courtesy of Rockwell Diamonds Inc

The Saxendrift diamond deposit comprises an extensive flat-lying alluvial sequence located on terraces of the Orange River. They are developed approximately 20–70 m above the left bank of the present Orange River. The fluvial-alluvial gravels comprise a sequence of 2–4 m thick of (basal) gravels overlain by a generally less than 5 m thick unit of variably calcreted sands and silts, covered by a thin layer of soil. The cobble-sized clasts within the gravels consist mostly of lava and quartzite with significant amounts of banded iron formation (BIF) and minor amounts of limestone, tillite, and agate. The gravels are commonly not well sorted and are typical of braid bars that have migrated through sections of river channels in response to variable water speed. Regarding the origin of the diamonds, the alluvial diamonds of the Middle Orange have several probable primary source areas: (a) the diamondiferous kimberlites of Lesotho eroded by the present Orange River, (b) diamonds from the same source as the Lichtenburg-Western Transvaal diamond fields eroded by the Vaal-Harts system, (c) diamonds derived from the kimberlites of the Kimberley area, and (d) diamonds from Botswana and the Postmasburg fields eroded by the palaeo-drainage.

The main characteristics of the extracting method of alluvial gravels are (a) low in situ grades in alluvial gravels, as low as 0.40 carat per 100 m^3; (b) low-cost mine of alluvial gravels but equally relatively low in situ revenues to be recovered; and (c) revenue recovery efficiency equal to that of any major kimberlite mine. The recovery process on the diamonds comprises two phases. Initially, the screened gravel is concentrated to eliminate oversize and undersize clasts as well as material that is too light or too heavy to contain diamonds. This step is followed by a physical separation of the diamonds from the gangue minerals/clasts. The Saxendrift processing plant has a nameplate capacity of 160,000 m^3 per month and processes gravels from the Saxendrift and Saxendrift Extension mining areas.

Screening

The primary screening plant or in-field screen (IFS) is barrel-fed. Run-of-mine throughput is 900 tons per hour supplying 600 tph to the double-deck screen. The plant scalps in two stages, initially at <75 mm (in the barrel screen) and secondly at 5/55 mm, and has been designed to remove about 94% of the (<5 mm) sand fraction. Then, the gravel with 5/55 mm size is fed through the plant by front-end loader machines into the feed bin. From this bin, the gravel is conveyed to feed to desanding screens. It has to be noted that the gravel often contains some 5–6 mm size particles that are considered as sand and must be still removed before further processing. The sand content of the gravel from IFS is a good indication of screening efficiency at the screening plant.

In addition to scalping/sizing, the screening plant also has two (5000 gauss) Nd-Bo-Ferrite magnets that remove significant amounts of BIF and other Fe-rich clasts from the gravels prior to transporting to the processing plant. Magnetic separation of the iron-rich component of the plant feed is of crucial importance to successful diamond recovery where using rotary pan plants in the Middle Orange River region. This is because the predominant BIF component of the gravels has a high density and can displace the diamonds.

Rotary Pans

The magnetic material in the gravel is removed before the gravel is fed into the rotary pans. Magnetic separation is through a (Nd-Bo-Ferrite) static magnet suspended above the individual pan feed belts in addition to those at the IFS plant. Thus, ferritic particles are attracted to the belt, conveyed out of the system, and stockpiled for use in rehabilitation of the mined areas around the mine.

The screened gravels are stockpiled and, subsequently, fed into the rotary pan plants by the front-end loader. The plant feed bin feeds into the primary scrubber, which disaggregates and washes the gravel and screens it at 36 mm. All oversize material is removed from the plant site by truck and dumped into open excavations as part of the rehabilitation process. The <36 mm gravels feed directly into the rotary pans (■ Fig. 6.44). Pans, which are a density-based processing technology, have been the mainstay of alluvial processing diamonds for more than 150 years. This is a particle density separation technique that relies on diamondiferous material being heavier than most of the gangue material. It typically reduces the incoming amount of material by 90%, concentrating heavy minerals. Pan efficiency is pinned at approximately 75%. There are four rotary pans in the processing plant with a combined capacity of 140 tons per hour. Rotary pans use a mixture of fine sand and water as a form of separation medium. The heavier material gravitates toward the periphery of a pan as it rotates, leaving the less dense material in the center of the pan. It is in the heavier and denser material that diamonds are concentrated. The density of the diamonds is roughly around 3.55 g/cm^3.

The float fraction is discharged onto a double-deck screen, the top deck of which is utilized as a relieving deck allowing for more efficient screening on the bottom deck which removes the <5 mm material. Undersize material and slurry from the screen is pumped to a separator cyclone situated above the pan tailings conveyor. The cyclone underflow discharges on a single-deck screen directly onto the tailings conveyor, while the cyclone overflow discharges into a sump, which is then pumped directly to the mine

Fig. 6.44 Pan plant (Image courtesy of Rockwell Diamonds)

Fig. 6.45 Secure hand sort (glove box and drop box) facility at the final recovery plant (Image courtesy of Rockwell Diamonds)

residue deposit. The oversize tailings are transported via a conveyor belt to the pan tailings bin where it is combined with the separator cyclone underflow; this material is then trucked to the relevant tailings dumps. The concentrate from each pan is removed as a batch from the pans using individual screw conveyors. The concentrate from each pan is then combined and transported along a conveyor belt to the final recovery.

Recovery

In Saxendrift plant, final recovery on the rotary pan plant is through the existing bank of twelve Flowsort X-ray machines. An X-ray machine uses X-rays to produce luminescence because diamonds and a range of other minerals exhibit this property. The final reduction of pan concentrate is down to only a few kilograms per shift. Here, the concentrate is separated into six fractions (5/8 mm, 8/10 mm, 10/13 mm, 13/17 mm, 17/21 mm, and 21/36 mm) and fed into the final recovery at a maximum feed rate of 27,000 kg/h. Thereafter, the concentrate passes through a bank of twelve single-pass X-ray Flowsort machines. Final hand sorting of the concentrate takes place inside a secure glove box (Fig. 6.45). The locked box into which the diamonds are dropped (after final hand sorting) is removed, and the diamonds are counted, weighed, and put into a «ziplock» envelope (in a second glove box). The glove boxes are locked with padlocks and seals which have to be broken to be opened. The envelopes containing the diamonds are dropped into another locked container before being stored in a high-security safe prior to dispatch to Johannesburg for sale.

6.8.2 Gravity Concentration

Gravity concentration has a long history that can be traced to before Roman times (e.g., Las Médulas Mine in Spain – see Box 1). Thus, it is one of the oldest concentration procedures beside the utilization of jigs and sluices for the concentration of heavy minerals at the sixteenth century. In 1556, Agricola, in his book *De Re Metallica*, described several gravity concentration devices used in Europe; verbatim: The practical effect of differences in specific gravity of the various components of an ore. Shaking tables were implemented in the late nineteenth century since industrialization requires a big scale production of minerals continually working. But froth flotation arrives in the twentieth century, and the significance of gravity concentration declined, although on average the gravity separation processes are comparatively cheaper and environment friendlier than froth flotation. Moreover, there are actually large tailings impoundment that could be «mined» in a non-expensive manner and processed to generate high-value products utilizing recently improved technology in gravity separation (Wills and Finch 2016).

Gravity concentration techniques find appliances in the processing of many types of raw materials and minerals such as coal, beach sands, iron ores, gold, barite, fluorspar, tin, and tungsten mineralization, among others.

Gravity separation of two minerals with different density is performed by the relative movement in response mainly to the force of gravity. Besides the density, features such as size and shape of the particle also influence the separation. A quantitative indicator commonly utilized to assess the ease with which minerals of different density can be separated in a gravity concentration process is the so-called concentration ratio, indicated in the following equation:

$$\frac{D_h - D_f}{D_i - D_f}$$

where D_h is the density of the heavy mineral, D_I is the density of the light mineral, and D_f is the density of the fluid medium. In general, gravity concentration is not possible if the concentration ratio is less than about 1.25. But if this indicator is greater than 2.5 (positive or negative), gravity separation is relatively easy; obviously, the effectiveness of separation is lower as the indicator values reduce. For example, gold has a density of 19.3 g/cm^3, whereas the average waste minerals have a density of about 3.0 g/cm^3. Consequently, the comparatively very high density of gold makes it very amenable to gravity separation. Thus, placer mining operations recover gold and other heavy minerals using gravity concentration.

From a theoretical viewpoint, there are three basic mechanisms used in gravity concentration:

1. Density: utilizes a fluid with the density value between those of the valuable minerals to be separated; the most common example is the heavy medium separation, which will be explained in the next section.
2. Stratification: the minerals are stratified by a fluidization originated by the pulsation of the fluid in a vertical plane; examples of these machines are different types of jigs utilized for separation.
3. Flowing film: the valuable minerals are separated by the relative movement through a stream of slurry that is flowing down a plane by the action of gravity; examples of devices achieving these mechanisms are sluice, spirals, and shaking tables.

Regarding separation in water, jigs, spiral concentrators, and shaking tables are the most used devices. Other ancient devices include sluices and cones. For example, pinched sluices of different forms have been applied for heavy mineral concentration for many years. In its common configuration, it is a tilted launder about 1 m long, narrowing from approximately 200 mm width at the feed zone to about 25 mm at the discharge.

Separation by Jigs

Jigging is possibly the most complex process because of its permanent varying hydrodynamics. Jig utilizes a basically vertical expansion and contraction of a bed of particles using a pulse of fluid generated mechanically or by air (◘ Fig. 6.46). A thick bed of particles is supported on a horizontal perforated surface and is introduced in water. The water level in the bed is obligated to rise and fall (the termed jigging action), and so the bed of particles is dilated and compressed in a cyclic procedure until the particles have stratified based on their density. In general, the stratified layers can be classified as heavy and light, although an intermediate middling zone is sometimes produced.

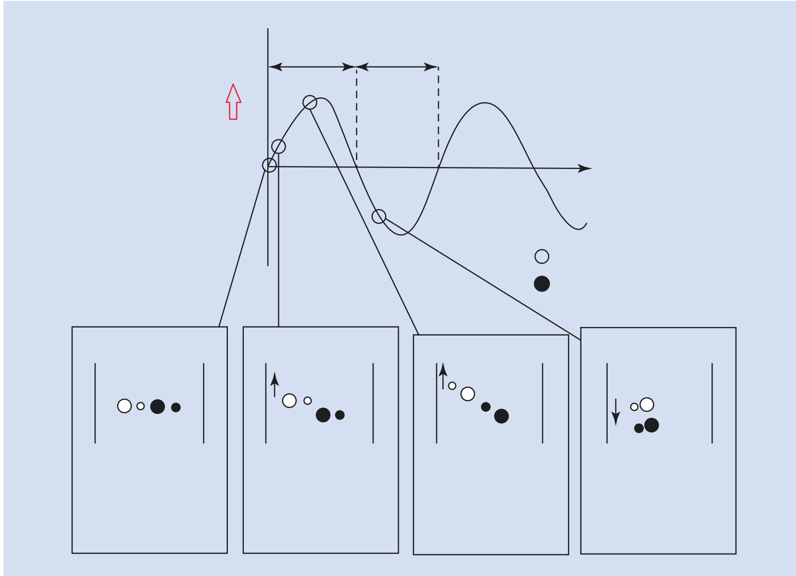


Fig. 6.46 The jig cycle (Modified from Woollacott and Eric 1994)

The frequency of pulsations commonly ranges from 50 to 300 cycles per minute. The pulsating water currents in the jig are produced by a piston having a movement that is a harmonic wave form. During the pulsion part of the cycle, when the water level rises, the bed dilates, and stratification by differential acceleration and hindered settling can take place. During the suction part of the cycle, when the water level falls, the bed consolidates and stratification progresses. The duration of each cycle and the mean water level in the bed control the relative contributions of these different mechanisms to the stratification that occurs (Woollacott and Eric 1994). As with most concentration devices, jigs rarely have five sufficient separations in one stage; thus they are commonly built with a number of devices in series.

From a design point of view, the jig is basically an open tank filled with water, with a horizontal jig screen at the top, and provided with a spigot in the bottom, or hutch compartment, for concentrate removal (Fig. 6.47). The jig is usually utilized to separate fairly coarse material, and if the feed is relatively closed sized (e.g., 3–10 mm), it is easy to obtain correct concentration of a narrow density range in minerals (e.g., fluorite −3.2 g/cm^3– from quartz −2.7 g/cm^3). For this reason, jigs are broadly utilized in mineral processing. Sometimes effective separations of particles down to 75 μm can be carried out. Many large jig circuits are still operated in the coal, cassiterite, tungsten, gold, barite, and iron ore industries. In jig procedure, high amounts of fine elements can interfere the right operation of the device, and for this reason the fine's amount is strongly controlled. The in-line pressure jig (IPJ) was an important device in the 1990s, which uses a moving jig screen attached to a hydraulic ram, treating particles up to 30 mm (Gray 1997).

Another example of this technology is the centrifugal jig (e.g., Kelsey jig – Fig. 6.48), which includes the classical components of a jig and, in addition, a centrifugal spinning motion to improve the gravity concentration. This allows sand-sized materials of relatively similar density as well as fine components to be concentrated more effectively. Due to the additional processes applied, these devices are significantly more expensive and complex to operate and maintain.

Separation by Spiral Concentrators

The most broadly utilized flowing film concentrator is the spiral one, based on Humphrey's spiral introduced in 1943. Spiral concentrators are high-capacity, low-cost devices marketed for the separation of low-grade mineralization and industrial minerals. The most typical application is possible for the concentration of sand deposits including heavy minerals such as ilmenite, rutile, zircon, and monazite. Materials in the size range of 3 mm to 75 μm can be readily processed in spirals. However, as in most gravity concentrators, the more closely sized the material being processed, the more efficient the separation. The

Fig. 6.47 Jigs (Image courtesy of DOVE)

Fig. 6.48 Kelsey jig (Image courtesy of Tronox)

new materials such as fiberglass and spray coated with polyurethane during the 1980s enabled important alternations to the spiral trough geometry and designed to tackle the requirements for more complex separation applications (Honaker et al. 2014). This generated higher-throughput spirals with a broad range of applications. The compound spiral was a further adjustment in the 1990s to enhance concentration effectiveness for coal and heavy mineral applications (Luttrell et al. 2007).

A spiral concentrator is formed by of one or more helical profiled troughs installed on a central column. As the slurry (between 15% and 45% solids by weight) travels down the spiral, high- and low-density particles are stratified and separated. Separation is obtained by stratification of particles generated by a combined effect of centrifugal force, differential settling, and heavy particle migration through the bed to the inner part of the device. Thus, the heavy fractions flow down the inner part and the light fractions down the outer part of the spiral. Discharged parts for the extraction of the higher-density particles are situated at the lowest points in the cross section. Splitters at the end of the spiral are often used to give three products

Fig. 6.49 Three splitters at the end of the spiral (Image courtesy of Daytal Resources Spain S.L.)

(Fig. 6.49) – heavies, lights, and middlings – giving possibilities for recycling and re-treatment in other spirals. Spirals are manufactured with varying slopes, the angle value of which is based on the density of separation. For example, shallow angles are utilized to separate coal from shale, while steeper angles are applied in concentration of heavy mineral and quartz separations.

The mechanism is really complicated, being much influenced by the slurry density and particle size. A great variety of modifications have been carried out by various manufacturers to enhance the general procedure of these units. Spiral capacity ranges from 1 to 3 t/h on low slope spirals to about double this capacity for the steeper units. Thus, as the capacity of an individual spiral is limited, multiple units are necessary for large throughputs (Fig. 6.50). To make a spiral installation more compact, two or three spirals can be incorporated in a single unit, known as two-start or three-start spiral, respectively (Woollacott and Eric 1994).

Separation by Shaking Tables

The most important gravity device that employs thin-film concentration as a primary separating mechanism is the shaking table. It is a relatively old device that has slowly evolved into the modern tables, which have a small but important place in mineral concentration. Generally, shaking tables treat finer materials than jigs are able to handle, although single-deck tables have relatively low capacity for their cost and space requirements. Shaking tables, sometimes called wet tables, are formed by a sloping deck essentially rectangular with an adjustable slope of about 0–6° and a riffled surface. Wash water is added continuously so as to flow in a thin film down the slope. At the same time, a motor drives a small arm that shakes

Fig. 6.50 Multiple units of spiral concentrator (Image courtesy of Daytal Resources Sapin S.L.)

the table along the length and perpendicular to the water flow and parallel to the riffle distribution. The differential shaking action is applied to the table along its horizontal axis. This action not only opens the bed to allow dense particles to sink but, by its asymmetry, provides particle transport along the table. The riffles in the table are distributed in such a way that heavy particles are caught and sent parallel to the direction of the oscillation.

The material is fed to the table at one corner (Fig. 6.51). It is acted upon by the flowing film and by the translational force imparted by the shaking motion of the table. Thus, the heaviest and coarsest particles are transported to one end of the table, while the lightest and finest particles tend to wash over the riffles and to the bottom edge. Intermediate positions between these extremes produce recovery of the middling particles. The shaking movement of the table transports the heavy minerals along the back of each riffle to the concentrate discharge. Successive steps of concentration is a characteristic of many shaking tables. The mineral is concentrated in as coarse a state as possible to obtain reasonably quick beneficiation and hence high throughputs.

Shaking tables (Fig. 6.52) are very flexible in their application, and a variety of table designs and riffle geometries can be employed, being possibly the Wilfley table the most common shaking device. In operation, the table

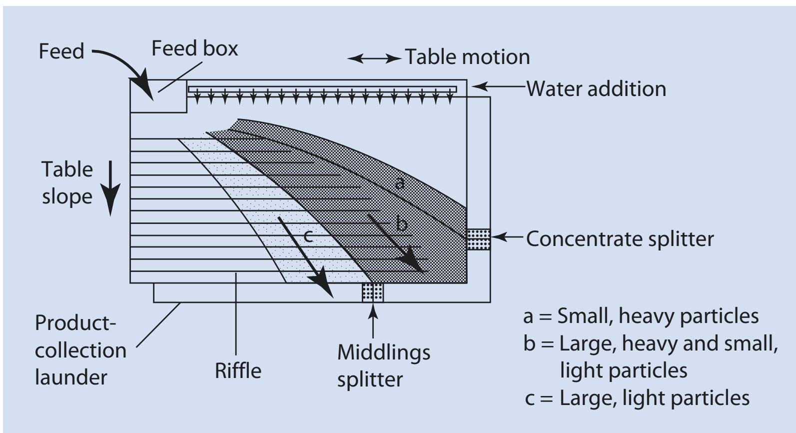


Fig. 6.51 Shaking table (top view) (Modified from Woollacott and Eric 1994)

Fig. 6.52 Shaking tables (Image courtesy of Tronox)

slope, shaking speeds, and water flowrates can be manipulated to optimize the separation achieved. They are used extensively to concentrate gold but are also utilized to beneficiate tin, iron, tungsten, tantalum, mica, barium, titanium, zirconium, and sometimes silver, thorium, and uranium. In all cases, the particle size must be between 7 and 0.10 mm. On the other hand, it is very usual that these machines are utilized together with other gravity separation equipment such as spirals, jigs, and centrifugal gravity concentrators for final cleaning previous to the refining or sale of product. Shaking tables are actually being also utilized in the recycling of electronic scrap to recover precious metals (Box 6.5: Los Santos Scheelite Processing Plant).

6

Box 6.5

Los Santos Scheelite Processing Plant (Salamanca, Spain): Courtesy of Daytal Resources Spain, S.L.

Los Santos deposit lies within the Lower Paleozoic sediments in the Central Iberian Tectonic Zone. The stratigraphy comprises a thick sequence of clastic metasediments, ortho-, and para-gneisses, with volcanic and carbonate formations. This stratigraphy was intruded by Hercynian granitoids in a series of plutons with numerous granite and aplite dykes, sills, and irregular pods intruding the metasediments up to 0.5 km from the regional granite contact. Los Santos deposit is a typical skarn-hosted scheelite deposit where intrusion of granitoids into carbonate-rich sedimentary rocks has resulted in their replacement by calc-silicate or siliceous minerals, together with mineralization. It formed from impure Fe-rich carbonates and contains pyroxene, scheelite, plagioclase, and locally magnetite. The scheelite is generally fine grained (<1 mm), but individual crystals may exceed 1 cm. In particular areas, sulfide-rich skarns also occur. They are up to 5 m thick and several meters in strike length and comprise massive or semi-massive sulfide horizons with scheelite mineralization. Sulfides comprise pyrite, arsenopyrite, pyrrhotite, and chalcopyrite as principal minerals.

Los Santos scheelite processing plant is located immediately to the south of the Los Santos Sur pit, close to the existing mine workings, the main waste dump, and other infrastructure. It is processing 500 ktpa (thousand tons per annum) of ore. The plant is primarily based on gravimetric separation, aimed at recovering scheelite, to provide a concentrate containing greater than 65% WO_3. The overall mill flow sheet is shown in Fig. 6.53.

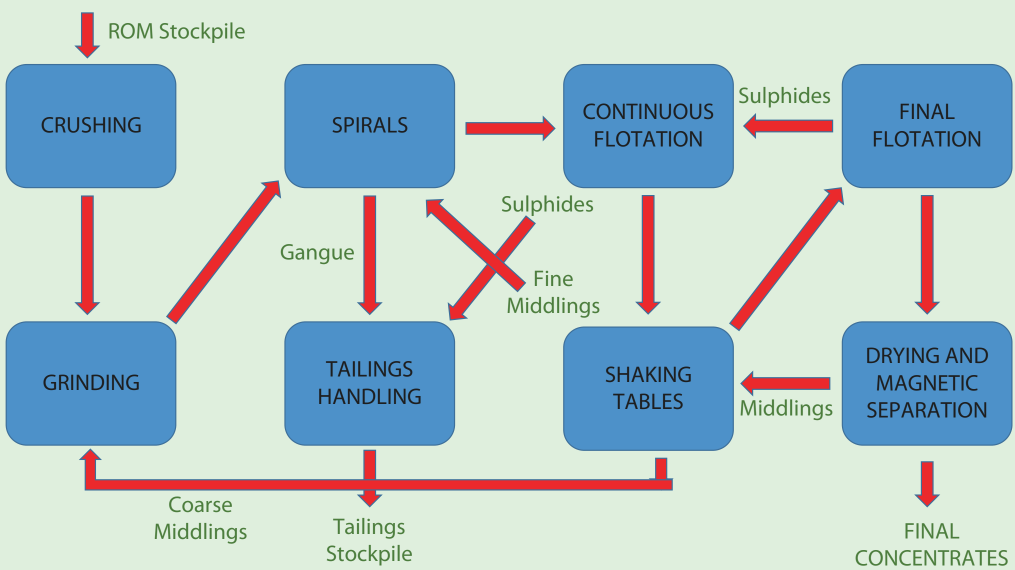


Fig. 6.53 Flow sheet of the scheelite processing plant

Crushing

Run-of-mine is dumped in various locations on the ROM pad according to grade-range categories and for weathered material. This material is blended by feeding into a primary jaw crusher using a front-end loader at a nominal 100 t/h rate. Then, a conveyor delivers the jaw crusher product to a primary cone crusher, and the product from this crusher is dry-screened on a double-deck vibrating screen. The top deck oversize (>27 mm) is returned to the primary cone crusher, while the bottom deck oversize (>12.5 mm) is passed to a secondary cone crusher. The secondary cone crusher product joins the primary cone crusher product and recycles back to the screen, while the bottom deck undersize at <12.5 mm size is discharged to a stack-out conveyor that forms a conical open stockpile ahead of the main process plant (Fig. 6.54).

Grinding

Crushed ore from the stockpile is delivered via variable-speed belt feeders and conveyors at 65 t/h to a 2.9 m diameter × 4.8 m long rod mill that wet grinds the ore in an open circuit to approximately 50% passing 250 µm. From the rod mill, the ground ore passes to a Derrick stack sizer where it is wet-screened at 1000 µm. The stack sizer oversize (>1000 µm) is fed to a regrind mill, which is a 2.1 m diameter × 3.7 m long ball mill. The re-milled product from the ball mill is sent to the gravity circuit.

Spiral Separation

The ore is first classified into separate coarse (<1000 µm, nominally >150 µm) and fine (nominally <150 µm, >30 µm) fractions using banks of hydrocyclones. The final <30 µm overflow is not processed in the gravity circuits and goes directly to the tailings thickener. The first separation step in the coarse circuit is the removal of strongly magnetic particles, mainly mill steel and pyrrhotite, using a low-intensity, wet magnetic drum separator in each of the two gravity circuits. The non-magnetics pass to a bank of coarse rougher spirals (36 starts), the concentrate from these spirals go to a bank of cleaner spirals (12 starts), and the middlings go to a bank of middlings-cleaner spirals (12 starts). The tailings from the rougher and middlings-cleaner spirals go to final tailings, as do the middlings from the last bank, while the tails from the cleaner bank go back to the regrind ball mill, and the middlings are recycled back to the feed.

In the fine circuit, the rougher spirals comprise one bank of 48 starts. The concentrate from these fine rougher spirals goes to a bank of fine cleaner spirals (12 starts), and the middlings, from both banks, recirculate back to the rougher bank. The tails from both banks of spirals go to the new fine scavenging circuit. The new fine scavenging circuit comprises a bank of rougher spirals (36 starts) and a bank of cleaner spirals (12 starts). The middlings from each bank recirculate over the respective feed, and the tailings are discarded as final tailings. The concentrate from the cleaner spirals is sent to the hydraulic classifier. The overflow from the hydrocyclones that control the pulp dilution in the rougher bank is thickened in a second group of hydrocyclones, being the underflow fed to another set of fine spirals (12 starts). The middlings from these spirals recirculate over the feed, and the tailings are discarded as final tailings. The concentrate is sent directly to the ultrafine group of shaking tables.

Continuous Flotation

The concentrate from the coarse and fine cleaner spirals, from the middlings-cleaner spirals, and from the fine scavenging spirals are collected and pumped to the continuous flotation circuit. This

Fig. 6.54 Intermediate crushed stockpile (Image courtesy of Daytal Resources Spain, S.L.)

circuit comprises one bank of four Denver flotation cells of 1.4 m^3 each. The sulfides are recovered in the froth, which are discarded as final tailings. The flotation is performed at a neutral pH, and potassium amyl xanthate (PAX) is used as a collector. Copper sulfate is used to help in the activation of the sulfides, and pine oil is used as a frother. The product that remains in the pulp is collected and pumped to the hydraulic classifier.

6

Table Separation

The hydraulic classifier comprises nine chambers and feeds four separate shaking table circuits. All the circuits are structured in rougher and cleaner steps with the exception of the fine and ultrafine circuit, which only has rougher tables. The tailings from the rougher step of the coarse and intermediate tabling circuits are sent to regrinding. The tailings from the rougher fine tables are sent to the fine rougher spirals, while the tailings from the ultrafine tables are considered final tailings. The tailings from the cleaner step of all tabling circuits are recycled back to the hydraulic classifier. Finally, the concentrate from all the cleaner tables and from the ultrafine rougher tables forms the gravity preconcentrate.

Tailings and Fines

Tailings from the spirals and shaking tables as well as magnetic fractions from the low-intensity drum magnets are pumped to a bank of dewatering cyclones, with the underflow reporting to a high-frequency dewatering screen, from where the undersize is recycled back to the sump of the pump that feeds the dewatering cyclones. Fine solids, mainly from the overflow of the second group of fines classifying cyclones of the scavenging circuit, go directly to a 12 m diameter thickener. Thickener underflow is batched to a 2.25 m^2 × 136-plate filter press, which produces a cake suitable for conveyor discharge onto a fines waste stockpile. Overflow from the thickener is recycled as process water. There is no tailings discharge from the process and no tailings dam: all plant waste is dewatered and transported back to the mine waste dumps for disposal.

Final Flotation

The combined concentrates from the gravity circuits (the so-called preconcentrate) is fed into a round screen (via a dewatering cyclone). The oversize (>500 µm) is reground in a 900 × 1200 mm rod mill in close circuit with the screen. The undersize is stored and semidewatered in a settling cone, which feeds a 3 m^3 batch flotation cell where flotation of the sulfides takes place. Flotation spend about 120 min at pH < 4.3 to remove sulfides as a froth concentrate. These sulfides recirculate to the continuous flotation circuit. The non-floating material, which is principally scheelite and calc-silicates, is discharged from the batch cell into a dewatering/settling cone and hence transferred to drying and final processing.

Magnetic Separation

The dry material is subjected to high-intensity, dry magnetic separation in a three-stage rare earth roll (RER) magnetic separator (◘ Fig. 6.55). More than 90% of the material collected by the separator comes out in

◘ **Fig. 6.55** Magnetic separator (Image courtesy of Daytal Resources Spain, S.L.)

the first roll. The mineralogical composition of this product is mainly magnetic pyroxene (hedenbergite). Due to the presence of scheelite and wolframite, the WO_3 content can be as high as 3%. The product that comes out in the second and third rolls is progressively less rich in magnetic pyroxene and with a higher content in WO_3. The intermediate products from the magnetic separator are now dressed on a double-deck table, from which it is possible to obtain a low-grade concentrate, of about 45% WO_3. It is important to notice that in this way, it is possible not only to recover scheelite but also wolframite. Finally, the product is packaged in big bags.

Centrifugal Gravity Concentrators

In order to recover fine particles utilizing gravity separation techniques, some devices have been engineered to apply the centrifugal force. Although these separators have been in practice since the 1800s, their recent resurgence can be devoted to the requirement to economically concentrate minerals such as gold at particle sizes approaching 1 μm at higher throughput capacities (Honaker et al. 2014). Thus, progresses in the last 20 years in the operation of centrifugal gravity concentrators have converted them the prevailing technique for gravity concentration of gold particles and expanded the utilization of this method for other heavy minerals.

Centrifugal gravity concentrators are formed by a riffled cone or bowl that spins at a high speed to generate forces in excess of 60 times that of gravity. Slurry is incorporated into the cone, and the centrifugal force produced by rotation drives the solids toward the walls of the cone. The fluidization procedure anticipates compaction of the concentrated bed and enables for effective concentration of heavy minerals. Thus, the main applications of these devices are (a) recovery of ultrafine slimes (tin, tantalum, tungsten, etc.) ranging from 38 to 5 μm, (b) scavenging from deslime cyclone overflow, and (c) high recovery upgrade of fine flotation concentrates.

6.8.3 Dense Medium Separation

Interest in preconcentration of run-of-mine material, after minimal attrition and prior to fine comminution, has been increasing over recent years. The advantages of ore preconcentration provide opportunities to not only lower operating costs but also lower capital costs by reducing the size of the downstream beneficiation circuit. This process can also reduce the quantity of problematic minerals reporting to downstream flotation and/or leaching processes. Preconcentration is achieved by exploiting a variety of differences in mineral properties such as optical characteristics, magnetic susceptibility, density, radioactivity, and conductivity. Of these, the most common characteristic utilized in preconcentration is the specific gravity differences of the minerals through dense medium separation (DMS). Thus, DMS (also known as heavy media separation (HMS) or the «sink-and-float» procedure) (Fig. 6.56), is one of the most important preconcentration techniques utilized for firstly waste rejection from run-of-mine mineralization at comparatively coarse particle sizes previous to further milling and beneficiation. A good example of this application is in Pering zinc open-pit mine (South Africa), with 50 mt reserves at 1.4% in situ Zn + Pb equivalent grade; simplified DMS procedure decrease the run-of-mine volume by a preconcentrate mass pull of 22% at 5.2 Zn + Pb (Haldar 2013). Heavy media separation has also been used industrially to produce finished concentrate and rejectable

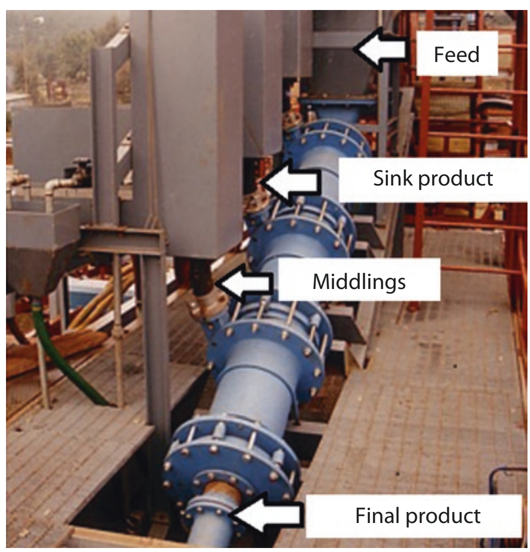


Fig. 6.56 Dense medium separator essentials

waste in one operation or to produce two finished products of differing composition (Aplan 2003).

DMS is probably the simplest of all the gravity procedures and has long been a standard laboratory technique to separate minerals of diverse density. It is utilized on minerals where there is a distinctive density difference between the meaningful minerals and the gangue minerals. If an ore is introduced into a dense medium, which is much higher in density than that of water, some particles are denser (heavier) than the dense medium and sink, while other particles are less dense (lighter) than the medium and float on top of the medium. The bigger the density difference between the valuable mineral and the waste mineral is, the easier the DMS separation should be. The process has the possibility to generate sharp separations at any needed density, with a high degree of effectiveness even in the appearance of high percentages of near-density material (material close to the desired density of concentration). The density of separation can be closely controlled and/or changed as required, although these changes are commonly very expensive. This is mainly due to the additional equipment needed to clean and recycle the medium and the proper cost of the medium.

Since most of heavy liquids such as tetrabromoethane or bromoform are expensive and toxic, their use on a commercial scale is impossible. Thus, the dense medium utilized in industrial applications is a suspension of some dense solid in water, which acts as a heavy liquid. Below concentrations of about 15% by volume, finely ground suspensions in water perform basically as simple fluids. In these procedures, agitation is essential to maintain the suspension and to lower the apparent viscosity. According to Gupta and Yan (2006), any substance utilized for this purpose must have the following characteristics: (a) hardness, it must not easily break down or abrade into a slime under working conditions; (b) chemical stability, it must not be chemically corrosive or liable to react with the ore minerals undergoing treatment; and (c) slow settlement at reasonable viscosity, it must form a fairly stable pulp without having to be ground very fine, otherwise the medium will be too viscous.

The two most common dense powders used for creating dense medium slurries are ferrosilicon (FeSi) and magnetite (Fe_3O_4), although silica, barite, and galena had also been used in the past. Obviously, the density can change by varying water/powder ratio in the blend. Ferrosilicon is an alloy of iron and silicon that contains not less than 82% Fe and 15–16% Si (Collins et al. 1974). It is most widely used for metalliferous ores, whereas magnetite is applied in coal preparation because separation densities are not as high as needed for metalliferous ores. The cause of the utilization of ferrosilicon and magnetite is due to their magnetic properties, which make them easy to remove from the medium by magnetic separation (see ► Sect. 8.5) and recirculate within the plant. Since ferrosilicon has a higher specific density (6.8 g/cm^3) than magnetite (4.5 g/cm^3), it can produce a higher range of relative medium densities. However, ferrosilicon is more expensive. With the objective of achieving a very good recovery and grades, the density of these heavy liquids can be changed, and many densities can be utilized to establish the most suitable density separation cut points.

Preconcentration using DMS (◘ Fig. 6.57) is most commonly carried out on metal mineralization that is related to relatively light country rocks such as silicates and carbonates. Thus, DMS has been widely utilized in iron, lead/zinc, chromite, and tin ores. Other examples of DMS are in the coal, manganese, fluorspar, and diamond industries. For example, the diamond-bearing gravels found along the coast of South Africa are being beneficiated by means of dense medium separation to reclaim the alluvial diamonds (Waanders and Rabatho 2005). The process is carried out by using granular ferrosilicon as the DMS material.

Dense media separators can be categorized into two main groups: gravitational (static baths) and centrifugal (dynamic) vessels. In gravitational units, the feed and medium are introduced into the vessel, and the mixture is gently agitated to maintain a fluidized bed. The less dense minerals are removed by overflow or a paddle, while sink removal varies depending on the vessel. They are largely restricted to treat feeds coarser than 5 mm in diameter. Regarding cyclonic dense medium separators, they are essentially a modified version of hydrocyclones used with a heavy medium as the separating fluid instead of water, being broadly utilized in the processing of ores and coal. This equipment provides high centrifugal force and low viscosity in the medium, allowing

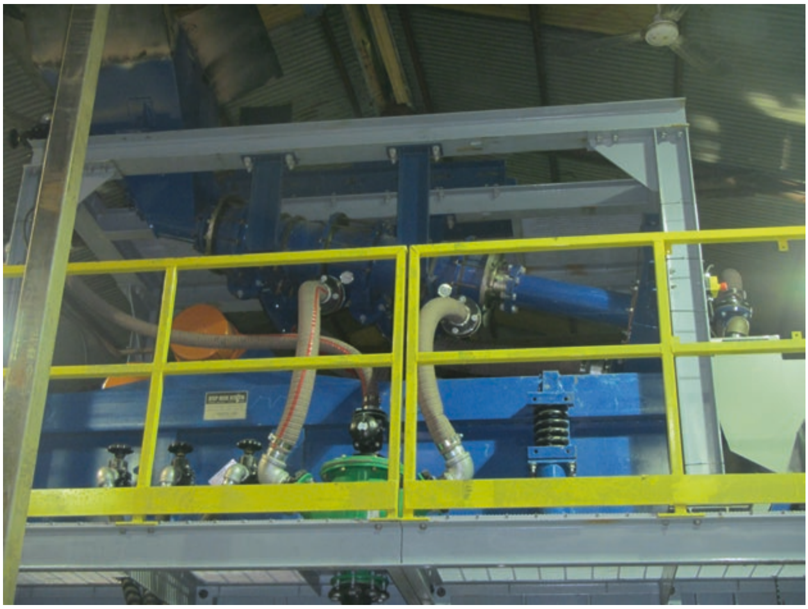


Fig. 6.57 Dense medium separator (Image courtesy of Sepro Mineral Systems Corp.)

much finer concentrations to be obtained than in gravitational devices. Centrifugal units utilize high speed and tangential pumping to create a vortex within the vessel. Any mineral with higher density than the medium will be subject to greater centrifugal forces and be pulled to the outer edge of the vortex, while any lower-density mineral will remain at the center of the vortex. The differing minerals are removed through separate discharge lines.

Although the separating vessel is the most necessary component of a DMS process, it is only part of a more complex circuit. Thus, other equipment is needed to prepare the feed and to remove, clean, and recirculate the medium (Symonds and Malbon 2002). For example, the underflows from washing screens, consisting of medium, wash water, and fines, are too dilute and contaminated to be returned directly as medium to the separating vessel. Therefore, they must be handled by magnetic separation to recover the magnetic ferrosilicon and magnetite from the nonmagnetic fines. Magnetic separators are fed at about 30–35% solids, and the rate is determined by the type of separator and magnetic susceptibility of the medium. In this sense, it is clearly stated that the largest expenditure in any dense medium separation system is for extracting and cleaning the medium that leaves the concentrator with the sink and float products.

6.8.4 Flotation

There are many sentences in the literature to show the importance of froth flotation technique in mineral processing. For example, «flotation is undoubtedly the most important and versatile mineral separation technique; both its use and application are continually being expanded to treat greater tonnages and to cover new areas» (Wills and Finch 2016) or «no metallurgical process developed in the twentieth century compares with that of froth flotation and the profound effect it had on the mineral industry» (Fuerstenau 2007). Recently celebrating its first centenary (the commercial introduction was in Broken Hill mine in 1905), flotation process has allowed the mining of low-grade and complex mineralization that would have otherwise been considered as unvaluable. One technology historian has described flotation as «perhaps the greatest single metallurgical improvement of the modern area» (Mouat 1996). Initially implemented to handle sulfide minerals of copper, lead, and zinc, flotation has spread to incorporate nickel, platinum, and gold-hosting sulfides to non-sulfide minerals including oxides such as hematite and cassiterite and nonmetallic minerals such as fluorite, talc, phosphates, potash and energy (fuel) minerals, fine coal, and bitumen (Rao and Leja 2004).

Flotation (Box 6.6: Principles of Flotation) is therefore the preferred method of mineral recovery for many of the most important minerals that are actually recovered, and large tonnages of ore (billions of tons) are processed by flotation annually. The importance of flotation as a unit operation in mineral beneficiation stems from its relative efficiency and selectivity, its applicability to the extraction of most minerals, and its high throughput capacities per flotation unit (Woollacott and Eric 1994). It is also especially valuable for treating fine-grained mineralization that is not sensitive to classical gravity separation or other beneficiation methods, or where the gravity contrast between minerals is too small (Gupta and Yan 2006).

Box 6.6

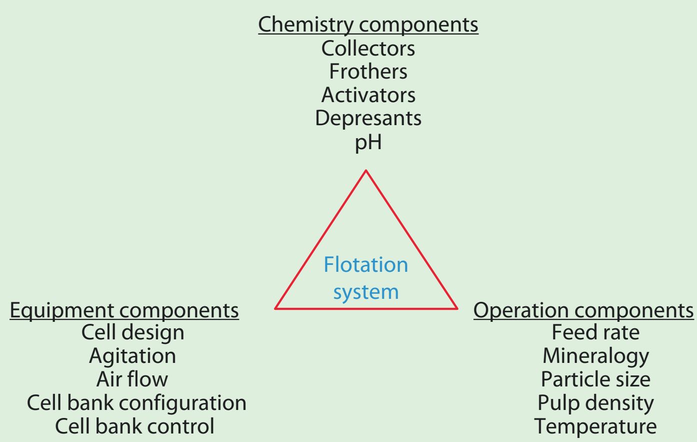
Principles of Flotation

Mineral separation in froth flotation is achieved by the exploitation of differences in the surface properties of the minerals to be separated. Surface properties are very specific to a particular type of mineral because they are determined by its chemical composition and the type of chemical bonding. As these are unique for each mineral, flotation offers a very selective separation capability. The flotation system includes many interrelated components, and changes in one area will produce compensating effects in other areas (Klimpel 1995) (Fig. 6.58). Thus, flotation is a good example of an engineering system, in which the various important parameters are highly interrelated. It is essential to take all of these factors into account in froth flotation operations because changes in the settings of one factor (such as feed rate) will automatically cause or demand changes in other parts of the system (such as flotation rate, particle size recovery, air flow, pulp density, etc.). Thus, froth flotation efficiency is determined by a series of probabilities: those of particle-bubble contact, particle-bubble attachment, transport between the pulp and the froth, froth collection into the product launder, and others.

Very briefly, mineral separations by the process of flotation are achieved in an agitated slurry into which a great amount of air bubbles has been introduced (Fig. 6.59). The system is set up with the aim of achieving the attachment of a selected class of mineral particles to the air bubbles. Particles that are attached to the bubbles will be climbed to the surface of the slurry, thus separating them from particles that do not become attached to bubbles. The particles with attached air bubbles are then removed, whereas the particles that remain completely wetted stay in the liquid phase. Consequently, the steps involved in the flotation process are (a) grinding the ore fine enough so valuable mineral particles become liberated from the waste rock and to a size range suitable to be floated (10–200 µm) – sometimes sizes smaller than 8–10 µm are required; (b) creating a rising current of air bubbles in the pulp; (c) making conditions favorable for the desired mineral particles to adhere to air bubbles; (d) forming a mineralized froth on the surface of the ore pulp; and (e) removing the froth from the flotation cell or vessel.

Although the process can sound relatively simple, many simultaneous subprocesses occur

Fig. 6.58 Interrelated components of the flotation system



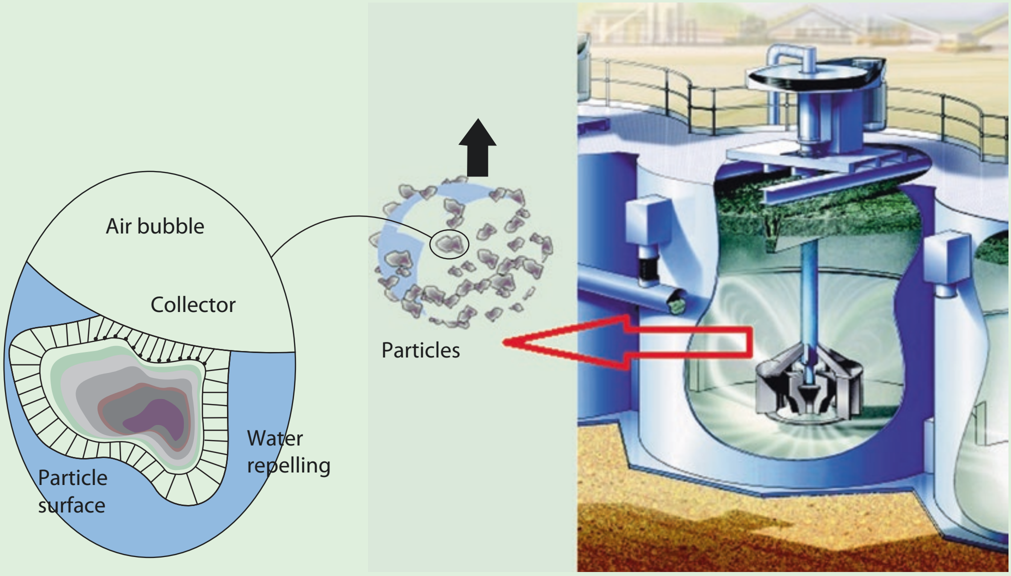


Fig. 6.59 Process of froth flotation

such as entrainment of gangue into the froth phase, coalescence of bubbles, detachment of valuable particles from bubbles as they impact the froth phase, etc. (e.g., Shean and Cilliers 2011). In reality, froth flotation is highly complex, and there are approximately 100 variables that affect to varying degrees of the flotation process (Arbiter and Harris 1962).

Particle-Bubble Attachment

The success of a flotation operation depends on the selectivity of the particle attachment to air bubbles. It is therefore essential to understand the factors that influence particle-bubble adhesion, although the overall process is very complex. The basis of froth flotation is the difference in wettabilities of different minerals. Thus, flotation exploits natural and induced differences in surface properties of the minerals, whether the surface is readily wetted by water, that is, is hydrophilic, or repeals water, that is, is hydrophobic. If a mixture of hydrophobic and hydrophilic particles are suspended in water and air is bubbled through the suspension, the hydrophobic particles will tend to attach to the air bubbles and float to the surface. Thus, the froth layer that forms on the surface will then be heavily loaded with the hydrophobic mineral and can be removed as a separated product. The hydrophilic particles will have much less tendency to attach to air bubbles, and so they will remain in suspension and be flushed away.

Particles can either be naturally hydrophobic, or the hydrophobicity can be induced by chemical treatments. Naturally hydrophobic materials include hydrocarbons and nonpolar solids such as elemental sulfur. Since the bubbles can only be differentiated between hydrophobic and hydrophilic particles, the selective separation depends very much upon differences in hydrophobicity between separated mineral particles. Hydrophobicity can be defined as the process of selectively converting the surfaces of particular minerals from a hydrophilic condition (provided that the mineral is not naturally hydrophobic) to a hydrophobic (water-repellant) one, which creates a condition for attachment to air bubbles. The conversion of the mineral surface from hydrophilic to hydrophobic is variable according to the different nature of the mineral groups (e.g., silicates, oxides, sulfides, etc.) (Bulatovic 2007). Thus, chemical treatments to render a surface hydrophobic are essential methods for selectively coating a particle surface with a monolayer of nonpolar oil.

Once the particles are rendered hydrophobic, they must be brought in contact with gas bubbles so that the bubbles can attach to the surface. The attachment of valuable minerals to air bubbles is the most important mechanism and represents the majority of particles that are recovered to the concentrate (Wills and Finch 2016). No flotation can occur if the bubbles and surfaces never come in contact. Particle/bubble collision is affected by the relative sizes of the particles. If the bubbles are large relative to the particles, then fluid flowing around the bubbles can sweep the particles past without coming in contact. It is therefore best if the bubble diameter is comparable to the particle diameter in order to ensure good particle/bubble contact. Contact between particles and bubbles can be accomplished in a flotation cell.

Where a particle and a bubble have come in contact, the bubble must be large enough for its buoyancy to lift the particle to the surface. The particle and the bubble must remain attached while they move up into the froth layer at the top of the cell. The froth layer must persist long enough to either flow over the discharge lip of the cell by gravity or to be removed by mechanical froth scrapers. If the froth is insufficiently stable, the bubbles will break and drop the hydrophobic particles back into the slurry prematurely. However, the froth should not be so stable as to become a persistent foam, as foam is difficult to convey and pump through the plant. The surface area of the bubbles in the froth is also important. Since particles are carried into the froth by attachment to bubble surfaces, increasing the amount of bubble surface area allows a more rapid flotation rate of particles. At the same time, increased surface area also carries more water into the froth as the film between the bubbles. Since fine particles that are not attached to air bubbles will be unselectively carried into the froth along with the water (entrainment), excessive amount of water in the froth can result in significant contamination of the product with gangue minerals.

Flotation Reagents

Reagents are the most important component of the flotation process since chemicals are required both to control the relative hydrophobicities of the particles and to keep the froth's characteristics. In mineral processing installation, the management of reagent dosages is essential in flotation strategy (Bulatovic 2007). Accordingly, reagents are divided into collectors, frothers, and regulators (◘ Table 6.4). The type of reagent and its dosage can be very varied based on the type of ore. However, the efficient separations by flotation, at least one collector and usually one or more of a variety of flotation reagents, depressants, activators, and pH control reagents, are needed. In addition, it is generally necessary to add one frother, although some circuits have adequate frothing characteristics and do not require a separate addition of this type of reagent. It is important to note that froth flotation is a very complex beneficiation method. For this reason, a flotation collector that works in one processing plant may not work correctly at all in another plan, even though both are floating the same minerals. For this reason, it is common to develop ore-specific and site-specific testing of products needed in the process.

Among the main areas of progress and innovation in special chemical compounds for mineral processing are (a) an increased focus on reagents that can improve mineral recovery and plant throughput; (b) the introduction of more specialized products to assist with environmental, health, and safety compliance; and (c) reagents that are developed utilizing renewable feed stocks (Goodbody 2011).

Collectors (◘ Table 6.4) are a big group of organic chemical compounds that differ in chemical features and function, being probably the most critical of the flotation reagents. They are polar molecules that attach to the mineral fragments and render them hydrophobic so that they are carried upward with the gas bubbles. These reagents are attached to the pulp in the conditioner tank and ball mill. It is important to remember that most minerals do not easily float in froth flotation, as they are hydrophilic, so the use of chemical reagents known as collectors is required to make them float. For example, sulfide minerals are hydrophilic under the usual conditions encountered in processing because they are readily oxidized when exposed to air. In the case of oxide and silicate minerals, all but two are hydrophilic (Fuerstenau 2007).

Based on the capability of collectors to dissociate in water, collectors can be classified into two principal groups: ionizing collectors and nonionizing collectors. The first consist of heteropolar organic molecules. They are soluble and have wide applications, being also classified according their major application: non-sulfide minerals or sulfide minerals. Nonionizing collectors, which are practically insoluble and strongly hydrophobic, are again separated into two groups. The xanthate family is the most widely used collector compound. It is applied to about three-quarters of global mining applications (Goodbody 2011). Xanthate and other standard collectors only float sulfides rather than oxide mineral species. Thus, if a mine has a mixed ore containing sulfides and oxides, the oxides would be lost to the tailings dam if another type of collector was not included as well.

Frothers are heteropolar surface-active compounds that increase the surface tension of the bubbles so that they are stable and can carry the

Table 6.4 Main types of flotation reagents

Collectors
Nonionizing
Liquid hydrocarbons
Ionizing
Anionic collectors
Xanthates
Dithiophosphates
Fatty acids (oleic acid)
Cationic collectors
Amines
Frothers
Aliphatic alcohols
Alcoxy paraffins
Phenols
Sulfonates
Pine oil
Methyl isobutyl carbinol (MIBC)
Polyglycol ethers
Triethoxybutane (TEB)
Cresylic acid
Modifiers
pH modifiers
Lime
Caustic soda
Soda ash
Sulfuric acid
Depressants
Sulfide dioxide
Sodium cyanide
Sodium sulfide
Sodium silicate
Starch/dextrin
Sulfur dioxide
Activators
Sodium hydrosulfide
Copper sulfate
Multivalent ions

mineral particles upward (Table 6.4). They have three main functions in flotation: (a) aid formation and preservation of small bubbles, (b) reduce bubble rise velocity, and (c) aid formation of froth (Klimpel and Isherwood 1991). Reduction in bubble size increases the number and total surface area of bubbles, which increases collision rate with particles and thus increases flotation kinetics. Reducing rise velocity increases the residence time of bubbles in the pulp, which in turn increases the number of collisions with particles. The third function, aid formation of froth, means the bubbles do not burst when they reach the top of the pulp, which enables the collected particles to overflow as the float product (Wills and Finch 2016). Initially, natural oils such as pine oil were used as frothers, but their use diminished over the years, alcohols and polyglycols being the major commercial frothers today.

Activators, depressants, and pH regulators are often referred to modifiers or regulators of the flotation process (Table 6.4). The main objective of these components is to change the action of the collector on mineral surfaces, and therefore they control the selectivity of the flotation process. The main use of activators and depressants reagents is in the differential flotation process. Thus, activator reagents cause flotation or certain ore minerals. On the contrary, other compounds inhibit or prevent the adsorption of a collector by a mineral particle and consequently prevent its flotation: these reagents are called depressants. This is because some gangue minerals have hydrophobic surfaces, quickly float, and contaminate the mineral concentrate; depressants are used to avoid this issue adsorbing on the surface of the gangue and stop them from floating. The utilization of depressants produces higher recovery rates and grades. It is conventional in non-sulfide flotation systems to use naturally derived substances such as starches and gums as depressants.

The third group of regulators or modifiers is pH regulators. The purpose of these is to adjust the pulp to the best pH range for separation of specific minerals by changing the concentration of the hydrogen ion in the pulp. The most common pH regulators are lime, soda ash, and sulfuric acid. These compounds are commonly utilized in significant quantity in almost all flotation operations.

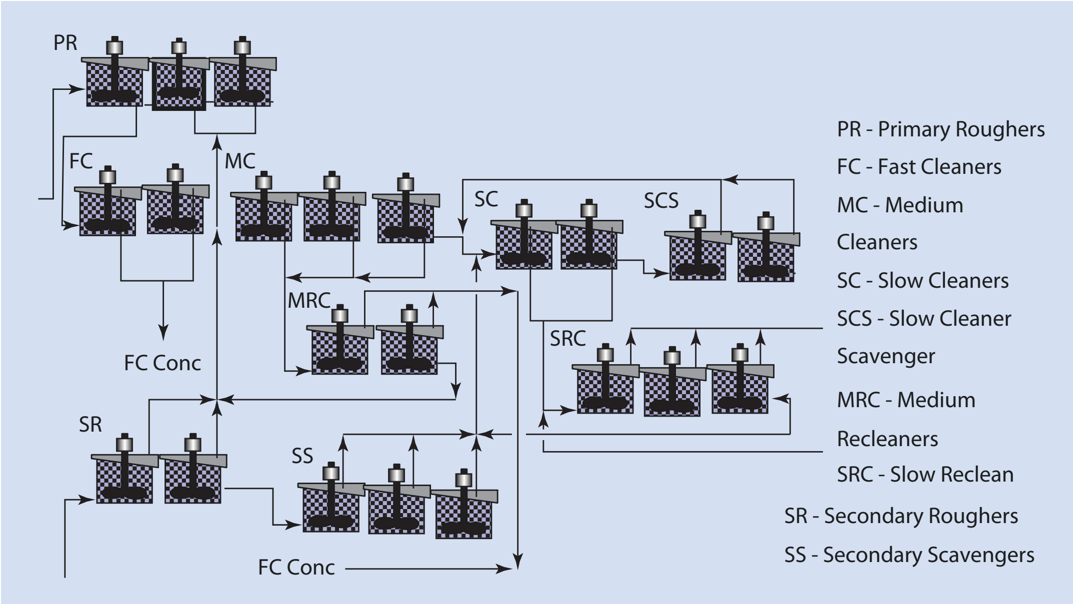


Fig. 6.60 Example of different stages in froth flotation

Flotation Stages

Flotation is usually carried out in different steps to maximize the extraction of the target minerals and their separation in the concentrate while reducing the energy consumption (Fig. 6.60). The first step is termed roughing, which generates a rougher concentrate. The aim is to extract the maximum quantity of meaningful components at a coarsest particle size as possible, with less interest on the purity of the concentrate originated. Thus, the initial beneficiation process using froth flotation is undertaken in the roughers. For roughing, the entire separation of particles is not needed, only adequate liberation to release enough gangue components from the meaningful mineral to get a high-recovery process. In some cases, there can be a preflotation stage prior to roughing. This is to carry out if some undesirable components that readily float are present in the product. They are extracted first in preflotation to prevent them from floating in the roughing, which contaminates the rougher concentrate. The rougher concentrate is commonly ground, termed «regrinding,» to obtain a more definitive separation of the meaningful minerals.

The rougher concentrate is normally subdued to subsequent steps of the flotation to remove a greater quantity of the troublesome minerals that also are included in the froth, in a process termed «cleaning» (in some cases, two or three steps of cleaner flotation). The obtained product of cleaning is usually called the cleaner concentrate or the final concentrate. Thus, the aim of the cleaning stage is to generate as high concentrate grade as possible. The waste material from the roughers can be directed to the so-called scavenger stage, in which some flotation cells are utilized to «scavenge» any remaining valuable particle from the waste before it is sent to the tailings. Logically, the valuable particles removed from the scavenger circuit move back newly to the rougher and cleaner stages. This process is highly reagentized, and so the concentrate is pulled off much rapid than in the roughers. The main differentiation between the concentrates from the roughers and scavengers is that the latter include both coarse and fine components, while the rougher concentrate is mainly formed as an intermediate-sized fraction.

Flotation Circuits

Commercial flotation is a continuous process, but it cannot be carried out in a single cell because of the losses due to short circuiting of pulp between the feed and pulp discharged. Thus, cells are arranged in series forming a

Fig. 6.61 Bank of flotation cells (Image courtesy of Iberpotash)

bank, being commonly to use 4–12 cells in series (Fig. 6.61). The disposition of several flotation cells in series enables extraction of different products from the various cells (rougher, scavenger, and cleaner). This increases the floating time, enabling the opportunity for particle-bubble attachment to carry out. The residence time of components in the bank of cells usually is in the range between 5 and 15 min (Gupta and Yan 2006). In general, separated particles float more quickly than composite particles. Thus, a high-grade concentrate can be delivered from the early some cells in a bank and froth from the rest cells can only generate a middling concentrate.

With the higher tonnage of lower-grade mineralization actually being operated by the minerals industry, the general trend in the market is toward large-volume flotation cells. The large cells offer important advantages such as decreased plant footprint, lower energy consumption, decreased maintenance and operating expenditures, and simpler control. In this sense, effective froth transportation and recovery is critical for efficient operation of large cells (Gorain 2016). By early 2000, the cell sizes had increased up to 200 m^3. Cells as large as 700 m^3 are actually manufactured by some flotation cell companies. As a general rule, flotation cells for utilization in an installation must be elected in accordance to laboratory and pilot-scale data.

The most complex flotation circuit is to select flotation of a mixing of different mineral, that is, the extraction in a sequential manner of two or more meaningful minerals from each other by flotation (e.g., separation of each copper, lead, and zinc sulfides from a single mineralization) (Box 6.7: Aguas Teñidas Polymetallic Sulphide Processing Plant). Obviously, this type of circuits is very expensive in both capital and operating cost requirements. In these installations, the mineralization is early managed to enable one species to be floated, while the other(s) stand hydrophilic with the gangue particles. Depressants are included to help in concentration with the quantities of collector and frother being constantly operated to generate conditions just enough to remove the desired mineral(s).

Box 6.7

Aguas Teñidas Polymetallic Sulfide Processing Plant (Huelva, Spain): Courtesy of Matsa, a Mubadala & Trafigura Company

The Aguas Teñidas mine is based on one of the east-west striking chains of volcanogenic massive sulfide (VMS) deposits on the northernmost limb of the Iberian Pyrite Belt. The mine geology is comprised of heavily tectonised volcano-sedimentary sequences, with cross-cutting thrust faults and shear zones. The main lithological units at the mine comprise a footwall rhyodacitic unit, massive sulfide mineralization, and a hanging wall volcano-sedimentary unit. The deposit includes four mineralization types: polymetallic lead/zinc rock, massive cupriferous, barren pyrite, and a cupriferous stockwork. The principal ore minerals are sphalerite, chalcopyrite, and galena. Pyrite generally forms 50–80% of the massive sulfide. Both massive sulfide ores (polymetallic and cupriferous) are hosted in a massive pyrite structure and are identified from the pyrite host rock by grade rather than any physical differences.

The polymetallic ore and copper ore are treated in separate process circuits. This is due to the differing mineralogical properties and consequent differences in target grind size for the two mineralization types as well as the logistical difficulties that would arise in maintaining a consistent feed blend and grade from the respective ore types to the plant. The polymetallic mineralization is best defined as massive sulfide with minor non-sulfide minerals consisting of silicates mainly quartz, muscovite, and chlorite. The main sulfide minerals are pyrite, sphalerite (zinc), and galena (lead) with lower levels of chalcopyrite (copper), arsenopyrite, tetrahedrite (copper-antimony), and tennantite (copper-arsenic). The following description is devoted to the polymetallic ore processing (Fig. 6.62).

Crushing

Run-of-mine (ROM) ore is transferred from underground via the main haulage ramp (Santa Barbara ramp) to the ROM ore stockpile. It has the capacity to store approximately 10,000 t of ore (Fig. 6.63) in an 8000 m^2 surface. Ore is fed by a front-end loader to an open feed bin with an approximate capacity of 50 t of ore. This ore feed bin flows directly to a vibrating grizzly feeder which scalps fines (<150 mm) with coarse ore from the vibrating grizzly flowing into a 1.1 m × 1.2 m crusher which produces a <150 mm crushed product. Crushing rate is nominally rated at 300 tons per hour (tph). The crushed ore is then conveyed to a covered stockpile, being the conveyors covered for dust control and to avoid contamination of the environment. The stockpile has a nominal live capacity of 6600 t, equivalent to 3 days production.

Grinding

Ore is delivered to the copper ore grinding circuit at a rate of 106 tph via a variable speed conveyor that is fed by up to four vibrating feeders located under each covered stockpile. The grinding circuit is a three-stage grinding process that consists of a semiautogenous grinding (SAG) mill followed by a ball mill operating with hydrocylones. Finally, a tertiary grinding stage uses stirred mill detritors (SMD). The P_{80} transfer size between the SAG mill and the ball mill will be approximately 1000 and 100 μm between the ball mill and the SMD. The target P_{80} grind size from the grinding circuit to the copper flotation circuit is 18–22 μm. The SAG mill is 5.5 m in diameter, and an average ball charge of between 4% and 8% is expected with 90–125 mm grinding media (steel balls). The SAG mill discharges through a rotating trommel screen with oversize material recirculated via a conveyor arrangement over a weightometer and back to the mill feed. The undersize reports to the ball mill discharge sump where it is combined with the ball mill discharge and pumped to a two-stage hydrocyclone classification system. The target overflow density and P_{80} grind size from the SAG/ball mill circuit is 35% w/w and 100 μm, respectively, and shall feed three SMD mills. The ball mill is 3.6 m in diameter, and the ball charge varies between 40% and 45% of the ball mill volume using grinding media (steel balls) of 40–25 mm in diameter. The first-stage cyclone (500 mm in diameter) underflow is returned to the ball mill feed, and the first-stage hydrocyclone overflow flows to a pump box feeding the second-stage hydrocyclone cluster. The second-stage hydrocyclone underflow flows to the three SMDs, and the cyclone overflow at a pulp density of 35% w/w solids and with a P_{80} of 18–22 μm will gravitate to the bulk rougher flotation circuit. The second-stage hydrocyclone overflow flows by gravity to the polymetallic bulk flotation circuit. The zinc minerals (sphalerite) require a fine grind in order to achieve a metallurgical recovery in excess of 75% at the rougher flotation stage and a minimum zinc concentrate grade of 50% Zn.

Flotation

Polymetallic ore is processed by sequentially floating (Fig. 6.64) a bulk copper and lead concentrate (bulk concentrate) followed sequentially by zinc flotation prior to discarding the final polymetallic tailings. The bulk concentrate is cleaned and will be processed in a copper/lead separation circuit where two concentrates are produced: a lead concentrate and a copper concentrate. The bulk, copper and lead separation, and zinc flotation circuits use a three-stage cleaning process. The purpose of the bulk flotation circuit is to separate copper and lead minerals producing a high-grade copper and lead concentrate that can be

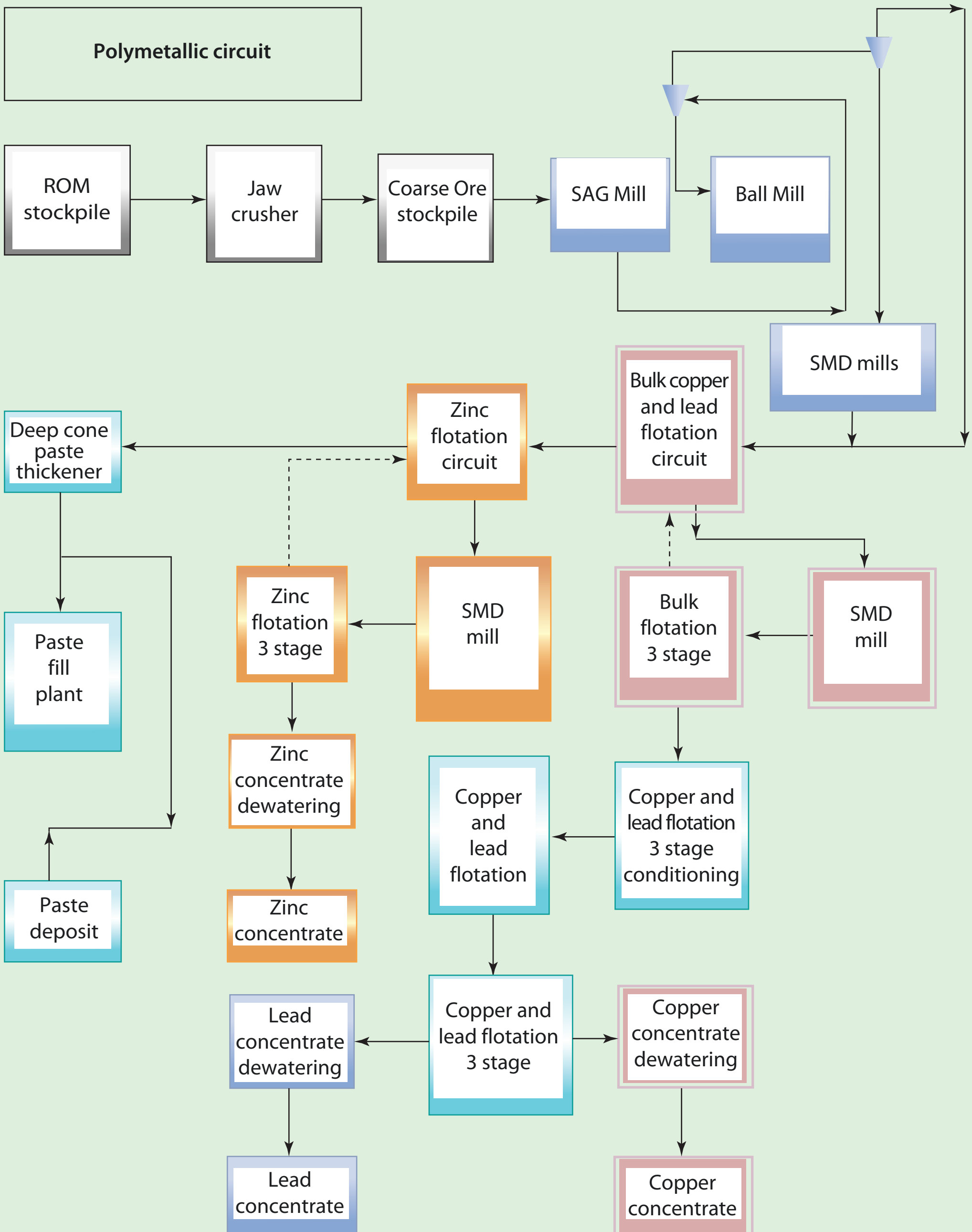


Fig. 6.62 General flow sheet of polymetallic sulfide processing plant

further processed by a copper and lead flotation separation circuit.

Reagents used in the bulk concentrate flotation process include potassium amyl and ethyl xanthate, Aero 3418 flotation collector/promoter, zinc sulfate (zinc mineral depressant), and lime as required for pH control. Sodium metabisulfite is used to depress pyrite at this time. Once a final bulk flotation concentrate is produced, this product will be processed for separation, and no recirculation of concentrate rejects (middlings or tailings) returns from the separation circuit to the bulk flotation circuit. The bulk flotation circuit maximizes copper and lead recovery from the polymetallic ore and minimizes contamination of these

Fig. 6.63 Covered stockpile (Image courtesy of Matsa, a Mubadala & Trafigura Company)

Fig. 6.64 Froth flotation of polymetallic ore (Image courtesy of Matsa, a Mubadala & Trafigura Company)

same elements in the zinc flotation circuit. Ideally, the ratio of lead to copper in the feed of polymetallic ore to the bulk rougher flotation circuit is greater than 2:1 to optimize the concentrate grades required to produce saleable copper and lead concentrates after separation.

Meanwhile, the purpose of the copper and lead separation flotation circuit is to separate the copper and lead from the bulk flotation concentrate to produce saleable copper and lead concentrates. The copper and lead flotation separation is accomplished by depressing the lead mineral (galena) using a mixture of sodium dichromate or sodium bisulfite mixed with the cellulose and phosphate compounds to produce a reagent called RPB that is added at the conditioning stage to depress galena and pyrite minerals. Prior to the addition of RPB depressant, the bulk concentrate is contacted with fine activated carbon to remove all prior reagents used to produce the bulk concentrate, which ensures the best possible separation using the RPB depressant. No regrinding of concentrates is done in the copper and lead flotation separation circuit.

Combined bulk flotation circuit rougher scavenger and cleaner scavenger tailings discharge into twin-stage zinc rougher conditioning tanks where the slurry is conditioned with flotation reagents and lime. The first-stage conditioning is with copper sulfate to activate the zinc minerals, and the second stage will be with zinc collector and promoter reagents. The zinc flotation circuit is operated at a pH of 10.5, as compared to pH 8–8.5 in the bulk flotation circuit. The elevated pH is required to depress pyrite.

Dewatering Systems

The polymetallic copper and lead thickeners are center-driven and 6 m in diameter. Flocculant is added to the copper, and lead concentrate slurries as it feeds the thickeners to improve settling rates and maintain a clear overflow. The concentrate thickener underflows are transferred by variable speed centrifugal pump into the respective copper and lead concentrate stock tanks which feed their respective pressure filters (Fig. 6.65). Concentrate thickener overflow are collected in a circumferential launder and flow by gravity into their respective overflow collection tanks for return to the main process water tank.

The concentrate thickener underflow pumps transfer the copper and lead concentrate into the stock tank ahead of filtration. The mechanically agitated tanks are sized to hold 18 h of slurry storage at the design plant concentrate production rate. The filters produce a cake within the required transportable moisture limit (TML) for shipment. This is as low as possible to reduce transport costs and will be approximately 7–8% w/w. The filter cake discharges from the plate, and frame filter presses (pressure filter) fall through chutes into the concentrate storage area, which is a fully enclosed area. The copper and lead concentrate storage capacities are approximately 1000 t, equivalent to 7 days production at the design plant production rate. An FEL will move the concentrate if it is to be stacked away from the immediate filter discharge zone and will load the haulage trucks when required. The loaded haulage trucks pass through a wheel wash system to avoid concentrate dust polluting the outside environment. The

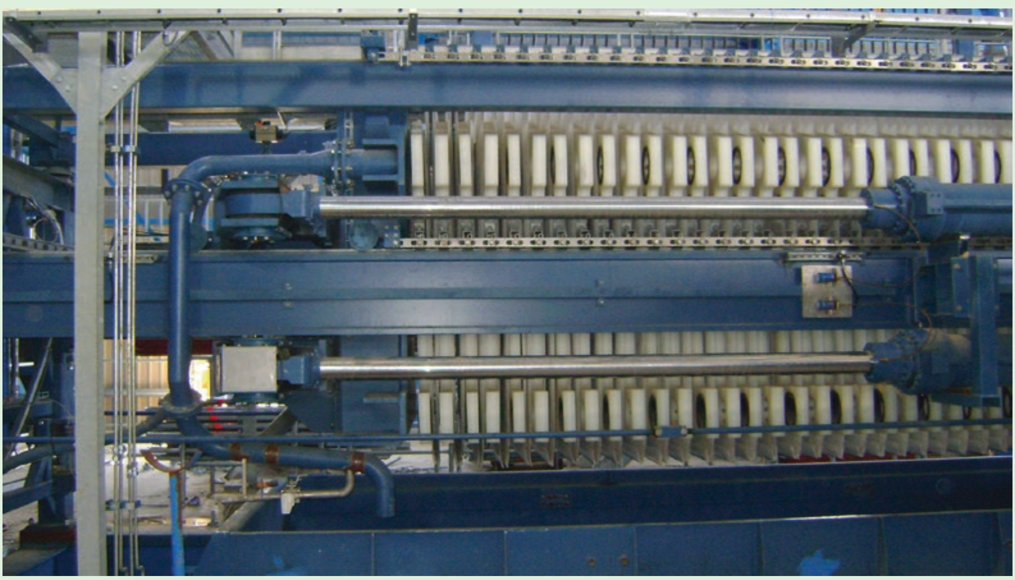


Fig. 6.65 Pressure filters (Image courtesy of Matsa, a Mubadala & Trafigura Company)

trucks are weighed, both empty and loaded, on the site weighbridge to determine the tonnage of concentrate hauled. Samples from each truck are taken for the assay of the moisture and metal content.

In turn, the zinc concentrate slurry from the third-stage cleaner flotation cells is pumped to the concentrate thickener feed box. The thickener is center-driven and has been sized at 10 m in diameter. Flocculant is also added to the concentrate slurry as it feeds the thickener to improve settling rates and maintain a clear overflow. The concentrate thickener underflow is transferred by variable speed centrifugal pump into the zinc concentrate stock tank that feeds the pressure plate and frame filter. Concentrate thickener overflow is collected in a circumferential launder and flows by gravity into the overflow collection tank for return to the main process water tank. The concentrate thickener underflow pumps transfer the zinc concentrate into the zinc concentrate stock tank ahead of filtration. The mechanically agitated tank is sized to hold 300 t (approximately 18 h of production) of concentrate slurry storage at the design plant concentrate production rate. The filters produce a cake with 7–9% w/w of moisture. The filter cake discharges from the presses and falls through chutes into the concentrate storage area, which is also a fully enclosed area. The zinc concentrate storage capacity is approximately 3000 t, equivalent to 7 days production at the design plant production rate.

Paste Processing

Combined flotation tailings with a pulp density of 28–30% w/w solids is pumped directly to a deep cone thickener feed box. An 18 m in diameter deep cone thickener produces a thickened underflow of 70–75% w/w solids that is pumped to a cemented paste fill plant located approximately 2 km from the process plant. Alternatively, where the cemented paste fill plant is not in production, the thickened tailings are pumped to an HDPE geomembrane lined paste deposit located within 1 km of the process plant where the paste tailings is stored permanently on surface. The thickener overflow solution gravitates to a hopper where it is pumped to a process water pond prior to being returned to the process water tank located at the process plant.

6

Flotation Equipment

Flotation machines utilized in processing plants can be separated into two main groups: mechanical and nonmechanical (pneumatic, column, and froth separators). Of all these, mechanical flotation devices have controlled the mineral industries all over the world since the first days of froth flotation (Gorain 2016). Mechanical flotation cells are the most commonly used in the flotation of metallic minerals (◘ Fig. 6.61). They consist of a highly turbulent zone generated by an impeller (◘ Fig. 6.66), which offers the sufficient agitation to maintain the particles in suspension, disperse the air bubbles, and bring about particle-bubble contact. The design of the agitator and gas dispersion mechanism has an effect on the efficiency of the kinetic processes that control the rate of flotation. As a rule, mechanical cells are easier to operate than, for instance, flotation columns. Thus, they are the selected option by many design engineers and operators.

Nonmechanical cells are evermore accepted by the minerals industry principally in situations where the classical mechanical devices do not achieve the objectives. Some examples of these applications are flotations of ultrafine particles (<10 μm) or particles greater than 250 μm, obviously always based on the features of the mineralization. Typical nonmechanical cells are column flotation and Jameson cell, although new technologies include Microcel, CavTube, and Imhoflot design. Microcel and cavitation devices are two fine bubble generation technologies that have gained popularity. They have helped columns to compete with the new generation of mechanical cells concerning better unit extractions in cleaner applications.

Column flotation (◘ Fig. 6.67) cells were first satisfactorily utilized in the market during the early 1980s. It is essentially a pneumatic device in which the slurry is deliberately not agitated, being the bubbles generated in specially designed bubblers. Moreover, the maintenance of particle suspension by different methods is not necessary. Thus, a column basically works as if it was a multistage flotation system designed vertically with slurry flowing downward while the air bubbles move upward (Degner and Sabey 1988). Under normal operating conditions, the column volume is split into two distinct regions according to their air content (volume fraction): a collection or pulp zone (less than 20% of air) and a cleaning

Fig. 6.66 Flotation cell impeller (Image courtesy of Daytal Resources Spain S.L.)

Fig. 6.67 Column flotation cells

or froth zone (more than 70% of air) (Bouchard et al. 2009). In specific cases (e.g., handling of very fine particles), the column flotation circuit presents diverse advantages such as enhancing material throughput and control, comparatively low power expenditures, and less floor area and service (Metso 2015).

The Jameson flotation cell was introduced at Mount Isa Mines (Australia) in the late 1990s. It is a pneumatic flotation cell that displays important advantages in comparison with mechanical and column cells comprising kinetics, footprint, and expenditure. Jameson cells are slowly being accepted in the lead, zinc, copper, and copper-gold operations (Young et al. 2008). In the Jameson cell, air or bubbles are incorporated into the slurry through the pipe that feeds the cell. This pipe is situated vertically so that particles and bubbles move downward. Froth washing is carried out in the same way as for column flotation.

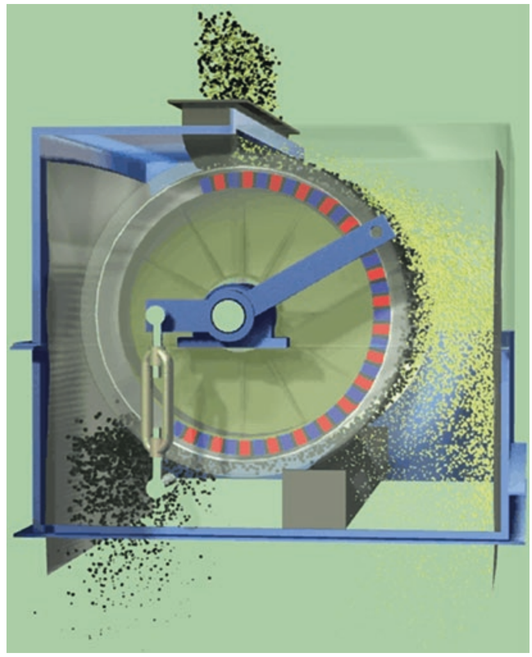
Flotation Circuit Design and Optimization Using Modeling and Simulation

There are different cases of undervaluation of flotation capability leading to the processing plant not being able of finding the mineralurgical design objective. This generates a risky position that causes loss of opportunities due to production losses and the requirement of subsequent capital investment for further flotation capability. The reason for high risk in classical designs is that the safety features depend on prior experience on simpler mineralization, while most of the present mineral deposits are minerallurgically complex needing an in-depth knowledge of the mechanisms that drive flotation throughput (Gorain 2016). Flotation modeling and simulation methods have newly developed as significant set of tools to offer a sounder basis to design and optimize flotation systems (e.g., Herbst and Harris 2007; Connolly and Dobby 2009). This effort has been carried out by several of the major mining companies principally to increment the confidence level in designing flotation systems with very low risks.

Once a complete modeling process has been carried out for a mineral deposit, simulations can be done to comprehend the influence of mineralization changes and circuit configuration on flotation performance. However, flotation modeling and simulation methods are not still perfect due to issues in modeling very complex mineralization types. It is possible that through continued improvement of new and better technologies (e.g., machine vision and air recovery estimation) and continuous simplification/modifications of processing plant operations, needing less-intricate control systems, long-term, automated progress, and optimizing flotation control will be attainable. Such result would indeed be financially rewarding (Shean and Cilliers 2011).

6.8.5 Magnetic Separation

Knowledge of magnetic forces dates back at least to the Greek philosopher Thales of Miletus, who lived about 600 BC. Thales knew some of the magnetic properties of the mineral lodestone, and he was also aware where amber would attract light, nonconducting particles (Venkatraman et al. 2003). The concentration of minerals in accordance with their magnetic susceptibility is a broadly utilized separation method that can be a very effective separation process (◘ Fig. 6.68). Moreover, magnetic separation is usually a low-cost technique of retrieval unless high-intensity separators are required.



◘ **Fig. 6.68** Magnetic separation process (Illustration courtesy of Metso)

Magnetic separators use the difference in magnetic properties between the minerals in a deposit. They are in use in many installations since they can be very effective, comparatively non-expensive, and more environment friendly than other methods. Therefore, where looking for a procedure to recover meaningful minerals, magnetic separation should not be ignored. Magnetic separators are used to concentrate a valuable mineral that is magnetic (e.g., magnetite from quartz), to remove magnetic contaminants, or to separate mixtures of magnetic and nonmagnetic valuable minerals. An example of the latter is the cassiterite, commonly associated with some amounts of the meaningful magnetite or wolframite, which can be extracted by magnetic separators. However, the most typical application of magnetic concentration is for iron ore processing in the minerals industry.

All materials are impacted in some way when situated in a magnetic field, although with many components, the result is too slight to be easily detected. Where minerals are situated in a magnetic field, there are three reactions that can take place. Firstly, the minerals attracted to the magnetic field: these particles are termed magnetic. There are two types of magnetic particles, strongly magnetic particles or ferromagnetic particles such as iron and

Table 6.5 Susceptibilities of some minerals

Mineral	Magnetic susceptibility ($X_m \times 10^6$ emu/g)	
Magnetite	20,000–80,000	Ferromagnetic (strong magnetic)
Pyrrhotite	1500–6100	
Hematite	172–290	Paramagnetic (weakly magnetic)
Ilmenite	113–271	
Siderite	56–64	
Chromite	53–125	
Biotite	23–80	
Goethite	21–25	
Monazite	18.9	
Malachite	8.5–15.0	
Bornite	8.0–14.0	
Rutile	2.0	
Pyrite	0.21	
Cassiterite	– 0.08	Diamagnetic (repelling)
Fluorite	– 0.285	
Galena	– 0.35	
Calcite	– 0.377	
Quartz	– 0.46	
Gypsum	– 1.0	
Sphalerite	– 1.2	
Apatite	– 2.64	

magnetite, and weakly magnetic particles such as pyrite and copper sulfides. Ferromagnetic minerals have very high susceptibilities (Table 6.5) and experience very strong magnetic forces in a nonuniform field. Thus, they can be easily separated with a device displaying a low-intensity magnetic field of 400–600 gauss. However, it is not possible to separate between different ferromagnetic minerals because they all experience the same separating force. In turn, weakly magnetic particles or paramagnetic particles need a higher intensity magnetic field to concentrate them, commonly ranging from 6000 to 20,000 gauss. Paramagnetic minerals report to the magnetic product of a magnetic separator due to the attractive magnetic forces.

The second option includes the particle's repulsion by the magnetic field (diamagnetic materials) or very weak separating forces in a strong magnetic field. Diamagnetic minerals will report to the nonmagnetic product of a magnetic separator as they do not experience a magnetic attractive force. The third possibility is that no noticeable reaction to the magnetic field occurs. Nonmagnetic particles such as gold or quartz are not susceptible to magnetic separation, but some magnetic material can be removed from the feed. For example, in installation utilizing gravity separation to concentrate gold, magnetic devices are used to extract the high amounts of magnetite that were recovered with the gold, prior to further processing.

Types of Magnetic Separators

There is a broad variety of devices for concentrating mineralization in accordance with their magnetic properties. The principal distinguishing feature is the magnetic field strength, with main ranks of low, medium, and high intensity. Although there are no clear limits to define each category, a good approximation is the following: (a) low intensity, 0–0.2 tesla (0–2000 gauss); (b) medium intensity, 0.2–0.7 tesla (2000–7000 gauss); and (c) high intensity, 0.7 tesla and up. Magnetic devices can be both wet and dry and with different configurations that enable beneficiation of coarse lumpy elements down to micron-sized particles. Wet magnetic separation has found major application in the treatment of fines at iron ore operations. One of its attractions is its ability to handle much wider and finer size range than dry systems. It is also applied in the cleaning of heavy media suspensions such as ferrosilicon (Lyer 2011).

There are actually tens of thousands of low-intensity magnetic separators (LIMS) (Fig. 6.69) and thousands of high-intensity magnetic separators (HIMS) utilized in mineral processing (Gorain 2016). The continuous wet high-intensity magnetic separators (WHIMS) were developed later in the 1960s to remove moderately magnetic particles from slurries. The high-gradient magnetic separators (HGMS) were also introduced in that time principally for treating kaolin clay. This device allowed to process mineralization

Fig. 6.69 Low-intensity magnetic separators (Image courtesy of Metso)

with a high content (>10%) of weakly magnetic material since weakly paramagnetic minerals can only be effectively recovered using high-intensity magnetic separators. The development of HGMS and superconducting separators capable of concentrating very fine or very weakly magnetic mineral particles has prompted the application of magnetic separation techniques to treat many waste streams from mineral processing operations (Wills and Finch 2016). For example, fine (<10 μm), weakly magnetic hematite and limonite have been recovered by a combination of selective flocculation followed by HGMS (Song et al. 2002). HGMS has also been used to recover fine gold-bearing leach residues from uranium processing. Currently, superconducting magnets are the only economically and technically viable way to achieve field strengths as high as 5 teslas.

An important magnetic separation improvement that generates major impact in the minerals industry was the drum separator for magnetite mineralization. It principally treats fine-grained and low-grade magnetite ores for providing high-grade concentrates and also for direct reduction iron processes (Arvidson and Norrgran 2014). In general, this machine commonly produces extremely clean magnetic concentrates. The rare earth drums (RED) are also actually being prevailing for the separation of some paramagnetic minerals such as hematite and ilmenite at a relatively high capacity (Gover et al. 2011). During the 1980s, the rare earth roll (RER) magnetic devices were introduced in the industry with quick acceptance. These machines are generally the best option for high-intensity separation for new operations now (Gorain 2016).

6.8.6 Electrostatic Separation

In electrostatic separation (ESS), also called electrical separation, particles come under the influence of an electrical field. They gather a charge that relies on the maximum attainable charge intensity and on the surface area of the particle. These charged components can be split by differential attraction or repulsion. Therefore, the important early stage in electrostatic separation is to impart electrostatic charge to the particles. The three principal manner of charging actions are contact electrification or triboelectrification, conductive induction, and ion bombardment. Where the particles are charged, the split can be carried out by devices with different electrode configuration. Due to nearly all minerals displaying some contrast in conductivity, this technique would represent the universal beneficiating technique. However, the method has limited applications due to the required processing conditions, notably a perfectly dry feed (Wills and Finch 2016). Minerals must be definitely dry, and the humidity of the surrounding air must be controlled, since the electron moving in dielectrics carries out on the surface and a film of moisture can change the behavior absolutely. Moreover, the main disadvantage of this technique is that the capacity of valuable sized units is low.

Electrostatic separation is principally utilized in mineral sands separation. Thus, conductive minerals such as ilmenite, monazite, and rutile are concentrated from nonconductive silica and zircon existing in mineral sands. Thus, the electrostatic separator is still the most reliable and economic unit operation for processing beach sand deposits rich in minerals such as ilmenite, rutile, leucoxene, zircon, and garnet (Box 6.8: Chandala Heavy Mineral Sands Processing Plant). The two main types of electrostatic separators in mineral sands have been a combination of high-tension rolls (HTR) and electrostatic plate separators (ESP), being principally utilized in Australia and South Africa (Gorain 2016). The stream particles are wanted between 75 and 250 µm, dry, closed-size distribution, and similar for effective separation to carry out (Haldar 2013). As another application of these devices, increased environmental awareness has promoted the demand for unit operations that process secondary materials (e.g., to remove plastics from metals) (Lyer 2011).

Box 6.8

Chandala Heavy Mineral Sands Processing Plant (Perth, Australia): Courtesy of Tronox Ltd.

The Chandala complex includes three major plants: a dry mill, which separates the heavy mineral sands; a synthetic rutile plant, which upgrades ilmenite into high-quality synthetic rutile; and a residue management plant. Chandala produces approximately 450,000 tons of ilmenite, 80,000 tons of zircon, 37,000 tons of rutile, and 20,000 tons of leucoxene per year. Zircon, rutile, and leucoxene are either bagged or sold in bulk. Ilmenite is further processed into synthetic rutile using reduction, aeration, and acid leaching.

Chandala dry separation processing plant beneficiates the Cooljarloo heavy mineral deposits that lie within the Perth Basin in Western Australia. The detrital heavy minerals of the Perth Basin include ilmenite, rutile, and zircon, which were derived from igneous and metamorphic rocks in the adjacent Archaean shield to the east in the interior of Western Australia, concentrated in nearshore sediments through multiple phases of weathering, erosion, and deposition. Most of the high-grade heavy mineral deposits at Cooljarloo occur as shoreline accumulations comprising detrital ilmenite, rutile, leucoxene, and zircon with subordinate monazite and a gangue of aluminosilicates like kyanite, staurolite, andalusite, and tourmaline.

The different magnetic and electrostatic properties of the mineral sands as well as their different specific gravities are used to separate the minerals from each other and from the waste. Minerals that are magnetically susceptible (magnetics) can be separated in special equipment from minerals that are not magnetically susceptible (nonmagnetics). This equipment can also differentiate between minerals with different levels of magnetic properties and is able to separate very magnetically susceptible minerals from slightly magnetically susceptible minerals. In turn, minerals that can discharge an electrical charge easily (conductors) can be separated in special equipment from minerals that cannot discharge an electrical charge easily (nonconductors). Like the magnetic separation equipment, the electrostatic separation equipment can differentiate between minerals with different levels of electrostatic properties with different specific gravities, and grain sizes can be separated easily in special equipment designed for these purposes. Using all these separation methods, the desired minerals can be separated from each other and the waste. Figure 6.70 shows a flow sheet of the process.

The mineral sands are delivered by road train, are stored until required, and fed into the plant. For processing, the mineral sands

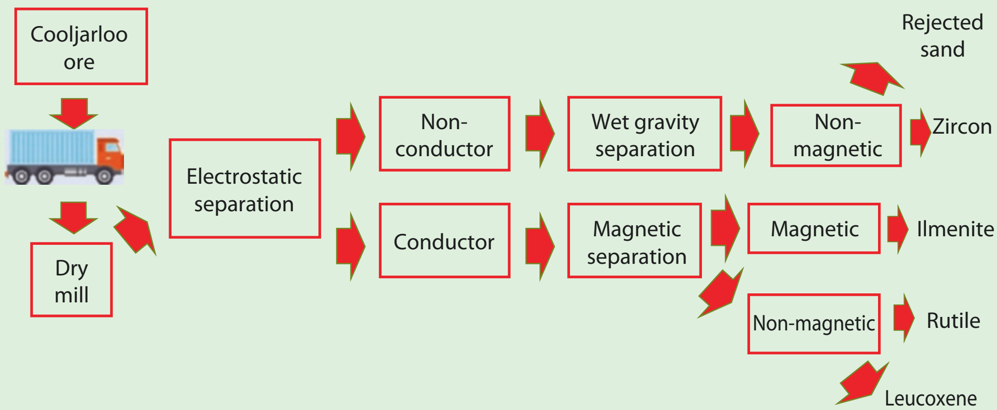


Fig. 6.70 General flow sheet of heavy mineral dry mill separation

Fig. 6.71 UltraStat separator (Image courtesy of Tronox Ltd.)

are first cleaned to remove any surface contamination from the particles. To do this, the mineral sands are formed into a slurry by mixing them with water before they are attritioned, a process of agitating the particles to make them rub together.

The cleaned mineral sands are then dried on filter belts and in dryers before undergoing electrostatic and magnetic separation. In electrostatic separation, the mineral sands are separated into two streams: conductors and nonconductors. The conductors contain the ilmenite, leucoxene, and rutile, while the nonconductors contain the zircon. The process of electrostatic separation is carried out using electrostatic plate separators (ESP) and UltraStat separators (Fig. 6.71). The former utilize a stationary grounded, sloped, or curved surface onto which a monolayer of mineral is fed. The mineral passes beneath a charged electrode which induces a polar opposite charge on the conductor particles; as a result, these charged particles are electrostatically attracted to the electrode and are drawn away from the grounded surface. A splitter located further in the separation zone separates the conductor particle and nonconductor particle trajectories dividing the feed into mainly nonconductor and conductor fractions. UltraStat separators use a very different mineral charging mechanism to ionized field separators. Thus, a strong static electric field is used to selectively induce charge onto the conductive mineral particles (conductive induction charging).

The conductor's stream is separated further into magnetics and nonmagnetics. The magnetics contain ilmenite, and the nonmagnetics contain rutile. Leucoxene is also separated at this stage, as it has magnetic properties between those of ilmenite and rutile. The oversize conductors are separated and rejected as waste as it does not contain significant amounts

Fig. 6.72 Rare earth drum magnet (Image courtesy of Tronox Ltd.)

of leucoxene or ilmenite. The magnetic separation is carried out using rare earth drum magnets (Fig. 6.72) and induced roll magnets. In the rare earth drum magnets, the drums have a high-intensity magnetic field and will affect particles of low magnetic susceptibility carrying them around the drum to discharge into the magnetic product chutes. Nonmagnetic particles follow a trajectory over both splitter blades and fall into the nonmagnetics chute. Another form of machine for separating magnetic from nonmagnetic is an induced roll magnetic separator, commonly referred to as an IRM. Magnetic and nonmagnetic particles are separated as they pass through a magnetic field between a rotating roll and a pole piece.

Using wet gravity separation, the nonconductors are divided into three streams: oversize, middlings, and undersize. They are then separated into magnetics and nonmagnetics before being sorted into different specific gravities. Shaking tables, spirals, and jigs are used for wet gravity separation. The oversize magnetics contain no significant zircon and are rejected as trash. The middlings are fed to the coarse zircon wet circuit. The undersize low magnetics is a fine nonconductor and is fed to the fine zircon wet circuit. Most of the remainder of the separation process is sorted by specific gravity-producing zircon and waste.

6.8.7 Pyrometallurgy/ Hydrometallurgy

As aforementioned, pyrometallurgy and hydrometallurgy must be briefly described in this section devoted to mineral processing because it is a method commonly used to remove metals (e.g., copper or gold) in a processing plant more or less near the mine. Thus, the location of the plant is similar to that of a typical beneficiation plant. Pyrometallurgy includes the use of heat for the treatment. It develops heating in a blast furnace at temperatures above 1500 °C to transform mineralization to a material that can be refined. Thus, the oxide is heated with a reducing agent such as carbon (e.g., coke or coal); the oxygen of the metal is combined with the carbon and is extracted as carbon dioxide gas. Hydrometallurgy, sometimes called leaching (leaching can be also considered only a stage in the process), includes dissolution of metals selectively from their waste, involving the utilization of aqueous chemicals and much lower temperatures to concentrate metal. The decision whether to apply hydrometallurgy or pyrometallurgy is commonly taken in accordance with several factors such as the environment and economy.

Conventionally, some metals are removed by a pyrometallurgical process termed smelting. For example, copper mineralization is mined, crushed, ground, concentrated, and smelted, and the resulted copper is finally refined. In the smelting operation, where heat is used to treat metal concentrates to obtain a raw material, the concentrate is fed to a smelter together with oxygen, and the copper and iron sulfides are oxidized at a high temperature resulting in impure molten metallic copper (97–99%), molten iron oxide, and gaseous sulfur dioxide. The impure copper is then purified by electrolytic purification to 99.99% pure copper, while the iron oxide is disposed of as slag. Smelting is generally followed by a refining process where the crude metal is refined into pure metal such as copper, zinc, or nickel. There are many processes in smelting depending on the metal to be recovered, and the treatment can become quite complex. Smelters are large and costly plants that often concentrate from several mines. Only very large base metal mining operations can afford to have their own smelter, and most smelters are situated away from mine sites but near transportation routes and sources of power (Stevens 2010). A typical by-product of many smelters is sulfuric acid that is produced by scrubbing and capturing of sulfur dioxide gas during the smelting process.

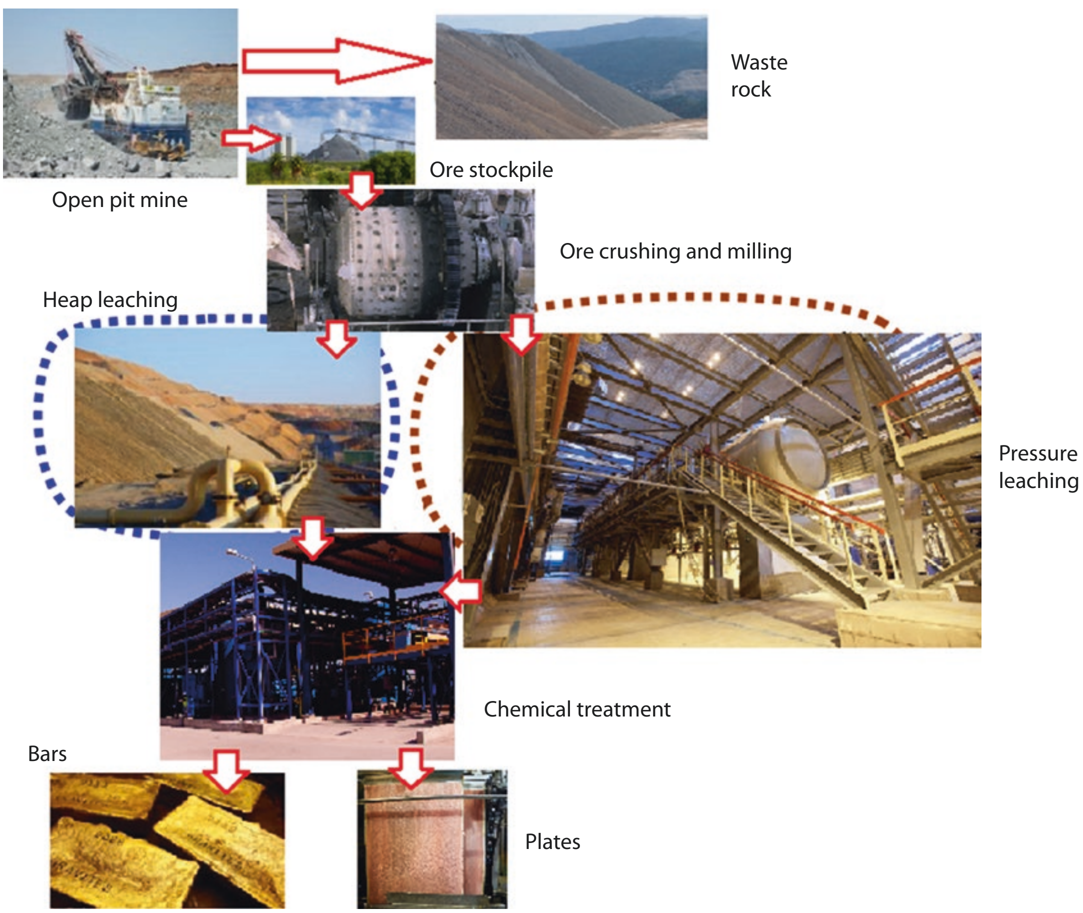
Because of environmental constraints (e.g., harmful gaseous emissions, dust production, among others) of pyrometallurgy, one of the most notable changes in the mineral processing technology has been the change from this technique to hydrometallurgy in nonferrous processing (Randolph 2011). Thus, hydrometallurgical techniques to recuperation of metals were introduced with the aim of improving environmental requests and reducing capital and operating costs (Lakshmanan et al. 2016). Hydrometallurgy is commonly separated into three main areas: leaching, solution concentration and purification, and metal recovery. ▪ Figure 6.73 shows different examples of metals obtained by pyrometallurgy/hydrometallurgy.

▪ **Fig. 6.73** Different examples of metals obtained by pyrometallurgy/hydrometallurgy. **a** Copper smelter (Image courtesy of Glencore), **b** gold pour (Image courtesy of New Mining Corporation), **c** smelting iron (Image courtesy of Rio Tinto), **d** gold doré (Image courtesy of Polymetal International plc)

Leaching

Leaching is a physicochemical process in which the minerals present in ores go through dissolution under percolating water and anion/cation exchange reactions to originate metal salts that travel and accumulate under hydrological forces. A good example of natural leaching processes in the earth is the existing laterite ore deposits. The prime features to elect correct leaching method and lixiviant chemicals are (a) composition and texture of the mineralization and (b) economic viability based on grade and reserve of the mineral deposit, calculated environmental costs, forecasted commodity market prices, and capital investment required for the project.

In hydrometallurgical processes, leaching of mineralization is carried out utilizing several procedure patterns. Some of them include leaching at atmospheric pressure or at higher pressures, while other configurations apply recovery in situ or by placing the raw material in a heap. Atmospheric leaching includes reaction vessels working at atmospheric pressure incorporating the leaching solution and ground feed material. The pressure leaching has been utilized for mineralization including metals such as uranium, zinc-lead, copper, nickel-cobalt, gold-silver, and platinum group metals (PGM). Responding to the growing prices of base metals such as copper, nickel, cobalt, and precious metals, heap leaching is converted to a major extraction method utilized for large-tonnage and low-grade mineralization that cannot be economically separated into a concentrate or operated through atmospheric or pressure leaching (Lakshmanan et al. 2016). A classical processing plant using ore leaching technologies has the general activities illustrated in ◘ Fig. 6.74. ◘ Box 6.9: Ranger Uranium Processing Plant.



◘ **Fig. 6.74** General activities of a mine using leaching technologies

Box 6.9

Ranger Uranium Processing Plant (Jabiru, Australia): Courtesy of Energy Resources of Australia Ltd.

Ranger Uranium Mine is one of the largest uranium mines in the world and is located in Australia's Northern Territory, approximately 260 km east of Darwin. Bordered by the Kakadu National Park, the mine was discovered in 1969 and began operations in 1980. The geology is dominated by Paleoproterozoic volcanic, carbonate, and sedimentary sequences that unconformably overlie the Archaean granitic gneiss. These sequences are folded, faulted, sheared, and crosscut by east-trending granite dykes and pegmatite veins. Regional metamorphism is to greenschist facies, and contact metamorphism is to hornblende-hornfels facies. Due to different theories, the absolute age of uranium mineralization at the Ranger Mine is open.

Energy Resources of Australia mines and processes uranium ore at Ranger mine. Since mining of Pit 3 ended in November 2012, Ranger Mine has been processing ore from stockpiles (Fig. 6.75). The ore comes actually from both Pit 1, which was depleted in 1994, and Pit 3. The process for extracting uranium ore from the stockpiles to producing drums of uranium oxide involves a number of complex operations (Fig. 6.76). It includes the following six steps.

Step 1: Uranium ore is crushed in a crushing (and screening) circuit to less than 19 mm. This fine ore is then mixed with water and ground in the milling circuit to less than 0.30 mm in a slurry of ore and water.

Step 2: The ore slurry is thickened and then pumped to leaching tanks. In these tanks, sulfuric acid and pyrolusite (a manganese-based oxidant) are added to the ore slurry and dissolve more than 90% of the uranium in the ore over about 24 h.

Step 3: The dissolved uranium is separated from the ore slurry in a washing circuit known as counter-current decantation (CCD). After separation, lime is added to the depleted ore slurry (tailings) to neutralize tailings acidity before being pumped to the Ranger tailings storage facility. The dissolved uranium solution is then passed through a clarifier and a set of sand filters to remove any fine solid particles.

Step 4: The uranium solution is pumped to the solvent extraction circuit to remove the many other dissolved elements carried with the uranium solution. During solvent extraction, a type of kerosene as well as an amine is added to purify and concentrate the uranium solution.

Step 5: After solvent extraction, the pure, but quite weak, uranium solution is pumped to the precipitation tanks. More ammonia is added, causing a uranium compound (ammonium diuranate – ADU) to form and precipitate from the clean uranium solution. ADU is bright yellow and commonly called «yellowcake.»

Fig. 6.75 Uranium ore stockpiles (Image courtesy of Energy Resources of Australia Ltd)

Step 6: In the final stage of the process, the yellowcake is heated to approximately 800 °C in a multi-hearth calciner (a large diesel-oil-fired, industrial oven). This «cooking» process produces the final product uranium oxide (U_3O_8), a dark green powder. The product is then packed into specially approved 200 liters steel drums that are sealed and loaded into shipping containers ready for transportation.

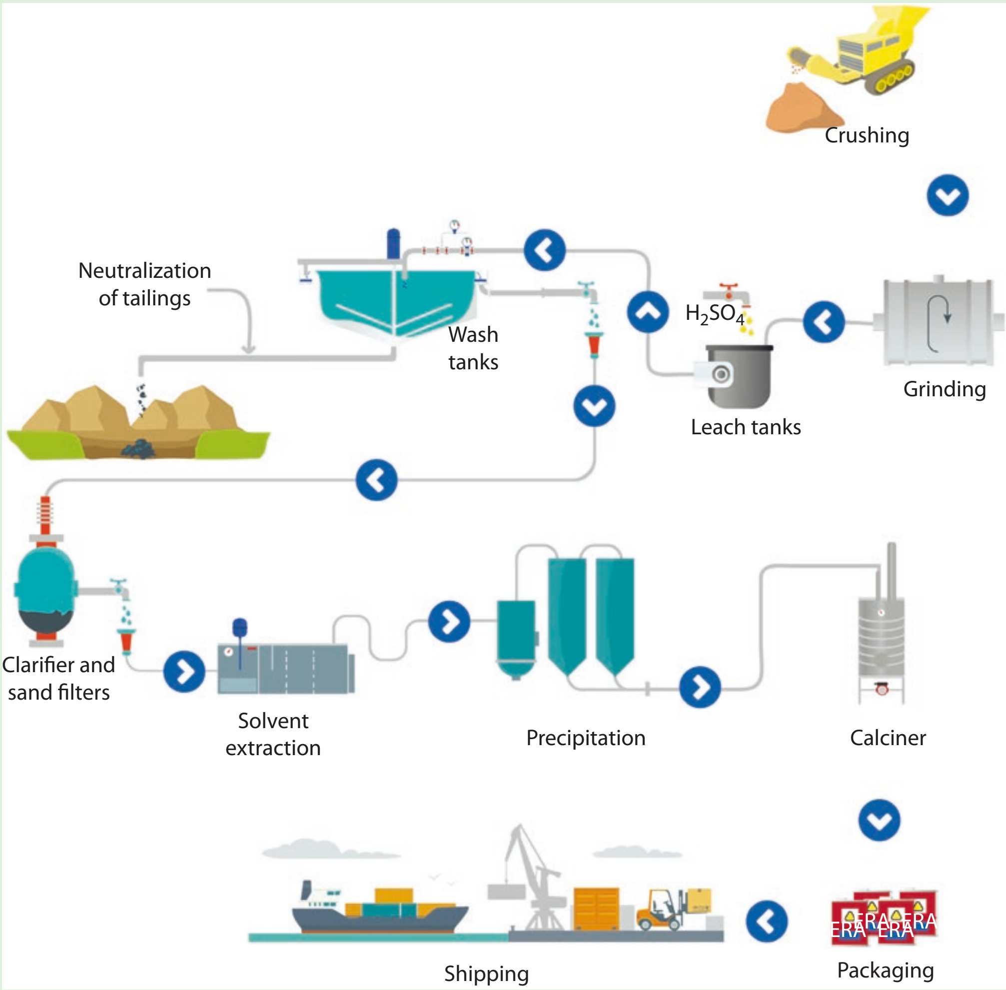


Fig. 6.76 General flow sheet of the uranium processing plant

Heap leaching is a method where materials or crushed mineralization are piled over an engineered impermeable pad and wetted with lixiviant chemical components under atmospheric status, being the leachate gathered for metal recovery procedures. Due to percolation of the lixiviant solution through the mineralization carried out under gravity and atmospheric conditions, completion of metal extraction needs longer periods (weeks to months, even several years in bioprocesses) for each pad collection sequence in comparison with tank leaching (hours to days). In some case, operation is carried out on a lined surface that is covered with stabilized surface (on/off pad) to enable extraction of the processed mineralization generally by loaders or mechanized devices. The treated mineralization is translated to a lined facility (spent ore repository) for further closure and reclamation, with the stabilized/lined leach pad area being reutilized (Zanbak 2012). This technique is a comparatively inexpensive method to dissolve some metals into a cyanide solution, being especially interesting in treating low-grade gold ores.

On the other hand, a singular heap leaching method (bio-heap leaching) is applied to determine types of sulfide copper mineralization where insoluble copper, nickel, zinc and cobalt sulfides, and uranium oxides are changed into water-soluble sulfates in a two-step leaching procedure with the assistance of natural iron-oxidizing bacteria in an acidic environment improved with sulfuric acid.

Tank leaching is a method in which crushed mineralization or flotation concentrates are chemically operated in open tanks under atmospheric pressure to recover metal salts from the mineralization at a quick rate. This method, also termed «semi-closed system,» needs grinding of all materials and disposal of processed components in tailings impoundments, or if a heap leaching installation is developed, the dewatered tailings may be submitted to the leach pad for a second leaching process. Tank leaching methods are broadly utilized in mining for the extraction of metals such as gold and silver. There are two tank leaching procedures where activated carbon is utilized for adsorption of cyanided gold: carbon in pulp (CIP) and carbon in leach (CIL). Another adsorption procedure, carbon in column (CIC), is commonly applied to gold extraction from heap leach solutions. Currently, between 60% and 65% of the world production of mined gold is processed utilizing tank leaching. In CIP process, the pulp from the grinding step is thickened up to about 55% solids and leached in a cyanide solution with the objective to dissolve the gold. The gold is then absorbed into activated carbon and sent to a new clean cyanide solution for further precipitation in steel wood. Finally, the gold recovery process is based onto melt the previous product in a furnace and generated doré bars, which can be further refined up to 99.99% purity.

Pressure leaching is a method where ground mineralization or flotation concentrates are chemically operated in reactors (autoclaves) under high pressure and temperature conditions to recover metal salts from the mineralization also in a quick rate. This method is generally termed «closed system» (Box 6.10: Albazino-Amursk Gold Processing Plant). Considering intrinsic early high investment and operational cost, this method cannot be utilized in low-grade mineralization.

Box 6.10

Albazino-Amursk Gold Processing Plant (Albazino, Russia): Courtesy of Polymetal International plc.

The Albazino deposit consists of several seemingly isolated northwest-trending mineralization zones separated by fault-bounded structural blocks. Gold mineralization at Albazino is of the low-sulfide, gold-pyrite-arsenopyrite, vein-disseminated ore type and is associated with moderately dipping dykes that crosscut sandstones. Mineralization is not confined to the dykes and may extend up to 20 m into the host sandstone wall rock. The most intense gold mineralization is associated with fold zones, averaging between 10 and 30 m thick with intense veining. The group's current life of mine plan provides for open-pit mining until 2023. It comprises operating open-pit mine and a 1.6 Mtpa on-site flotation concentrator that started up in April 2011. The gold concentrate produced on-site is trucked to the Amursk pressure oxidation (POX) hub for processing into doré bars. Figure 6.77 shows the general flow sheet of the combined Albazino-Amursk facilities.

The Albazino high-grade gold ore is refractory with the majority of gold intimately associated with arsenopyrite and pyrite in microscopic and submicroscopic form. The gold is refractory because micron gold is capsuled in sulfides (pyrites and arsenic pyrites), and the way to overcome refractoriness is to destroy sulfide matrix. As such, it is not amenable to recovery by conventional cyanidation. It is important to note that more than 30% of the world's gold resources are refractory, which implies low recoveries when using conventional processing technologies.

Albazino Concentration Plant

The company considered the recognized processes available for treating refractory ores such as autoclaving, roasting, and

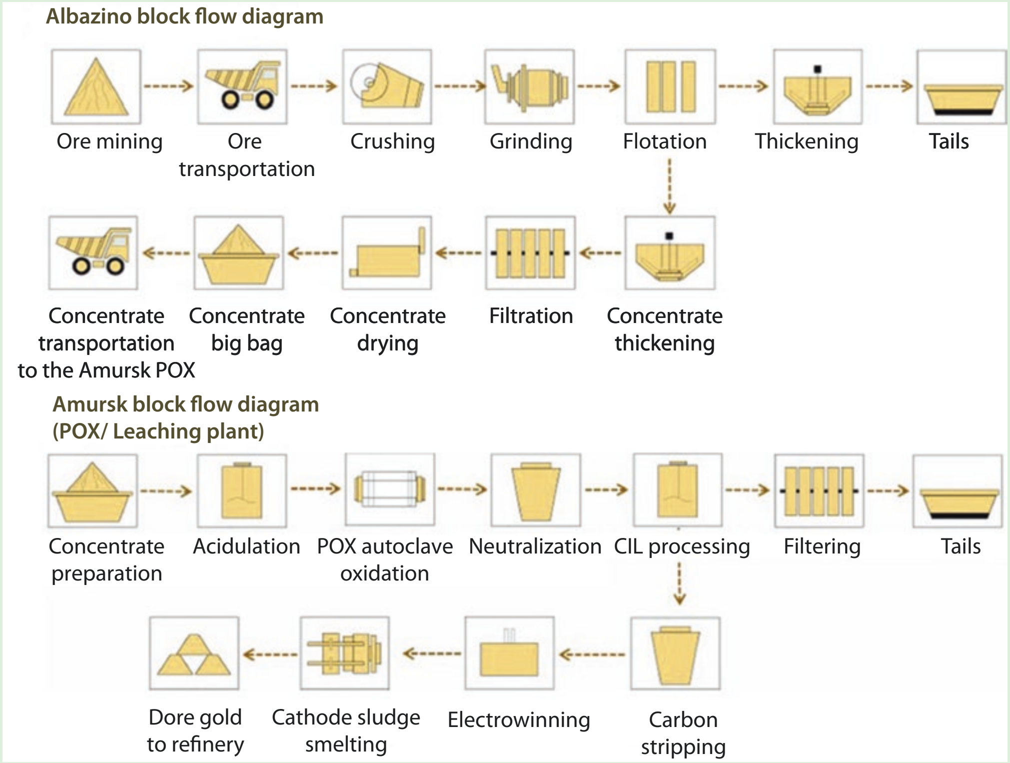


Fig. 6.77 Albazino-Amursk processing flow sheet (Illustration courtesy of Polymetal International plc.)

bio-oxidation, which would liberate the gold and render it recoverable by conventional cyanidation. It finally selected autoclaving as the preferred processing option with the preliminary flotation at the mine site to produce a flotation concentrate. As an option to processing at the company's POX plant in Amursk, the concentrate can be sold to a third-party offtaker. First concentrate sales were made to China in 2011.

Albazino on-site concentrator has a capacity of 1600 ktpa and is now operating at full design capacity. The design flotation recovery is estimated at 87.5% and was achieved in April 2012. The concentrator is located 3 km away from the Anfisa pit. The run-of-mine ore will be fed to the primary jaw crusher and reduced to 250 mm. Crushed ore is subject to three-stage milling (one SAG mill and two ball mills) and two-stage conventional flotation in two parallel circuits. The concentrate is filtered, dried, and then shipped by road and barge to the operating POX plant in Amursk. Road transportation will be year-round, but the barging operation will be restricted to a 6-month window when the river is ice-free.

Amursk Processing Plant

Located in the Russian Far East, the Amursk hub is the first gold pressure oxidation plant in Russia, processing ore from two high-grade refractory gold deposits (Albazino and Mayskoye). The Amursk hydrometallurgical plant uses POX and cyanidation technologies to process refractory concentrates that require pretreatment oxidation before conventional cyanidation to extract the sulfur component. Construction of the Amursk POX plant was completed in December 2011, and the first gold was poured in April 2012. Amursk hub design capacity is 225 Ktpa of concentrate depending on sulfur grade in concentrates. It results in approximately 400 koz of gold produced per annum with the inclusion of concentrate from the Mayskoye mine.

Regarding autoclave oxidation of refractory ores and concentrates, POX is a technological operation in which the slurry is subjected to high pressure and high temperature (22.7 bars and 200 °C, respectively) during 2 h in an autoclave of 22.3 m length (Fig. 6.78) with the goal to destroy sulfide particles enveloping gold particles and make the slurry amenable to cyanide leaching. The process is particularly well suited for treating refractory gold ores, which give low recovery rates when directly leached with cyanide. Despite higher initial capital costs and high technical expertise, POX offers several advantages over alternative processes including (a) more flexible and more stable in terms of feed variability, (b) higher recoveries due to high levels of oxidation (>98% S), (c) lower operating costs (less energy intensive and lower neutralization

Fig. 6.78 Autoclave in Amursk processing plant (Image courtesy of Polymetal International plc.)

costs), and (d) more environment friendly (dry stacking possible, lower cyanide consumption).

The POX circuit comprises the following steps:

1. Incoming concentrates are unloaded from 14 tons big bags in source-specific batches into a bin.
2. Concentrates are fed from the bin by a high-angle conveyor into a ball mill, where the material is diluted with water and stored in source-specific agitated tanks.
3. The slurry from various tanks is carefully blended in the feed tank to achieve stable sulfur grade in the autoclave feed.
4. The slurry is acidified to destroy carbonates in the feed and preheated with recirculated process water.
5. The slurry is pumped by two positive-displacement pumps into a five-compartment autoclave; oxygen is produced on-site at the oxygen plant and injected in the autoclave to achieve at least 98% sulfur oxidation; high-temperature steam from a special boiler is injected to initiate the chemical reaction during start-ups, and freshwater is injected to control the temperature.
6. Oxidized slurry is discharged through a flash vessel where both temperature and pressure drop; off-gas from the autoclave is scrubbed from sulfur oxides.
7. Autoclave discharge is neutralized by the addition of limestone, and the slurry's pH is further increased by the addition of lime; limestone and lime are crushed, milled, and diluted with water in separate two-stage crushing and milling sections.
8. pH-adjusted slurry is sent to the carbon-in-leach (CIL) circuit where it undergoes carbon desorption, carbon regeneration, electrowinning, and doré smelting; gold is absorbed onto activated carbon in parallel with leaching.
9. Chemically inert tailings are filtered and dry-stacked in a fully lined tailings storage facility.
10. Filtrate water is washed in a clarifier and sent to a reverse osmosis facility where deleterious elements are removed with clean water recirculated to the process.

Solvent Extraction/Electrowinning

Beginning in the mid-1980s, a new technology, generally termed the leach-solvent extraction-electrowinning process (SX/EW), was broadly applied. The idea of selectively recovering copper from a low-grade solution and further stripping the copper into an acid solution from which electrowon copper cathodes could be generated was developed by the Minerals Group of General Mills in the early 1960s (Kordosky 2002). This simple concept has converted into a method by which refined copper production from SX/EW covered nearly 17% of the global copper-refined production (ICSG 2015). The SX/EW procedure is really a hydrometallurgical process because it is performed at atmospheric temperatures, being the copper in an aqueous or organic environment during its treatment until it is reduced to the metal. The SX/EW process has very little environmental issues since its liquid streams are very easily entrapped. Other advantages of the procedure is its low capital investment need relative to the smelting process and its capacity to operate economically in a small scale.

In summary, SX/EW procedure includes leaching the material in a weakly acid medium. This solution, called «pregnant leach solution» (PLS), is extracted and then contacted with an organic solvent, known as the «extractant,» in the solvent recovery stage (SX). Thus, copper is recovered from the aqueous phase. Because the copper ion is exchanged for hydrogen ion, the aqueous phase is reverted to its original acidity and recycled to the leaching stage of the procedure. In the meantime, the copper-bearing organic phase is deprived of its copper by contacting it with a strongly acidified aqueous solution at which time the copper is translated to the aqueous phase, while the organic phase is reconstructed in its hydrogen form. Industrial electrowinning requires pure copper-rich electrolytes, and SX process provides the means for generating this class of electrolyte from dilute, impure leach liquours; it is a crucial stage in SX/EW process. In the strip step, copper is deprived from the loaded organic solvent into the advance electrolyte from which the copper is electrowon, being the stripper organic solvent recycled to the extraction step (Schlesinger et al. 2011).

In copper electrowinning, Cu^{2+} in the purified advance electrolyte from SX is reduced to copper metal at the cathode by the use of a DC electrical current. Sulfuric acid generated at the anode of the electrowinning cell is restored to the SX system in the copper-depleted spent electrolyte to strip more copper from the loaded organic solvent. The copper in solution electrically plates over several hours to a day until the copper is approximately 12 mm thick. These copper plates are termed «copper cathodes» and generally are formed by 99.99% copper. They are usually named as copper grade «A» brand in London Metal Exchange (Box 6.11: Cobre Las Cruces Copper Processing Plant).

Box 6.11

Cobre Las Cruces Copper Processing Plant (Sevilla, Spain): Courtesy of Cobre Las Cruces – First Quantum Minerals Ltd.

The Cobre Las Cruces deposit occurs in the eastern end of the Iberian Pyrite Belt, a 300 km long and 80 km wide geologic belt that extends eastward from southern Portugal into southern Spain. Mineralization consists of syngenetic massive sulfides containing polymetallic mineralization, similar to most other Iberian Pyrite Belt deposits. The massive sulfide mineralization on the property is hosted by late Devonian to early Carboniferous Period volcanic and sedimentary rocks deposited in a submarine setting within a narrow and relatively shallow intracontinental sea and characterized by bimodal volcanism and sedimentation. Post depositional secondary copper enrichment occurred in the upper part of the massive sulfide deposit, forming the secondary mineralization of interest. The deposit was subsequently buried under 100–150 m of sandstone and calcareous mudstone. Thus, Cobre Las Cruces is a blind deposit with no outcrop because of the 100–150 m of sedimentary rocks overlying the deposit. The copper in the ore is primarily found in chalcocite with some minor amounts found in chalcopyrite, tennantite-tetrahedrite complex, and enargite. The ore from the open-pit mine ranges in grade from 5% to 10% copper, and the design grade is 6.02% Cu.

The metallurgical plant (Fig. 6.79) relies on an atmospheric leaching process to recover copper from the rich secondary sulfide chalcocite mineralization. A unique feature of the plant is the use of eight agitated

Fig. 6.79 Cobre Las Cruces processing plant (Image courtesy of Cobre Las Cruces – First Quantum Minerals Ltd.)

reactor tanks to dissolve the copper under conditions of high temperature and high acidity. Oxygen is also added into the reactors to complete the reaction. The feed to the leaching reactor tanks is mined ore that has passed through three stages of crushing, a single stage of grinding and has then been thickened to eliminate as much process water as possible and three mixing tanks where reactions start (pre-reactors). The ore processing facility is designed to operate 365 days per year, 24 h per day, and 1.5 Mtpa of ore to produce 72,000 tpa of cathode copper. A simplified process flow diagram for the plant is presented in Fig. 6.80.

The Cobre Las Cruces plant uses a hydrometallurgical plant to recover cathode copper using a process consisting of conventional crushing and grinding, agitated ferric sulfate leach in atmospheric leach reactors, and recovery of copper with conventional SX/EW technology. The copper produced is ready for sale directly from the plant. The final product is 99.999% pure copper classed as grade «A.» This modern process technology for hydrometallurgical copper production has several benefits: it offers high yields, low operating costs, and flexible operation. This copper is also called «Five Nines» copper, which refers to the highest quality of cathodes produced by the company, one of the very few copper producers in the world that certifies its cathodes as 99.999% pure, compared to the usual 99.99%. The London Metal Exchange establishes that «four nines» is a requirement to obtain grade «A» quality, the highest in existence.

Crushing and Grinding

Crushing and grinding of the ore brings it down to a fine pulp. Primary crushing is carried out by a jaw crusher and then onto a second crushing circuit with cone crushers. As a result, the ore is <15 mm in size. The ore is then moved onto the grinding circuit where a wet ball mill, loaded with 80 mm balls, further reduces the ore to a fine pulp. Hydrocyclones control the oversize material in a closed circuit. The output ore of the ball mill is <150 µm in size with a grinding P_{80} of 75 µm. Classification and thickening provide a final product at desired size and minimum water.

Leaching

Ferric-leach or SX/EW technology is common at copper heap leach operations that treat chalcocite-rich ores. However, the agitated atmospheric ferric-leach technology that is used for treating the Las Cruces ore is a relatively new technique in the copper industry. The reactors or leaching tanks, an essential part of processing the ore, enable the copper to be extracted using a

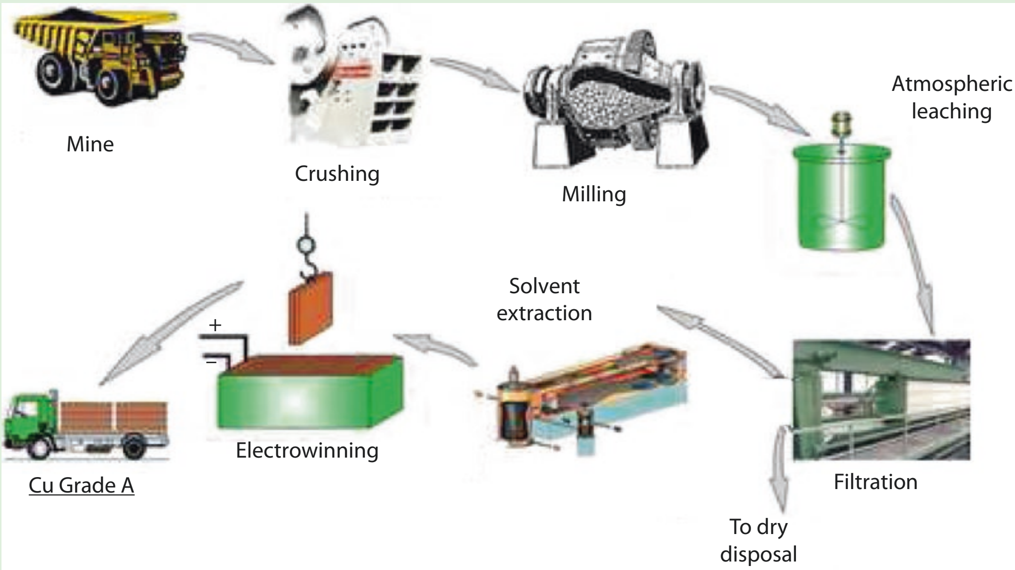


Fig. 6.80 General flow sheet of copper processing plant

hydrometallurgical process that is cleaner than conventional pyrometallurgical processing. Thus, the pulp obtained in the grinding process is leached in an acid aqueous solution at 90 °C in the presence of oxygen and ferric iron to dissolve more than 90% of the copper minerals in the ore. The 350 m^3 atmospheric leachers (agitated tanks) convert the pulp to a concentrate. The distribution of mixing energy favors reaction kinetics. The process provides high recovery rates, high oxygen utilization, and full suspended material at low mixing energy levels.

Solven Extraction and Electrowinning

Once leached, the liquid is separated from the ground solids to become a pregnant leach solution, the feed for the solvent extraction (SX) area. Thus, the concentrate is then ready for SX/EW processing. In the copper process, it is necessary to remove the gypsum along with other residual ultrafine gangue material prior to passing the pregnant liquor through to solvent extraction. This is because the rapid formation of gypsum scales within the filters means that the filters failed within a very short space of time. For this reason, following effective coagulation and flocculation, a set of clarifiers effect efficient separation so that the total residual suspended solids going through to subsequent processing are measured consistently at less than 50 mg/l. Thus, the objective is to treat the overflow from the gypsum thickener supplying acidified pregnant liquor to the solvent extraction process.
Once the concentrate is dissolved in the aqueous solution, the resulting electrolyte is introduced to the cell tank house that holds 138 operating cells and 22 scavenger cells, and copper is deposited over a 7-day growth cycle. This forms plates (Fig. 6.81) of copper metal (cathode copper), which is the final product, ready for sale and further processing into copper rod or wire. The electrowinning section is equipped with an acid mist capturing system in order to meet the strict environmental demands. The electrowinning cells produce London Metal Exchange grade «A» copper cathodes weighing approximately 50 kg each. An automated crane and stripping machine then harvests and packages the cathodes for shipment. Leach residues are filtered, stacked, and encapsulated in a mine waste dump. The disposal design exceeds regulatory requirements. A process bleed stream will be treated to remove heavy metals and then discharged. The effluent quality meets discharge standards as evidenced by sampling of the discharge.

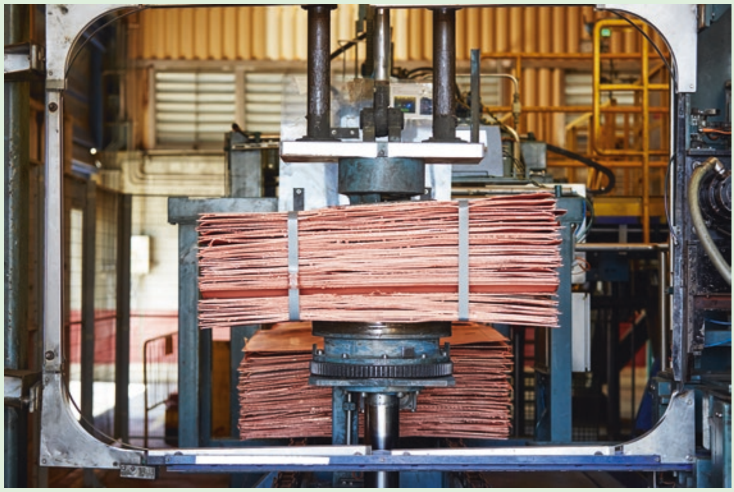


Fig. 6.81 Plates of copper (Image courtesy of Cobre Las Cruces – First Quantum Minerals Ltd.)

6

6.9 Dewatering

Most mineral separation processes, commonly between 80% and 90%, include the utilization of significant amounts of water. Thus, the final concentrate has to be concentrated from a pulp in which the water/solids ratio can be high. Dewatering, sometimes also termed solid-liquid separation, generates a fairly dry concentrate for marketing. The objective of dewatering, an essential process in mineral processing, is to recover water absorbed by the particles, which increments the pulp density. The reasons to carry out dewatering are numerous such as to allow mineralization handling and concentrates to be moved easily, to enable supplementary processing to occur, and to dispose off the waste. Partial dewatering is also carried out at different steps in the operation, so as to prepare the feed for further procedures. The water removed from the material by dewatering is later recycled after being sent to a water treatment plant.

Dewatering methods can be broadly classified into three groups (Fig. 6.82): (1) sedimentation, (2) filtration, and (3) thermal drying. These methods clearly increment in difficulty and cost as the particle size reduces. Thus, liquid-solid separation by mechanical means (e.g., sedimentation or filtration) is much less costly than thermal drying, primarily because those methods consume less energy per unit of water removed. Accordingly, liquid-solid separation is a major cost in mineral processing, probably exceeded only by the cost of comminution and flotation (Dahlstrom 2003).

Dewatering in mineral processing is commonly a mixing of the three techniques cited above. Since the initial product after beneficiation contains about 75% water by weight, the bulk of this water is early extracted by sedimentation or thickening, which generates a thickened pulp of perhaps 55–65% solids by weight. Up to 80% of the water can be removed at this step. Filtration of the thick pulp then provides a moist filter cake of between 80% and 90% solids by weight, which finally can need thermal drying to originate a final product of about 95% solids by weight. In general, the water content of the concentrate where it is shipped (approximately 5–7% by weight) is a commitment between the cost of moving water and the requirement to diminish concentrate losses as dust during the transport (Schlesinger et al. 2011). It means that partial presence of water is desirable for easy handling and safe transport. It is important to bear in mind that smelters are commonly placed at great distances from mineral processing plants caused by inadequate infrastructure. For this reason, translating of concentrate in pulp manner to great distances is not recommended.

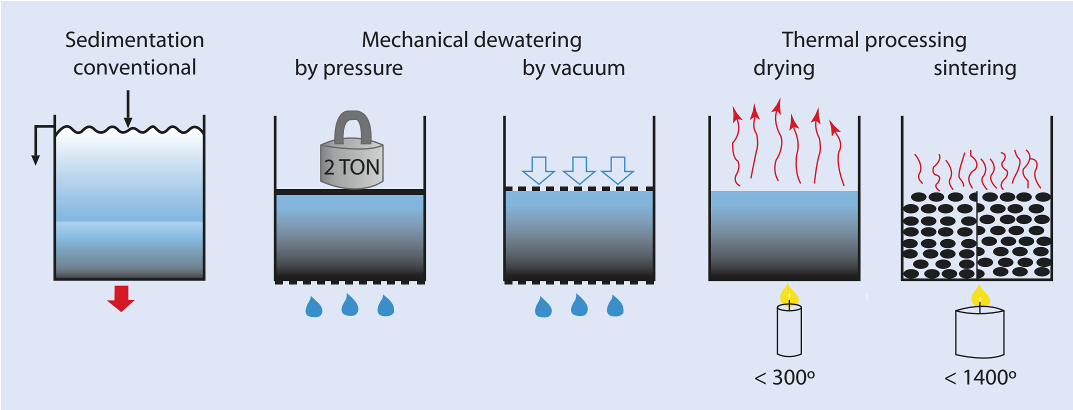


Fig. 6.82 Dewatering methods (Illustration courtesy of Metso)

6.9.1 Sedimentation

Gravity sedimentation or thickening is the most broadly applied dewatering method in mineral processing. It is a moderately non-expensive, high-capacity procedure, which includes very low shear forces thus offering good characteristics for flocculation of fine components. Quick settling of solid particles in a liquid generates a clarified liquid that can be decanted, leaving a thickened slurry that need additional dewatering using filtration. Gravity sedimentation is most effective where there is a large density difference between liquid and solid. This is constantly the situation in mineral processing where the carrier liquid is water. Very fine particles (only some microns diameter) decant very slowly using gravity process exclusively, and centrifugal sedimentation can be carried out.

Alternately, the components can be coagulated and/or flocculated into moderately large lumps that decant more quickly; coagulation occurs where two or more fine particles collide and remain in contact, whereas flocculation involves the attachment of particles to long-chain polymers (Woollacot and Eric 1994). It should not be forgotten that thickener applications usually need particles finer than 0.1 µm, and these particles are hard to decant due to electrostatic charge causing them to repel each other and hinder gravity settling. Regarding the flocculants, the new presence of synthetic polymers is a huge progress in physical concentration during the last five decades, and they have extensively replaced the organic polymers (Ballentine et al. 2011). For flocculant utilization, a handling system is needed, which includes provision to mix, store, and dilute the polymer. The dilute polymer is later combined with the feed slurry and enabled to condition previous dewatering procedure.

Thickeners (Fig. 6.83) are formed by moderately shallow tanks from which the clear liquid is removed from the top and the thickened suspension at the bottom. Thus, the thickener is utilized to increment the concentration of the suspension by sedimentation, accompanied by the origination of a clear liquid. Obviously, the solid concentration in a thickener changes from that of the clear overflow to that of the thickened underflow being extracted. Because the procedure is constant, there are no areas of steady composition in the thickener. As a consequence, several zones are present within the thickener than correspond broadly to the diverse steps of settlement (Fig. 6.84).

The design of a continuous thickener includes a large, shallow cylindrical tank with a conical bottom sloping toward the center, the diameter ranging from about 2 to 200 m, and of depth 1–7 m. Thickeners of various design are available, including differences in geometry and in the way in which the feed slurry is introduced to the thickener. Because capital investment is the principal cost of thickening, election of the appropriate size of thickener for a certain appliance is essential. The two early roles of the thickener are

Fig. 6.83 Thickener (Image courtesy of Anglo American plc)

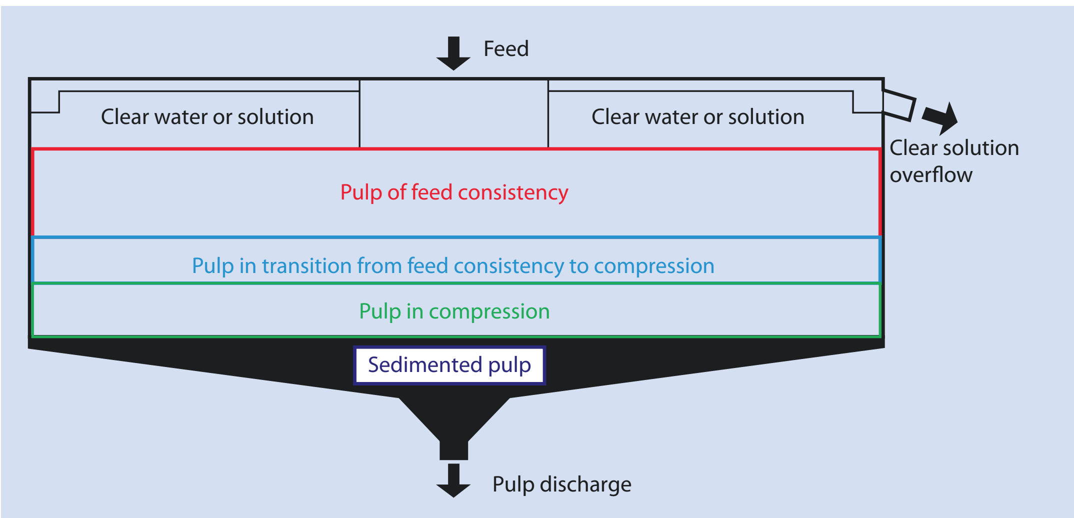


Fig. 6.84 Thickener zones

the generation of a clarified overflow and a thickened underflow of the needed concentration. For a given throughput, the clarifying capability is established by the thickener diameter. This is because the surface area must be large enough so that the upward speed of liquid is always lower than the settling speed of the slowest-settling component which is to be removed. The degree of thickening produced is controlled by the residence time of the particles and hence by the thickener depth.

Conventional thickeners have the disadvantage that large floor areas are needed since the throughput is based only in the size of the area, while depth of the tank is less important. In the last years, devices known as high-capacity thickeners have been incorporated in the market by various manufacturers. The key to high-capacity production in thickeners has been the development in flocculants technology, which enabled incremented tonnage in a smaller footprint. Thus, high-capacity thickeners are actually the norm in minerals industry. Other innovations incorporate diminishing surface area in high-scale utilizations such as alumina refining and including spikes on blades to resuspend heavily thickened components like magnetite (Gorain 2016).

6.9.2 Filtration

Filtration is the procedure of concentrating solids from liquid utilizing a porous medium (the filter) that entraps the solid but enables the liquid to pass. Filtration in mineral processing applications commonly follows thickening by means of a porous medium; the thickened pulp can be sent to storing agitators from where the pulp is drawn off at uniform rate to the filters. For particles coarse enough that capillary pressures are negligible (e.g., natural sand and gravel for aggregates), gravimetric dewatering can be employed using devices such as spiral dewaterer, dewatering screen, or dewatering wheel (◘ Fig. 6.85). This is not usually the case in mineral processing concentrates, and cake filters are the type most frequently used. The requirements under which filtration is performed are numerous, and the selection of the most adequate machine will be based on a large number of features. Whatever device is utilized, a filter cake progressively builds up on the medium, and the resistance to flow gradually increments throughout the process.

In general, there are five steps in the cake filtration process: cake formation, moisture reduction, cake washing, cake discharge, and medium washing. The filtering procedure is finished where almost all the liquid has been removed from the pulp and the filter cake is extracted from the filtering medium. Prior to extracting the cake, it can be washed to remove the attached fluid, that is, the fluid maintained in the pore spaces in the cake and any solute in the feed that is entrapped within the cake. Regarding the factors affecting filtration, a variety of options are available in order to maximize the capacity of a given filtration device. The more important are (a) increased filtration pressure, (b) increased pore size, (c) manipulation of the cake formation, (d) use of flocculants or filter aids, and (e) washing of the filter cloth.

The design of the supporting base of the filtering medium is a guidance to the names of filters in the market. Thus, if the filtering medium is between grooved plates, the filter press is called plate filter; when it is in the form of disk, the filter press is termed disk filter; and when in the form of a drum, it is known as drum filter (Gupta and Yan 2006). On the other hand, the technique of pressure appliance also plays a part in the nomenclature. Thus, cake filters can be pressure or vacuum types. In pressure filters (e.g., plate filter), positive pressure is applied at the feed end. In vacuum filters (e.g., disk and drum filter), there is a vacuum at the far side of the filter, the feed side being at atmospheric pressure.

◘ **Fig. 6.85** Dewatering wheel (Image courtesy of CEMEX)

The Filter Medium

A filter medium (Fig. 6.7) can be defined as any material that, under the operating features of the filter, is permeable to one or more components of a mixture, solution, or suspension and is impermeable to the remaining components (Wakeman and Tarleton 2005). The main role of the filter medium is therefore to produce a constant separation of particles from the fluid but always including the lowest energy consumption. Thus, correct choice of the filter medium is an essential part of efficient filter operation, both in terms of operating cost and filtrate clarity (Cox and Traczyk 2002). Although the early objective of the medium is to maintain solids, other features such as to have a low resistance to filtrate flow, to resist chemical attack, or to enable efficient discharge of the cake are also significant (Kelly and Spottiswood 1982). Relatively coarse materials are commonly utilized, and clear filtrate is not produced until the initial layers of cake are constituted, the early cloudy filtrate being recycled (Wills and Finch 2016).

Filter media are manufactured from different materials such as cotton, wool, silk, glass fiber, paper, metals, etc. Cotton designs are among the most usual type of medium because of their low initial price and accessibility in a broad range of weaves. However, the trend of incrementing utilization of synthetic fibers since the 1940s is keeping, with components such as nylon, polyester, polypropylene, polytetrafluoroethylene (PTFE), and polyether ether ketone (PEEK). Advances in weaving and finishing during the 1990s have enabled enhanced filtration process of finer particles utilizing the synthetic fibers. They offer significant advantages over the natural fibers. This is because they have the best wear features, enhanced strengths, and greater stability along with enhanced filter cake liberation and decreased blinding depending on the fibers utilized (Gorain 2016). Synthetic fibers also generate longer lifetime, which leads to decreased downtime and lower operating costs. The quality of finishing substantially enhances the filter media's efficiency (Hoijer and Grimm 2011). In summary, media technology has progressed to offer a broad scope of materials and construction to provide the necessary cycle life and performance to ensure economic operation.

Pressure Filters

Filtration under pressure has certain advantages over vacuum because of the increasing fineness of mineral concentrates (those of Cu, Pb, and Zn are commonly 80% <30 μm) coupled with shipping schedules calling for moisture contents 8–10% by weight on these fine concentrates. The fact that many operations are at high altitude is also an additional drawback for vacuum units (Wills and Finch 2016). The common pressure filters in mineral processing applications come in two basic forms, horizontal and vertical, defined by the orientation of the filter plates (Concha 2014).

For example, in vertical plate filter (Fig. 6.65), the slurry is situated between two plates captured together by an exteriorly driven screw system or hydraulic ram. The filtering medium is located against the sides of the plates, and the slurry is pumped between them. The slurry pressure presses the pulp against the medium obliging the liquid to pass across the cloth and leaving the solids as a cake on both surfaces of the frame. The plates are usually squared shaped, although circular plates are also forthcoming in the market. The plate sizes range from 450 mm × 450 mm to 2000 mm × 2000 mm and frames from 10 to 200 mm in thickness (Gupta and Yan 2006). The common number of plates in the industry ranges between 25 and 50, but up to 100 plate filters are sometimes utilized.

Vacuum Filters

The most broadly utilized type of filtration in mineral processing of ores is vacuum filtration. The principle in this type of filter process is to generate an efficiently differential pressure through the filter cloth and cake by the application of a vacuum. There are three different types of devices usually utilized: drums, disks, and horizontal filters. They commonly provide the most economical continuous operation. All vacuum filters work in a similar manner. Although the design can be very varied, vacuum filters are characterized by a filtration surface that moves by different means from a point of slurry deposition under vacuum to a point of filter cake extraction. The liquid (filtrate) exits the filter using an internal piping and the vacuum head. These filters operate with an endless groups of batch cases that approach a continuous pattern. The manner in which the

Fig. 6.86 Vacuum drum filter working in a processing plant

different steps of filtration is carried out distinguishes the three major groups of vacuum filters.

Drum Filter

Vacuum drum filter (Fig. 6.86) has an extended range of application and is considered to have lower maintenance costs compared to disk unit. Drum filters are preferred in applications that require lower moisture and/or where effective cake washing is required. Rotating drum continuous filter is formed by a horizontal drum with its bottom one-third section submerged in a tank of slurry that needs to be filtered; the slurry is fed and maintained in suspension by agitators. The outside of the drum is covered with shallow compartments about 25 mm depth, each of which is covered with a drainage grid and being the compartments designed in rows. The interior of each compartment is in turn connected by a pipe to a valve mechanism on the central drum shaft, which enables vacuum or pressure to be applied to the compartment at different steps of the cycle. Thus, as the drum rotates, each row of compartments is translated progressively across the different steps of filtration. On further rotation, the drum achieves the final area where the cake is blown out using a reverse air pressure. The cake finally is discharged with a scraper against the cake along the width of the drum and is transported away (Fig. 6.86). In general, a number of smaller vacuum drum filters in parallel are preferred to a single filter to provide the best operation.

Fig. 6.87 Vacuum disk filter

Disk Filter

Vacuum disk filter (Fig. 6.87) is especially adequate for simple dewatering applications where high capacity is the main request. Thus, it has a very important role as the cheapest filtration option for slurries that are easy to filter. The disk design permits a greater filtration zone per unit

■ Fig. 6.88 Horizontal belt filter and discharge area (Image courtesy of Compositech Filters)

of floor space compared to other types of designs. Another advantage is the low initial investment cost, whereas disadvantages include that the washing of the filter medium for cloth-type systems is difficult and effective cake washing is not possible (Cox and Traczyk 2002). Another disadvantage is that both the cake formation and the cake discharge are inferior to those in a drum filter. The disk filter is similar in concept to the drum filter, but disks are used instead of a drum in order to increase the surface area to filtration. However, the filtration process follows the same sequence as described from a drum filter.

Horizontal Belt Filter

Horizontal belt filter (■ Fig. 6.88) is generally used for handling coarse solids and/or where high washing efficiency is needed. They also can achieve lower cake moistures compared to vacuum disk or drum types. Belt filter equipment size ranges from 1 m^2 up to as large as 154 m^2 (Cox and Traczyk 2002). In this device, the filter cloth is formed into a continuous belt that is passed over an arrangement of pulleys. Below this drainage belt, a series of suction boxes is installed along the length of the filter. The slurry to be filtered flows into a feed box at one end of the belt. Cake formation begins immediately, and as the belt moves toward the discharge end, it draws the formed cake with it. To discharge the cake, the belt is passed over a small-diameter roll.

6.9.3 Thermal Dewatering (Drying)

The level of dewatering that can be achieved through the methods considered previously is limited. If further upgrading is required, it must be attained through thermal processing. These methods are comparatively costly because the solid must be not only heated but also the water needs to be evaporated so that it can be carried away in a gas stream. Thermal processing is commonly classified in accordance with operating temperature: (a) thermal low (100–200 °C): utilized for drying-evaporating of liquids from solids; devices incorporate direct or indirect heat rotary dryers, steam tube dryers, and fluid bed dryers. (b) thermal medium (850–950 °C): applied for different calcining, clay swelling, limestone burning, and foundry sand burn out; equipment includes direct and indirect heat rotary kilns (■ Fig. 6.89), vertical kilns, and fluid bed calciners. (c) thermal high (1300–1400 °C): used for various calcining operations and obtained mainly with direct heat rotary kiln (Metso 2015). In mineral processing, the equipment of thermal low group is the most commonly used. For thermal processing systems, fuel consumption is one of the most important operational costs. Therefore, it is essential that procedures are engineered in a manner that optimizes the fuel effectiveness and the overall heat balance.

■ **Fig. 6.89** Rotary kiln form thermal dewatering (Image courtesy of SAMCA)

6.10 Waste/Tailings Disposal

Mining and mineral processing generate large volumes of mineral waste. At large mines, the mass of mineral waste generated can commonly be measured in the tens of millions to billions of tons. Similarly, the surface area that must be disturbed for mineral waste disposal is often measured in the tens to thousands of hectares and can account for the majority of disturbance (Borden 2011). The mineral waste generally includes waste rock and tailings. Because waste rock is unmineralized or weakly mineralized rock and unconsolidated sediments that must be removed to access the underlying ore, it is typically a poorly sorted mixture of clay, silt, sand, gravel, and boulder-size material. The most common disposal method for these materials is placement within dumps (■ Fig. 6.90) and stockpiles, although in-pit disposal is frequent in strip mines. Consequently, the disposal of tailings from mineral processing operations is a major issue in the mining cycle, although the problems involved in the disposal can be very varied according to the features of the material to be placed in the dump.

Tailings are the fine-grained waste than remains after the minerals or elements of economic interest have been removed from the ore. They are composed of the gangue minerals in the ore and residual minerals of economic interest that were not recovered along with process water and any reagents that were added during the milling and beneficiation processes. If the mineral waste is also chemically reactive, the potential environmental impacts and the complexity of waste management increase dramatically (see ▶ Chap. 7). Moreover, this environmental issue is becoming more important since the increasing exploration for metals and the working of lower-grade deposits. It is clear that the problem is serious because the quantity of tailings needing storage can commonly exceed the total amount of the mineralization being extracted and processed. Over the last century, the amounts of tailings being generated have increased dramatically due mainly to the increment in demand for raw materials and the treatment of mineralization with very low grades. Moreover, the more extensive grinding required to liberate valuable elements in this low-grade mineral deposits, fine grained ores results in a tailings material that is less satisfactory for both backfilling and tailings dam construction. In the 1960s tens of thousands of tons of tailings were generated each day, and by 2000 this approximation had incremented to hundreds of thousands (Jakubick et al. 2003).

In addition to the visual effect on the landscape of tailings dumps, the main ecological effect is commonly water contamination. It takes place from the discharge of water polluted with solids, heavy metals, mill reagents, and sulfur compounds, among

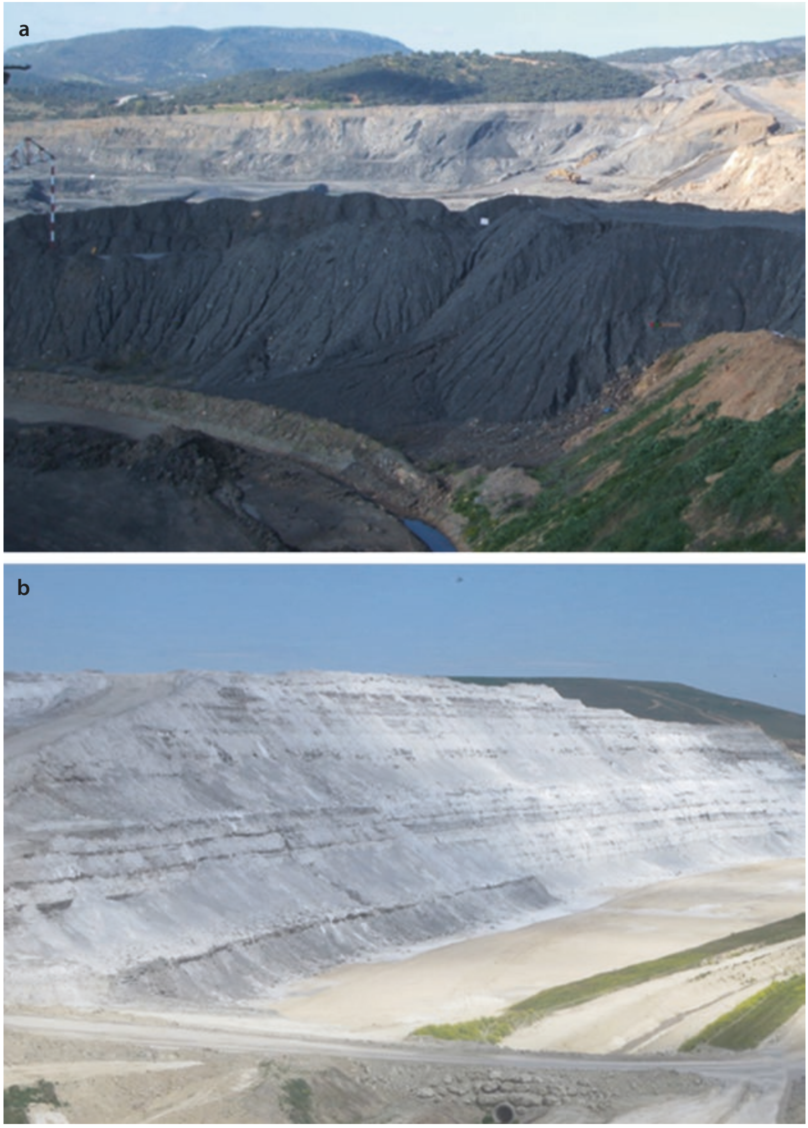


Fig. 6.90 Waste rock dumps. **a** Coal (*black*), **b** salt (*white*)

others. The waste must therefore be place in both an environmentally suitable and, if possible, economically feasible manner. In many countries, disposal is governed by the legislation and can commonly include rehabilitation of the site for a long time. For instance, the European Commission establishes the following features of the tailings to help determine the design requirements of a tailings storage facility: (a) chemical composition, including the change of chemistry through mineral processing and weathering; (b) leaching behavior; (c) physical composition and stability (static and seismic loading); (d) behavior under pressure and consolidation rates; (e) erosion stability (wind and water); (f) settling, drying time, and densification behavior after deposition; and (g) hardpan behavior (e.g., crust formation on top of the tailings). Fortunately, significant advances in mineral waste characterization and management have been made over the past several decades. Proactive mineral waste management can significantly reduce the intensity, footprint, and duration of environmental impacts.

The nature of tailings varies widely and can be formed by very coarse dry material such as the rejected components from dense medium installations. They are generally translated and disposed of as slurries of high-water content. Tailings features can change greatly, depending on the composition of the mineralization together with the physical and chemical procedures utilized to recover the valuable product. Tailings of the same type can commonly include different mineralogy and therefore will have diverse physical and chemical features (Ritcey 1989). In this sense, it is crucial to determine tailings features for establishing the behavior of the tailings after deposition in their final storage location and the potential short- and long-term liabilities and environmental issues.

Tailings are usually disposed of in specially engineered repositories capable of containing the fine-grained and often saturated tailings mass without the risk of geotechnical failure. It is important to note that tailings are waste products without economic interest to a mineral operator at that particular point in time. Thus, it is usually stored in the most non-expensive manner possible to meet regulations and site-specific characteristics. Historically, tailings were often disposed in flowing water or directly into drainages, being common this practice in some countries until recent times. Tailings and waste rock repositories change vastly in size, from tailings ponds with a size of a swimming pool to places of over 1000 hectares, and from small tailings or waste rock piles to waste rock zones of several hundred hectares or tailings heaps over 200 m high. Since the disposal of tailings is one of the greatest environmental problems of mining, it is the only item covered in this section devoted to mining waste disposal.

The ultimate objective of tailings disposal is to include fine-grained tailings, usually with a secondary purpose of conserving water for further utilization in the mine and mill. Liberation of water from the tailings once discharged in a facility and the amount available for return pumping to the processing installation are a crucial operation parameter that influences the water balance of the mining project. The process of tailings disposal has to be carried out in a cost-effective manner that provides for long-term stability of the embankment structure and the impounded tailings and the long-term protection of the environment. According to the Environmental Protection Agency of USA (EPA 1994): «In the process of designing any tailings disposal, three interests, cost, stability, and environmental performance, must be balanced, with situation-specific conditions establishing the balance at each stage of the process; it is worth noting that the long-term costs of tailings disposal depend in part on mechanical stability and environmental integrity, such that stable and environmentally acceptable structures promote cost effectiveness».

The management of tailings is of severe importance to the achievement of any mining project because many failures of tailings facilities have originated loss of life, devastating environmental issues, closure of mining operations, dramatic decreases in share value, and in some countries, the personal liability of the mine management. The first step in the development of tailings facility is to determine all regulatory requirements and laws that are applicable. In cases in which there are no such regulations or the regulations that exist are not enforced, the guidelines and standards of the lending agencies that finance the project are often applied (Equator Principles – see ► Chap. 7). Notwithstanding the existence of the lack of regulations in a given jurisdiction, responsible mining companies apply appropriate measures to the international standards. Some developed countries have laws that require companies to apply as a minimum the standards and regulations of the home country when mining in other countries (Brown 2002).

6.10.1 Methods of Tailings Disposal

The methods utilized to the location of tailings have improved due to environmental pressures, changing milling operations, and realization of profitable applications. Due to the damage caused by early disposal methods in rivers and streams and the much finer grinding necessary on actual mineralization, other methods have been developed. In general, there are the following options for managing tailings and waste rock: (a) discarding slurried tailings into ponds (◘ Fig. 6.91), (b) backfilling waste rock into underground mines or open-pits or utilizing them for the construction of tailings dams, (c) dumping nearly dry tailings or waste rock onto heaps or hillsides, (d) utilizing the tailings and waste rock as a material for land use (e.g., as aggregates or for mining reclamation), (e) dry stacking of thickened tailings, and (f) discarding tailings into surface water (e.g., sea, lake, river) or groundwater. The most typical techniques applied nowadays are those commented below, although the tailings dam continues to be the classical chosen method for many mining companies.

Fig. 6.91 Tailings pond (Image courtesy of Anglo American plc)

Tailings Dams

The design, construction, and operation of tailings dams are a major concern for new mining projects as well as for many existing operations. Development features in tailings dam design include aspects such as site selection, site characterization (climate and hydrology, regional geology, terrain analysis, hazard classification), waste characterization, geotechnical design, and legislation requirements (Brown 2002; MMSD 2002). In this sense, two principal features are essential in tailings dam design: safety and economic viability. Regarding the safety area, the dam structure must be designed and constructed in such a way that it will not fail during construction and utilization or later where it is no longer in use. On the other hand, contamination of lakes and rivers must be avoided. But it is also essential that design and construction of dams must be carried out at the lowest possible cost, obviously compatible with safety items. This always implies the location of the dam in close proximity to the mine operation.

The type of tailings embankment is generally determined by the local seismic activity, water clarification, tailings properties and stability, tailings distribution, foundation and hydrological conditions, and environmental conditions (Azizli et al. 1995). The ground underlying the dam must be structurally sound and able to bear the weight of the impoundment. If such a site cannot be found close to the mine, it can be necessary to pump the tailings, at high slurry density, to a suitable location. However, it is economically advantageous to site the impoundment close to the mine, although this imposes limits on site selection. Tailings dams can be built across river valleys or as curved or multi-sided dam walls on valley sides, this latter design facilitating drainage. On flat or gently sloping ground lagoons are built with walls on all sides of the impoundment (Wills and Finch 2016). Since the disposal of tailings adds to the production costs, it is essential to make disposal as cheap as possible. There are three main methods to design a tailings dam: upstream, downstream, and center line (Fig. 6.92).

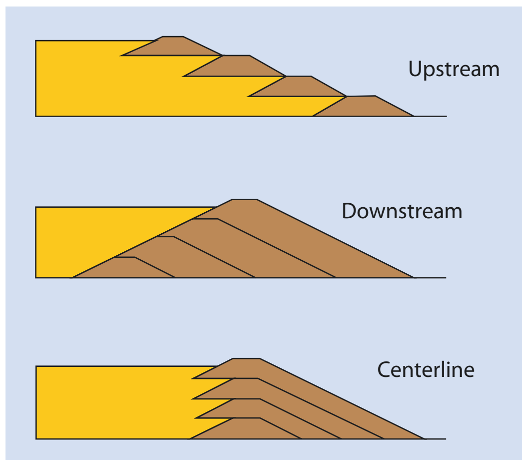


Fig. 6.92 Methods to design a tailings dam

Fig. 6.93 Upstream tailings dam (Image courtesy of Alrosa)

The Upstream Method

The upstream method is the lowest early cost and most common design for a tailings dam in low-risk seismic areas (Fig. 6.93). One of the justifications for this option is principally due to the minimal quantity of fill material needed for starting construction and further raising of the structure, which usually is formed entirely by the coarse product of the tailings. In this technique, as the height of the tailings dam rose, each successive structure moved subsequently to the upstream side and so overlay an unstable bed of unconsolidated tailings. Any change that originates saturation of the lower dykes could quickly lead to dam failure. The tailings are generally discharged by a spigot from the top of the dam crest generating a beach that becomes the foundation for future embankment raises (Vick 1990). The upstream dam should have a downstream slope of less than 1:3, and the beach should be wider than the height of the dam.

Upstream methods are best suited to regions with arid climate. This helps to generate wide beaches and avoid common water level deviations that can dramatically alter pond geometry and the phreatic surface within the impoundment area. The main disadvantage of this technique is the susceptibility of the dam to liquefaction. Thus, the upstream method is the most frequent design to fail, which causes dramatic environmental issues worldwide. Several major failures have involved tailings dam constructed with the upstream method (Van Zyl 2014).

The Downstream Method

The downstream concept was oriented to decrease the risks associated with the upstream technique, especially when subjected to dynamic loading as a result of earthquake shaking. The construction of impermeable cores and drainage areas can also enable the impoundment to hold a significant quantity of water directly against the upstream face of the embankment without jeopardizing stability. The downstream method is basically the reverse of the upstream technique, in that as the dam wall is incremented, the center line shifts downstream, and the dam continues founded on coarse tailings. An advantage of this method is that the raised parts can be designed to be of different porosity to tackle any issues with the phreatic surface of the embankment. This can be especially helpful where a mineral processing installation has made changes to increment effectiveness and as a result disrupt the tailings features. Most procedures include the application of cyclone devices to generate sand for the dam construction. The main disadvantage of this method is possibly the large volume of coarse particles required. Where fine grinding in the processing plant is indispensable, sufficient coarse product is not easy to be obtained.

Downstream dam building is the only technique that ensures design and construction of tailings dams to suit engineering standards. All tailings dams in seismic areas and all major dams, regardless of their location, must be constructed


Fig. 6.94 Tailings discharge by spigot

utilizing some form of the downstream method. The downstream design is versatile for a varied range of site-specific design features and behaves similarly to water retention dams. Their main advantage is that the downstream design can have unrestricted heights since each increment is structurally independent of the tailings. The main disadvantage is the cost of raising the embankment as large volumes of fill are needed, which increment largely as embankment height increases. Moreover, a large area around the dam itself is required as the toe of the dam moves out as more increments are added (Vick 1990).

The Center Line Method

The center line method is in fact an intermediate situation between the upstream and downstream methods of design (Benckert and Eurenius 2001). The embankment crest is being incremented vertically and does not move related to the upstream and downstream positions of further raises, hence the term center line design. It is more difficult to fail than the upstream method but does not need as much construction product as the downstream method. Thus, the dam can be increased more rapidly, and there is less problem maintaining it ahead of the tailing pod in the first steps of construction. Like the upstream method, the tailings are commonly discharged by spigoting (Fig. 6.94) from the embankment crest to constitute a beach behind the dam wall. Where further increment of the dam is needed, material is placed on both the tailings and the existing embankment. By using this method, the forthcoming surface area and therefore the storage capability do not reduce with each dam increment. A fourth design method is the so-called modified center line. This method allows the embankment crest center line to move slightly upstream optimizing the quantity of construction materials required in the downstream shell zone of the embankment.

In-pit Disposal

In-pit tailings disposal is merely the procedure of backfilling abandoned open-pit mines with tailings. This technique is very appealing to a mine operator as worked out voids can be filled at lower cost than other possibilities such as designing, constructing, and operating a conventional, thickened, paste, or dry-stack facility. Another advantage to in-pit storage is that the tailings do not need retaining walls, being on this manner the risks associated with embankment instability eliminated (EPA 1994). Thus, the disposal of tailings in mined-out open-pits can be an alternative to constructing a new tailings storage facility, provided that the open-pit and the tailings have the appropriate characteristics. The main risk associated with this method of disposal is the contamination of the groundwater network by leachates from the tailings. This risk is minimized for open-pits that act as water sinks, that is, groundwater

Fig. 6.95 Underground mine hydraulic backfilling (Spain) (Image courtesy of Matsa, a Mubadala & Trafigura Company)

flows toward the pit, and/or where the bedrock is mostly impermeable (Wills and Finch 2016). Other disadvantages include the possibility that the stability of underground mines near the in-pit storage can be jeopardized or the poor consolidation that can produce long durations of surface deformation once a pit has been filled.

Another possibility related to the disposal of tailings in mines (abandoned or active) is backfilling. It is the reinstatement of products into the mined-out part(s) of the mining site. These materials are commonly overburden, waste rock, and tailings, either alone or combined with other products (e.g., cement). Slurried and dry tailings are sometimes used in underground mines (Fig. 6.95), abandoned pits, or portions of active pits as a backfill. There are different types of mine backfilling such as dry backfill, cemented backfill, hydraulic backfill, and paste backfill. For example, cemented backfill commonly is constituted by waste rock or coarse tailings materials mixed with a cement or fly ash slurry to enhance the bond strength between the rock fragments. The techniques of location all include mixing the rock and cement slurry in a hopper before positioning it in voids (e.g., stopes or mined-out longwall) or percolating a slurry over the rock once it has been located. The waste rock or tailings can be classified or unclassified. Cemented backfill contains a mixture of coarse aggregate (<150 mm) and fine aggregate (<10 mm fraction), being the cement slurry concentration commonly about 55% by weight (1:1.2 water/cement ratio) (European Commission 2009).

Densified Tailings

In some cases, tailings are thickened to 60% pulp density or more or dried to a moisture content of 25% or below prior to disposal. Collectively, these techniques are known as densified tailings. The effectiveness and feasibility of utilizing thickened or dry tailings is based on the mineralization grind as well as the helpfulness of other methods of disposal. Excepting certain circumstances, these techniques can be extremely expensive due to further equipment required and energy expenditures. However, the advantages include diminishing seepage volumes and land required for an impoundment and simultaneous tailings deposition and rehabilitation (Vick 1990). The densified tailings methods include thickened tailings, paste tailings, and dry stacking, which depends on the percentage of solids in the tailings. It usually ranges between 50% and 70% in thickened tailings and up to above 85% in dry stacking.

These are all procedures that enhance water and reagent recovery and reduce tailings amounts and footprint, which clearly assist site rehabilitation. Densified tailings are expected to become more common in operating mines, particularly those with low ore grade and high throughput, which produce large volumes of tailings and require

Fig. 6.96 Densified (dry stacking) tailings dam (Spain) (Image courtesy of Matsa, a Mubadala &Trafigura Company)

large impoundment areas. For those high tonnage operations, reduction in tailings volume has significant impact on dam size and the area of disturbed land (Wills and Finch 2016). Regarding the most densified tailings disposal method, dewatering to higher degrees than paste generates a filtered wet (saturated) and dry (unsaturated) cake that can no longer be translated by the pipeline because this product has a low moisture content. These types of tailings are usually moved by a conveyor or truck, located, spread, and compacted to constitute an unsaturated tailings deposit (Davies and Rice, 2001) (Fig. 6.96). Thus, this tailings storage originates a stable structure commonly needing no retention bunding, being termed dry stack. A typical moisture content of less than 15–20% is obtained by utilizing a combination of belt, drum, horizontal, and vertical stacked pressure plates and vacuum filtration systems (Martin et al. 2002). On the other hand, dry-stack structures are also easier to close and rehabilitate, need a smaller footprint in comparison with other surface tailings storage facilities, and originate better regulator and public perceptions of tailings storage (Davies and Rice 2001).

Subaqueous Disposal

Subaqueous disposal is also a possible disposal method, being an alternative to conventional tailings disposal. In general, subaqueous tailings disposal is the discharge of tailings to rivers, lakes, and seas. Subaqueous disposal can be utilized if onshore methods are not suitable due to high-seismic activity of the terrain, high rainfall, and/or land accesibility (Coumans 2002). In this sense, submarine tailings disposal is probably the most common subaqueous disposal technique, especially for operations that are close to the sea. It includes the deepwater unloading of tailings to the sea. Sometimes, river tailings disposal is used for water-soluble materials such as salt. Thus, some potash operations unload saline waters into rivers.

One of the main objectives of subaqueous tailings disposal is to minimize the contact between atmospheric oxygen and the tailings. It understates the oxidation of reactive materials (e.g., oxidation of sulfides), inhibiting acid generation. The objective is commonly to keep a constant water cover on the tailings during operation as well as after closure. Underwater deposition is a little bit more expensive in comparison with conventional discharging above the water level, but final decommissioning expenditures are drastically lower.

The major disadvantages of this storage method is that submarine tailings management is often considered risky due to the unpredictable characteristics of the tailings flows as they leave a discharge outlet and the potential for pollution migration. The tailings cover large zones on the ocean or lake floor and produce turbidity issues if the disposal technique is not designed properly (Ripley et al. 1978). Another concern is that many features of the subaqueous environment are unknown, and hence an impact study is almost impossible to undertake.

The tailings need to be preferentially located under the euphotic zone (normally >100 m deep), where photosynthesis and reproduction of marine plants take place, and below the surface mixed layer, which is the zone of turbulence generated by wind and waves (MMSD 2002). Regarding the disposal techniques, two principal methods are generally utilized such as a floating pipeline and a submerged pipeline.

Riverine (e.g., rivers, lakes, and lagoons) or shallow marine tailings disposal is not accounted as good international industry technique. Deep-sea tailings placement (DSTP) may be analyzed as an alternative only in the lack of an environmentally and socially sound land-based alternative. If a DSTP is accounted, such consideration should be based on detailed feasibility and environmental and social impact assessment of all tailings management possibilities (IFC 2007).

6.11 Questions

Short Questions

- What is mineral processing?
- Explain the concept of penalty elements in mineral processing
- What net smelter return means?
- List the major steps or stages in mineral processing.
- What are the major physical methods used to concentrate ore?
- What comminution means? Explain the main goal of comminution.
- What is a vibrating screen for screening process?
- Describe the hydrocyclone device used in classification process.
- What is a shaking table in gravity concentration?
- Explain briefly the dense medium separation process.
- Explain the importance of froth flotation process in mineral beneficiation.
- What leach-solvent extraction-electrowinning process (SX/EW) means?
- What is a filter medium in dewatering process?
- Explain the concept of tailings in mineral processing and the main methods to design a tailings dam.
- What are the objectives of densified tailings?

Long Questions

- Explain in detail the autogenous and semiautogenous mills.
- Explain the principles of froth flotation.

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Environment and Sustainability

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Summary
This chapter draws attention to all topics related to environment and sustainable development in the mining world. In this sense, financial aspects (e.g., the Equator Principles) are essential to meet the sustainable development objectives. The description starts with some comments about the final stage of the mining cycle: closure and reclamation, and the concept of an Environmental Management System. The «social license to operate,» which is the acceptance of a mining operation by stakeholder communities, is featured below. The potential environmental and social impacts and their management are then discussed. Environmental and social impact assessments are also emphasized because the integration of environment into development scheduling is the essential tool in achieving sustainable development. Finally, eight reclamation case studies are developed at the end of the chapter.

7.1 Mining and the Environment

7.1.1 Introduction

Society needs mining industry since products derived of this activity improve our wealth and quality of life and allow society to growth. In parallel, new technologies that help to minimize human impact on the Earth's environment require metals and other mineral products (Stevens 2010). However, the environmental impacts of mining are perhaps part of the price that humankind has to pay for the benefits of mineral consumption because some environmental degradation due to mining is unavoidable.

In the recent past, mining was carried out with little concern for its effects on the environment, resulting often in significant environmental damage. As political and cultural norms evolved and new legal requirements were enacted, almost all major mining companies adopted rigorous policies and procedures for sustainability, community engagement, and environmental risk assessment and mitigation. These companies apply such policies throughout their operations, many of which are worldwide. Up to date, many mining companies work actively to remediate environmental damage caused by historic mining operations in areas where they developed operations and/or have current activities. Thus, environmental considerations are an important part of the modern mining industry. They must be included in all project plannings, and feasibility studies must account for the influence of environmental considerations on project schedules and costs (Nelson 2011). It is clear today that a zero-harm environment is achievable and that all fatalities, occupational diseases, and injuries are preventable.

The mining industry has followed the same trend as our society. Big mines affect surrounding environment similar to other industrial operations. In the second half of the twentieth century, the mining industry developed a better understanding of its impact on the environment. Today, mines are designed, developed, operated, and closed in an environmentally sound manner, and considerably effort is put into continually improving environmental standards (Stevens 2010). Consequently, the mining industry has changed in the last decades. It is not the industry it was 100, 50, or even 30 years ago. Modern mines operate under modern laws that place far greater importance on environmental protection and use knowledge and technologies that limit the impact of a mine on the environment. Moreover, the modern mining industry also considers the environment in a broader context than in the past. Today, it is not just about the physical environment but also the social and economic environment in which a mine operates. Therefore, the policy makers and organization conducting the mineral development must foster the social well-being of the people living in the mining areas. Mines must have the support of the communities and countries in which they operate in order to be successful. This support is garnered ensuring high standards of environmental stewardship (Haldar 2013).

Large mining operations affect surrounding communities, flora and fauna, land and water, similar to other major industrial operations. The extent to which mining becomes an environmental impact depends largely upon the number of people that a mine affects. High quantities of waste are a consequence of most mining and quarrying operations. Although the major part of this waste is inert and nonhazardous, disposal is often a space problem, at least in densely populated areas. Since economic growth cannot take place without mineral raw materials, the rational conclusion is that the exploitation of mineral resources is not the problem, but it must be developed in a green and modern execution (Pohl 2011). Obviously, the larger the size of a mining operation, the larger the impact is likely to be because it will produce more waste, occupy more land, and have a greater number of buildings.

Old mining works commonly dumped wastes without interest for their physical or chemical stability and the disposal of waste has led to the pollution of surface streams and groundwater. Moreover, urban areas have suffered subsidence damage by underground mining. Thus, although the mining companies generally showed a lack of concern for the environment, this does not indispensably mean that society was not aware of the environmental issues that could be generated with mining. For example, in 1306 a Royal proclamation prohibited the use of coal in London for domestic and industrial purposes because of the nuisance caused by smoke, but it proved impossible to enforce. In addition, Agricola (1556) commented the environmental issues generated by mining such as the devastation of fields and the contamination of streams.

The greater awareness of the importance of the surrounding environment has led to tighter regulations being implemented by many countries to lessen the impact of mining operations. The concept of reclamation of a site after mining works has entered definitely in the country laws. Thus, in most developed countries, mining is closely regulated now, and environmental impacts are increasingly being controlled. Modern mines are bound by present environmental legislations that are becoming strict in the developed world. In this sense, it is important to remember that there is legacy of older operations in most countries worldwide, many of which have been abandoned. Therefore, there is a combination of modern impacts (the impact of current mining in developing countries is still more marked) matched with ancient legacies.

Due to the above, mining companies are carrying out considerable efforts to decrease the environmental impact of mine works and diminish the footprint of their operations throughout the mining cycle, including working to reclaim ecosystems post-mining. To achieve this objective, many mining companies have developed their own codes of practice to assure that mining operations do not so significant harm to their surroundings.

7.1.2 Closure and Reclamation: The Final Stage

All mines have a finite life, and, once the ore is extracted, the mine will close, and the mine site will be reclaimed to a productive natural state. Thus, generally the final step in the operation of mine works is closure and reclamation, the procedure of closing a mine and recontouring (Fig. 7.1), revegetating, and restoring the water and land features to the previous configuration. Closure and reclamation plans are part of mine planning and environmental assessment and must be in place prior to mine development (Stevens 2010).

Before the incorporation in the 1970s of mine closure requests and best practices in the laws, mines were commonly abandoned (Fig. 7.2). Thus, mine land reclamation and closure planning are actually needed by regulatory agencies worldwide, and they are frequently a part of the environmental impact assessment procedure carried out in many countries. Regarding the financial assurance requirements of a closure and reclamation plan, it is the responsibility of the mining company to pay for closure and reclamation costs. To avoid mine abandonment, mining companies are ever more requested to supply financial warranty in the form of a deposit or bond to governments and communities as a guarantee that the resources to meet closure needs will be available.

Closure

In a perfect world, mine works only finish their activity if the mineral resources are exhausted and a mine closure planning is gradually implemented. However, in the real world, mines extract reserves not resources, and the grade and tonnage of reserves change daily based on the commodity price, mineralization grades, geotechnical issues, and other features that can produce the closure before the calculated reserves have been wholly mined. In these cases, the reputation of the mineral industry is dependent on the legacy it leaves. The reasons why mine works close preterm are numerous, including low commodity prices or high expenditures, reducing grade of the mineralization, unfavorable geotechnical conditions, policy changes, social or community influences, and many others. This closure position

Fig. 7.1 Recontouring the waste dumps for reclamation (Spain)

Fig. 7.2 Abandoned facilities without reclamation (Image courtesy of Miguel Cabal)

previous the entire extraction of the mineralization can generate important issues for the mining enterprise, the community, and the regulator (Commonwealth of Australia 2006a). The concept of «community» is usually used in the minerals industry to describe those who live in the geographic region of an operation, either in defined settlements or dispersed settings.

A mine closure plan including physical rehabilitation and socioeconomic aspects must be an integral portion of the project life cycle and should be determined so that (a) the future public health and safety are not compromised, (b) the after-use of the site is beneficial and sustainable to the affected communities in the long term, and (c) the adverse socioeconomic impacts are minimized and socioeconomic benefits are maximized (IFC 2007). Moreover, the closure plan is a dynamic document that must be constantly updated to express variations in mine development and operational planning as well as the environmental and social conditions. Closure and post-closure planning should incorporate adequate aftertreatment and continuous monitoring of the mine site. The duration of post-closure monitoring can be organized on a risk basis; however, site features commonly need a minimum period of 5 years following closure or longer.

Whether an operation has 10 or 50 years of operational life remaining, implementing closure plan into the mining business originates a great value for both the company and the wider community. For this reason, mining companies must involve governments, communities of which they are part, and other stakeholders in closure planning to achieve a successful closure outcome. External mining stakeholders such as local communities, conservation groups, and biodiversity advocates are becoming more and more sophisticated about the outcomes of good and bad closure planning practices (Bingham 2011).

Planning for a successful closure is a complex, multidisciplinary task that is essential to minimizing long-term risk for the mining company, the environment, and the affected stakeholders. Assessing closure risk requires a systematic, structured evaluation. Thus, the risk assessment forms the basis of the closure plan and cost estimates. The focus of the closure risk assessment will be to establish an acceptable risk profile for the company and all other stakeholders upon completion of the closure project. To address closure planning issues and meet business objectives to manage risk, it is necessary to assure that closure planning is wholly integrated in the core business of the asset (Bingham 2011). Additionally, the detail and accuracy of closure plans must change through the life cycle of the asset, starting out as conceptual and progressively becoming more detailed over time.

Closure Objectives

Closure objectives establish the closure results for the mining project and must be realistic and attainable. They can form the requirement to restore a site to its original state and rehabilitating the site to a condition compatible with the surrounding terrain. These goals are designed in accordance with the suggested post-mining land use(s) and are as precise as possible to afford a specific indication to the government and the community on what the proponent commits to attain at closure. Timing and the methods to achieve these objectives are through the life of the asset; life cycle is commonly very specific. For example, some of mines may not enable for any concurrent or progressive reclamation during the operating stage since the disturbed areas are in constant utilization during the mining works, while other mining operations (e.g., several types of coal mines) display generally the possibility to carry out reclamation activities during the life of the asset.

The Environmental Protection Authority of Western Australia summarizes the main closure objectives of a mining project:

1. Landforms: constructed waste landforms will be stable and consistent with local topography; constructed tailings storage facilities will be nonpolluting and non-contaminating, and toxic or other deleterious materials will be permanently encapsulated to prevent environmental impacts; surface water bodies shall not be left in mining voids unless the operator demonstrates there will be no significant environmental impact (e.g., salinization, reduction in water availability, toxicity, algal problems, attraction to pest species, or a local safety hazard) (Fig. 7.3).
2. Revegetation: vegetation in rehabilitated areas will have equivalent environmental values as surrounding natural ecosystems; soil properties will be appropriate to support target ecosystem.
3. Fauna: rehabilitated areas will provide appropriate habitat for fauna; abundance and

Fig. 7.3 Water sampling (Image courtesy of Kinross Gold Corporation)

diversity of fauna must be present in appropriate proportions given the specified post-mining land use.

4. Water: surface and groundwater hydrological patterns/flows will not be adversely affected; any water runoff or leaching from tailings dams, overburden dumps, and residual infrastructure shall have quality compatible with maintenance of local land and water values.
5. Infrastructure and waste: during decommissioning and through closure, wastes will be managed consistent with the waste minimization principles; no infrastructure left on-site unless agreed to by regulators and post-mining land managers/owners; disturbed surfaces must be rehabilitated to facilitate future specified land use; the location and details of any buried hazards will be clearly defined, and robust markers will be installed and maintained (EPA 2015).

Financial Assurance

In recent years, numerous specialty documents and guidelines have been prepared by governments, industry associations, and nongovernmental organizations (NGOs) on the subject of financial assurance for the closure activities, including those available through the International Council on Mining and Metals (ICMM). Although these documents and guidelines vary greatly country to country, it is essential that the closure team properly reviews the site-specific regulatory requirements for estimating financial assurance. Minimum considerations about this item must incorporate the accessibility of all funds to cover the costs associated to mine closure at any stage in the mine life, including provision for early or temporary closure. In the case of a financial guarantee, an acceptable manner of financial guarantee must be presented by a renowned financial institution (IFC 2007).

Governments should apply the financial assurance requirements for contributing to the goal of environmental protection but do not put pressure to existing operators and result in premature closure. The timing and nature of new requirements as well as transition provisions should be set accordingly (ICMM 2005). In this sense, it is important to note that efficient environmental financial assurance policies reduce the scope for public criticism of mineral industry practices (ICMM 2005). Since predictive mine works' reclamation costs are very difficult since they are usually inexact, especially where long-term care is needed, governments commonly include a «safety factor» into the amounts of EFA environmental financial assurance applied.

Closure Plan Stages

In general, there are three basic stages to developing an effective closure plan. The first stage is the development of a target closure outcome and goals, which are manifested in a conceptual closure plan. This plan is developed and used during exploration, pre-feasibility, feasibility/design, and construction to guide the direction of activities.

Fig. 7.4 Restoration of the land surface (Spain) (Image courtesy of Carlos García)

Its active life can be 3–5 years. If well defined and based on effective community and stakeholder engagement, it cannot change much during this time. The second stage involves the ongoing development and implementation of a detailed closure plan, which increases the understanding of specific goals and milestones as well as the actions and outcomes of activities to meet these. This plan is used continuously during operations and has an active life that could range from 5 to 30 years or more; obviously, during this time it must be updated. The final stage is the effective transition to closure, which can be manifest as a decommissioning and post-closure plan. Its active life can be as little as a year or two, although it can extend many years past that time depending on post-closure responsibilities (ICMM 2008).

Reclamation

Mine reclamation is the procedure of taking land after utilized by mining operations and changing it into land with alternative uses. Reclamation includes aspects related to surface and groundwater and air purity, erosion issues generated from storm water and sometimes wind, revegetation of appropriate plant species, and wildlife habitats. The best time to start the reclamation procedure of mine works and associated installations is just before the first excavations are undertaken.

In planning for the reclamation of a given mine, there are many issues that must be considered. The first of these is the safety of the mine site, especially if the area is open to the general public. The second major issue is rehabilitation of the land surface (Fig. 7.4), the water quality, and the waste disposal zones so that long-term water contamination, soil erosion, dust production, or vegetation issues do not occur. The final concern is the subsequent use of the land after mining is completed. The last stage in reclamation is monitoring. In this process, all reclaimed areas are monitored and assessed for vegetation survival and growth rates. Plants in areas that are to be used for grazing will be tested to ensure that they contain acceptable levels of metals and other possible contaminants (Stevens 2010).

Reclamation has been used in a general way simply to mean returning a mine site to some other land use whether it be the same as before mining began or different. It includes the physical stabilization of the land (e.g., waste rock piles), landscaping, rehabilitating topsoil, and return of the land to a helpful finality. The art of mine reclamation has progressed from straightforward revegetation operations to a more complex discipline that includes utilization of native plants to mimic natural ecosystem. In most cases, entire reclamation is almost impossible, but sound remediation and rehabilitation can result in the opportune setting of a functional ecosystem.

By planning the mine for a subsequent development, it is possible to improve the value of the disturbed land and help to change it to a utilization that the public will consider clearly positive. Thus, old mine sites can be converted to wildlife habitat and refuge (Box 7.1: Cabárceno Natural Park)

Box 7.1

Cabárceno Natural Park (Santander, Spain)

Cabárceno Natural Park (Fig. 7.5) is located in Pisueña Valley, 17 kilometers from Santander (Spain), being probably the biggest tourist attraction in Cantabria (Spain). It is so successful than more than five million people have visited it since its opening in 1990. It is a man-made space created from the karst landscape of a former open-pit iron mine. Iron mining was the oldest and most common one in Cantabria, as shown by the abundant remains of exploitations. The Peña Cabarga iron mines in Cantabria, Spain, were active for more than 2000 years (Pliny the Elder wrote about these mining works). The mining exploitation of Peña Cabarga (Cabárceno) stands out in the central area of Cantabria. This mine benefited from iron oxides and hydroxides filling the karstic cavities, coming from oxidation and hydration of the iron sulfides (pyrite and marcasite) within the dolostone rock. As the iron ore was developed, the column shapes from the karst were almost cleaned. The result is a ruin-like landscape of great beauty that nowadays is used as a natural park. Cabarceno's iron was extracted until 1989, when the mine was no longer profitable, and the conditioning of the Natural Park started.

The natural park is home to a hundred animal species from five continents living in semi-free conditions, which are distributed in large enclosures where one or more species coexist. Cabárceno covers an area of more than 750 hectares and is the largest park of its kind in Europe. More than 20 kilometers of roads cross the park, leading to gorges, lakes, and rock figures, and several lakes that were open-pit exploitations complement this exceptional space. Cabárceno Park has two main objectives: (a) conservation of endangered species and (b) environmental education. A network of roads, very well laid out and tens of kilometers in length, make it possible to contemplate a variety of fauna that finds protection, refuge, and food in the park. The visitor can observe at close distance, in 21 ample areas, hundreds of all the zoological

Fig. 7.5 Cabárceno Natural Park (Santander, Spain)

community's animals: jaguars, giraffes, lions, Siberian and Bengal tigers, leopards, hyenas, bisons, elephants, hippos, rhinos, dromedaries, camels, llamas, zebras, ostriches, and many others as well as Cantabria's fauna including wolves, deer, wild boars, and the most important Hispanic reserve of brown bears.

The facilities that host animals are internationally recognized as one of the best existing in the world. The Park welcomes different animal species from five continents in a semi-free environment, which have been distributed in boxes of large areas where one or more species can coexist. The Nature Park's life develops in the most natural environment for these animals. Besides the food provided them, the rest of the activities are marked, almost, by their complete freedom and instinct as wild as in their natural habitat. The park collaborates with zoos and partnerships in the conservation of endangered species like tigers, lions, bobcats, rhinos, etc. For instance, the park has pioneered, in collaboration with the Deutsches Primatenzentrum and the University of Göttingen (Germany), in the development of techniques that have allowed the knowledge of the sexual cycle in female African elephant by noninvasive methods.

recreational areas (◘ Fig. 7.6), shopping mall, golf course, airport, lake, underground storage facility, solid waste disposal area, mining and power plant waste storage, museum, site of special scientific interest and regionally important geological site, industrial land, pisciculture pond, and many other economically or ecologically productive land utilizations that can benefit society. The conversion of an abandoned mine for practical commercial purposes depends upon the geological and hydrogeological conditions as well as the nature and geometry of the mining that took place.

There is a great variety of terms used in mining reclamation. The terms remediation, restoration, rehabilitation, and reclamation itself are all applied to express mine closure activities that attempt to alter the biological and physical state of a site. However, they have slightly different meanings. They are many times utilized interchangeably, but refer to different stages in the preparation of the site for another utilization. Thus, remediation is the cleanup of the polluted area to safe levels by extracting or isolating contaminants. At mine sites, remediation commonly consists of isolating contaminated material in pre-existing tailings storage facilities, capping tailings and waste rock piles with clean topsoil, and gathering and processing polluted mine water. As far as restoration

◘ **Fig. 7.6** Aggregate quarry reclaimed to recreational area (Las Madres, Madrid, Spain) is concerned, it commonly refers to returning a mined area to its previous condition and land use such as where a surface mine is filled and the restored land returned to agriculture.

The meaning of rehabilitation and reclamation in the context of mining is not as widely accepted as the meaning of restoration. Nevertheless, rehabilitation (also referred to as regeneration) can be regarded as the establishment of a stable and self-sustaining ecosystem, but not indispensably the one that existed previous mining works started. Therefore, rehabilitation is the procedure utilized to remedy the impacts caused for the mining activities on the environment. The long-term aims of rehabilitation procedure can change from merely converting an area to a safe and stable condition to restoring the pre-mining conditions as closely as possible to support the future sustainability of the site.

7.1.3 Environmental Management System

An Environmental Management System (EMS) is an essential part of a larger management system or an organization. The EMS is utilized to define an environmental policy and to control the environmental aspects of the organization activities, products, and services. This control includes interrelated components such as responsibilities, authorities, relationships, functions, processes, procedures, practices, and resources. A management system utilizes these components to define policies and goals and to develop ways of using these policies and achieving these objectives.

Thus, EMS is a group set of procedures and practices that allow an organization to decrease its environmental impacts and increment its operating effectiveness, being a powerful tool for managing the unfavorable impacts of activities of an organization on the environment aspects. The profits of an Environmental Management System are the following: (a) diminishes the environmental responsibilities applying well-defined mitigation techniques, (b) increases the effective utilization of resources, (c) decreases waste production by appropriate planning, (d) proves a well-accepted corporate image, (e) motivates awareness of environmental concern, (f) increments better knowledge of environmental impacts of business activities, and (g) increments the skill and effectiveness generating higher productivity at lesser costs and higher benefits.

Applying an EMS, the company can prove to all people that they take environmental impacts actively. Moreover, an efficient EMS can also enhance company operations and generating economic profits. The bigger organizations decide certification is more meaningful when taking into account the potential trade and market benefits of an internationally identified and certified EMS. For this reason, ISO 14000 families of certifications assure diminishing the negative effect of operations on environmental aspects and comply with applicable laws, regulations, and other environmentally oriented mitigations. All these standards are periodically reviewed by ISO to assure that they still meet market needs.

It is important to note that Environmental Management Systems do not by themselves define environmental objectives. Rather, they only lead the management procedure of a company to assure that environmental programs can be efficiently developed. Setting of policies and goals is one of the important functions defined within such a management system. ISO 14001, issued in 2004, offers standards by which an organization may put in place and implement a series of practices and procedures that, when taken together, result in an Environmental Management System (EMS) (Box 7.2: Environmental Management – ISO 14001). Other relevant families of certification used in an EMS are ISO 9000 and ISO 18000. For

Box 7.2

Environmental Management – ISO 14001

The ISO 14001 standard is the most important within the ISO 14000 series, and it sets out the criteria for an Environmental Management System (EMS). ISO 14001 is the internationally accepted environmental management standard that certifies that an organization is committed to reducing the environmental impact of its products and operations and is constantly monitoring and seeking to identify ways of reducing that impact further. It prescribes controls for those activities that have an effect on the environment. These include the use of natural resources, handling and treatment of waste, and energy consumption. Thus, the standard requires the company has a procedure for monitoring that regulatory requirements are being met.

International standard ISO 14001:2004 from the International Organization for Standardization (ISO) defines an Environmental Management System (EMS) as «Organization structure, responsibilities, practices, procedures, processes, and resources for implementing and maintaining environmental management.» It is a flexible, risk-based, plan-do-check-act continual improvement approach that requires formal documented processes for many of its elements.

Most mining companies are committed to managing its environmental aspects, impacts, and risks through adherence to the internationally recognized ISO 14001: 2015 EMS standard. It is known as a generic management system standard, meaning that it is relevant to any organization seeking to improve and manage resources more effectively. Today, many large-scale mines operating worldwide have already attained ISO 14001 certification. The ISO 14001 EMS standard requires every mining company the highest, most acceptable level of efficiency in terms of extracting minerals while at the same time ensuring that the environment is not compromise.

The five main stages of an EMS, as defined by the ISO 14001 standard, are (Fig. 7.7):

1. Commitment and policy: top management commits to environmental improvement and establishes the organization's environmental policy.
2. Planning: an organization first identifies environmental aspects of its operations and then determines which aspects are significant by choosing criteria considered most important by the organization.
3. Implementation: an organization follows through with the action plan using the necessary resources (human, financial, etc.); an important component is employee training and awareness for all employees.
4. Evaluation: a company monitors its operations to evaluate whether targets are being met.
5. Review: top management reviews the results of the evaluation to see if the EMS is working; management determines whether the original environmental policy is consistent with the organization's values; the plan is then revised to optimize the effectiveness of the EMS.

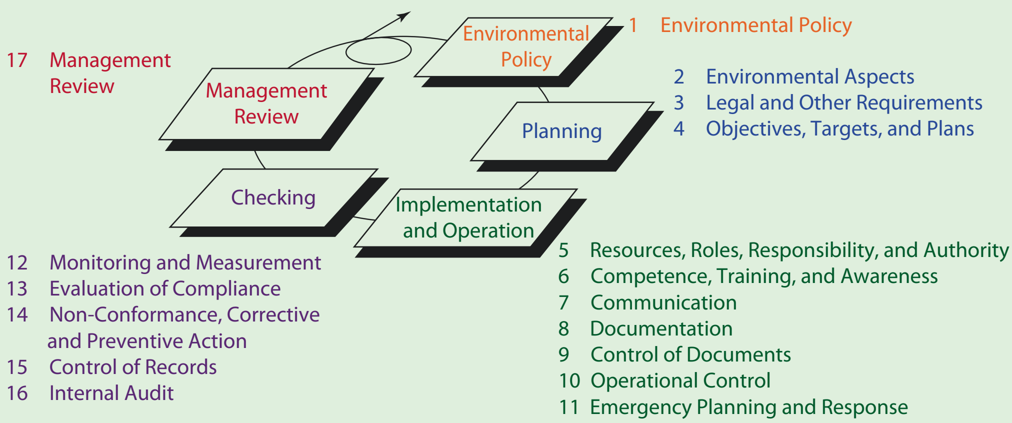

Fig. 7.7 Main stages of an EMS as defined by the ISO 14001

example, the former ensures quality system management, and it is established to help organizations that they satisfy the requirements of customers.

7.2 Mining and Sustainable Development

In the last three decades, the terms «sustainable development» and «sustainability» have been in use by the governments and policy makers worldwide. It is commonly agreed that sustainable development was early defined in 1987 by the Brundtland Commission (Our Common Future, World Commission on Environment and Development, United Nations) as «a system of development that meets the basic needs of all people without compromising the ability of future generations to meet their own life-sustaining needs.» Since then, a rich discussion has ensued about what this means in practical terms. Though many other sets of words have been suggested for defining sustainable development, the Brundtland Commission definition has stood the test of time. In mining world, these words were possible first utilized in the early 1990s in the Rio Summit (1992).

In recent years, the word sustainability has also found its way into common use. The ideas of sustainable development and sustainability are different but synchronous. Sustainability is a more general term that captures the idea that we need to maintain certain important aspects of the world over the long term. These features vary from primary requirements of human society such as air, water, food, clothing, shelter, and basic human rights to a group of perceptions that would collectively be termed «quality of life» and not only for people but also for other forms of life. Sustainable development is the human or action part of this set of ideas. Together, these ideas are very appealing, but their translation to practical action remains much debate. This is not surprising since there are about 200 countries across the world, and the global ecosystem is complex and not fully understood (Hodge 2011).

At the base of the interlinked ideas of sustainability and sustainable development is the easy perception that the human activities, obviously including mining, should be carried out in such a way that the activity itself and the products originated together afford a net contribution to human and ecosystem well-being over the long term. An optimum balance clearly requires to be maintained between sustainable development and eco-friendly environment (Haldar 2013). Since the release of «Our Common Future,» including the aforementioned Brundtland Report, many of the major industries in the world, including mining companies, many of its governments, and the United Nations have adopted a policy of sustainable development.

In this sense, the mining industry has been a particularly active locus of sustainability-related policy and practice innovations because: (a) the potential implications of mining activities and the minerals and metals that result are significant; (b) many interests are touched by mining; (c) the role of many of these interests in decision-making is growing (e.g., communities and indigenous people); (d) the nature of contemporary communications systems has brought the often dramatic nature of mining operations into the public eye; and (e) industry, governments, civil society organizations, and the public, in general, are all anxious to ensure mining makes a positive contribution that is fairly shared (Hodge 2011). However, the focus is not on how mining can be sustainable, identifying that mining operations has a finite useful life, but on how mining industries can help to sustainable development. This is a conceptual change from a singular analysis and mitigation of impacts to a more complete study that looks at the broader contribution of the industry and its products (ICMM 2012a). In this sense, financial aspects are essential to meet the sustainable development objectives (Box 7.3: the Equator Principles).

Box 7.3

The Equators Principles

Before financial institutions invest in mining projects, they require that companies produce evidence of a business program that adequately addresses sustainability issues in their projects; they apply stringent rules on resource companies looking for funding. In this sense, many banks belong to the Dow Jones Sustainability World Index (DJSWI). The index was launched in 1999 as the first global sustainability benchmark, and it tracks the performances of sustainable companies and provides money managers with tools to better manage their eco-conscious portfolios. Moreover, many financial institutions have also adopted the Equator Principles (EP), which ensure that projects are developed in a manner that is socially responsible and reflects sound environmental management practices. However, it needs to be recognized that the DJSWI and the Equator Principles are voluntary and nonbinding, and many investors, particularly in developing countries, are not required to adhere to them.

The Equator Principles are a voluntary set of standards adopted by financial institutions for determining, assessing, and managing environmental and social risk in project finance activities.

They are considered the financial industry gold standard for sustainable project finance. The Equator Principles Financial Institutions (EPFIs) have adopted the Equator Principles in order to ensure that the financed projects are developed in a manner that they are socially responsible and reflect sound environmental management practices. Thus, the importance of climate change, biodiversity, and human rights is recognized, and negative impacts on project-affected ecosystems, communities, and climate should be avoided where possible. If these impacts are unavoidable, they should be minimized, mitigated, and/or offset. EPFIs review the Equator Principles from time to time based on implementation experience and in order to reflect ongoing learning and emerging good practice.

The EP are primarily intended to provide a minimum standard for due diligence to support responsible risk decision-making. According to this, EPFIs commit to implementing the EP in their internal environmental and social policies, procedures, and standards for financing projects and will not provide project finance or project-related corporate loans to projects where the client will not, or is unable to, comply with the EP. Where a project is proposed for financing, the EPFI will, as part of its internal environmental and social review and due diligence, categorize it based on the magnitude of its potential environmental and social risks and impacts (principle 1 – Review and Categorization). Such screening is based on the environmental and social categorization process of the International Finance Corporation (IFC). Using categorization, the EPFI's environmental and social due diligence is commensurate with the nature, the scale and stage of the project, and the level of environmental and social risks and impacts. The categories are as follows: (a) category A, projects with potential significant adverse environmental and social risks and/or impacts that are diverse, irreversible, or unprecedented; (b) category B, projects with potential limited adverse environmental and social risks and/or impacts that are few in number, generally site-specific, largely reversible, and readily addressed through mitigation measures; and (c) category C, projects with minimal or no adverse environmental and social risks and/or impacts. Obviously, the Equator Principles have greatly increased the attention and focus on social/community standards and responsibility since 2010. They include robust standards for indigenous peoples, labor standards, and consultation with locally affected communities within the project finance mining market. The most important lending institutions worldwide, many of whom provide financing for mining activities, have adopted the Equator Principles. Thus, currently 83 EPFIs in 36 countries have officially adopted the EP, covering over 70 percent of international project finance debt in emerging markets.

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Today, mining companies employ the principles of sustainable development in their environmental policies. This has resulted in positive development for the industry and has allowed mining companies to view the impacts of their operations in a more comprehensive manner. The process has not been easy; conflict is still present and consensus is not always possible (Stevens 2010). In summary, a sustainable mining operation must be safe, proves significant practices in EMS and community engagement, is financially robust, and which, very importantly, effectively utilizes the mineral resource. Thus, mine managers establish a sustainable mining operation if they focus on the five areas: safety, environment, economy, efficiency, and community (Laurence 2011). At present, almost all mining companies include in their Web pages a heading or subheading entitled «Sustainability» or «Sustainable Development.» Thus, headings such as socioeconomic development, environment, community, or indigenous relations are common, and annual sustainability reports updated regularly are available in almost all mining Web pages. But application of sustainability concepts to the mining, minerals, and metals industry needs attention paid to the full project and mineral life cycles, that is, including exploration, design and construction, operation, closure, and reclamation.

In the late 1990s and faced with growing concern about access to capital, land, and human resources, the chief executive officers of nine of the world's largest mining companies took an unprecedented step. Working through the World Business Council for Sustainable Development, they started the Global Mining Initiative (GMI). They commissioned the International Institute for Environment and Development (London) to carry out a global review that would lead to identification of how mining can contribute in the best form to the transition to sustainable development. The resulting project, «Mining, Minerals, and Sustainable Development,» sparked a large and rich literature, including the final report of the project, entitled «Breaking New Ground: Mining, Minerals, and Sustainable Development.»

As a direct result of MMSD, the International Council of Mining and Metals (ICMM) was founded at 2001. It is an international organization devoted to enhance the social and environmental performance of the mining industry.

Formed by 23 mining and metal companies and 34 regional and commodity associations, ICMM represents the views of most of them in addressing the core sustainable development issues facing the industry. In May 2003, ICMM's CEO-led Council committed member companies to implement and measure their performance against ten sustainable development principles. They are based on the issues identified in the Mining, Minerals, and Sustainable Development project, and all ICMM member companies have committed to following this set of ten principles.

The ten principles published by ICMM are the following:

1. Implement and maintain ethical business practices and sound systems of corporate governance.
2. Integrate sustainable development considerations within the corporate decision-making process.
3. Uphold fundamental human rights and respect cultures, customs, and values in dealings with employees and others who are affected by our activities.
4. Implement risk management strategies based on valid data and sound science.
5. Seek continual improvement of our health and safety performance.
6. Seek continual improvement of our environmental performance.
7. Contribute to conservation of biodiversity and integrated approaches to land use planning.
8. Facilitate and encourage responsible product design, use, reuse, recycling, and disposal of our products.
9. Contribute to the social, economic, and institutional development of the communities in which we operate.
10. Implement effective and transparent engagement, communication, and independently verified reporting arrangements with our stakeholders.

Regarding the role of the United Nations in sustainable development and mining, in «The Future We Want» Resolution (Res/66/288 – 2012), the United Nations contributed with the following considerations: (a) that minerals and metals make a major contribution to the world economy and modern societies; (b) that mining industries are important to all countries with mineral resources, in particular developing countries; in this sense, mining offers the opportunity to catalyze broad-based economic development, reduce poverty, and assist countries in meeting internationally agreed development goals when managed effectively and properly; (c) that countries have the sovereign right to develop their mineral resources according to their national priorities and responsibility regarding the exploitation of resources described in the Rio Principles; and (d) that mining activities should maximize social and economic benefits and effectively address negative environmental and social impacts. The Resolution claims to governments and businesses to foster the continued enhancement of responsibility and transparency and the efficiency of the significant existing mechanisms to avoid the illicit financial flows from mining activities.

Moreover, building on the Millennium Development Goals (MDGs), the 17 Sustainable Development Goals adopted by all United Nations member states in 2015 after extensive global consultation process seek to rebalance and integrate the economic, social, and environmental pillars of sustainable development, with a central focus on people, planet, prosperity, and peace. In order to align the activities of the mining sector with these global goals, the United Nations Development Programme (UNDP) has conducted a mapping exercise that identifies the key ways in which mining activities can have a positive or negative impact on the achievement of each of the SDGs. The mapping exercise they have undertaken has identified those positive direct and indirect impacts of mining on sustainable development that should be enhanced and those negative impacts that must be mitigated. Mining activities do not always produce economic and social profits to the countries in which the operations are located since the mines in some cases are situated where there is bad governance, including corruption.

7.3 Social License to Operate

At a meeting with World Bank personnel in Washington in 1997, Jim Cooney, at that moment director of international and public affairs with Placer Dome, proposed that the industry had to act positively to recover its reputation and gain a «social license to operate» in a process that, begin-

Fig. 7.8 Local community members are essential to obtain and maintain the social license to operate (Image courtesy of Anglo American plc.)

ning at the level of individual mines and projects, would, over time, create a new culture and public profile for the mining industry (Thomson and Boutilier 2011). The concept of a social license to operate (or simply social license) soon entered in the vocabulary of the industry, civil society, and communities that host mines and mining projects. Thus, the concept is in fact an outcome of sustainability. The social license has been defined as existing where a mine or project has the ongoing approval within the local community and other stakeholders. This includes not only local communities (Fig. 7.8), indigenous people, and governments but also the international community. Inherent in this concept is the belief that local communities should benefit from the mining project (Stevens 2010). Therefore, mining companies must communicate openly with all interested parties and stakeholders, and they must have a solid sustainability record in order to have the social license.

Social license to operate is intangible, dynamic, and nonpermanent because beliefs, opinions, and perceptions are subject to vary as new information is obtained. Hence, the social license has to be gained and later retained. It is commonly granted on a site-specific basis. Thus, a company can have a social license to operate for one mine but not for another one. Obviously, the larger the effects, the more difficult it becomes to get the social license to operate. Moreover, the term «to operate» is in some cases confusing with the exclusively operational phase of a mine life cycle where mineralization is extracted for processing. A better sense of the term to operate is to continue the project, no matter where in the mine life cycle, from starting exploration to closure and reclamation (Thomson and Boutilier 2011). The exploration stage is especially important because that is when first impressions are made. It is a challenging period that can affect community relations during the whole mine life cycle. A positive relationship can lead to the early acquisition of a social license. If that is maintained, it can create the tolerance and mutual understanding needed to deal with conflicts and different interests during the whole life of the mine.

The normative components of the social license include the community/stakeholder perceptions of the legitimacy and credibility of the mine or project and the presence or absence of true trust. These elements are obtained sequentially and are cumulative in building toward the social license. The mine or project must be seen as legitimate before credibility is of value in the relationship, and both must be in place before meaningful trust can develop. These concepts are extended in the following subheading. Sometimes, the social license can transcend approval if an important part of the community and other stakeholders include the project into their identity. At this level, it is common for the community to become defenders of the mine or project since they consider themselves to be co-owners and emotionally involved in the future of the mine or project.

The license is granted by the community, a term used generically to describe the network of stakeholders that share a common interest in a mining or exploration project and make up the granting entity. Use of the terms community and stakeholder network implies that the license is not granted by a single group or organization. It is a collective approval granted by a network of groups and individuals. Therefore, the existence of a handful of supporters in the middle of a larger network of opponents would mean that the license has not been granted. However, the condition that the license be a sentiment for a very different group of individuals originates great complexity into the process. In this sense, individuals and groups will cooperate with the company for many reasons, including courtesy, a desire for gain, a perception of having no alternative, or, as is common in many cases, a sense of obligation with authorities. Cooperation for these motives does not indispensably need a confidence relationship.

7.3.1 Phases of Earning a Social License

As aforementioned, a social license has distinguishable levels. At the same time, the process of moving from one level to another can be thought of as a smooth gradient of continuous relationship improvement through increasing social capital. ▪ Figure 7.9 shows the four levels of social license and the three boundary criteria that separate them. The levels represent how the community treats the company. The boundary criteria depict how the community opines on the company, principally based on the behavior of the company. The levels and boundary criteria are organized in a hierarchy, and it is possible to go both up and down the hierarchy. For example, if a company loses legitimacy, the project will be shut down. If full trust is gained, the community will support and protect the project as its own (Thomson and Boutilier 2011).

Starting from the base, the rejection level of a social license is the worst-case scenario. This is when the community stops progress on the project. Many mineral deposits cannot be exploited because the community does not grant any level of social license to proceed. The withholding/withdrawal level is shown as narrower than the acceptance level above it in order to symbolize the possibility that, globally, more projects are accepted than rejected. Regarding legitimacy boundary criterion, legitimacy can be defined in the context of stakeholders and politics as the approval by the individuals and by relevant organizations of an association's right to exist and to pursue its affairs in its selected way (Knoke 1985). This adequately summarizes the bare minimum of legitimacy even when the company has no social license.

Where legitimacy is established, the community response is that they will listen to the company and consider its proposals. If, by their own standards, they have no reason to doubt the company's credibility, they can allow the project to tentatively proceed. This constitutes the acceptance level of social license. It is a minimal objective for any company. The acceptance level is bounded by the legitimacy criterion and the credibility criterion. This represents how acceptance requires that the company's legitimacy must be firmly established and its credibility should at least not be damaged. Legitimacy can be earned by just listening; credibility requires doing something about what has been heard. Credibility is the basis of confidence, and where an enterprise is considered as credible, it is seen as following through on promises and dealing with everyone. An essential component of credibility comes from openness and transparency in the provision of information and decision-making.

Where a company has established both legitimacy and credibility, a community is likely to grant approval of the project. This means the company has secure access to the resources it needs. The community regards the project favorably and is pleased with it. This level of social license depicts lack of sociopolitical risk. Regarding the full-trust boundary criterion, in management research trust has been shown to be essential in relationships between and within organizations. Trust is especially important where bridging the boundary between businesses and civic sector organizations, which include many community groups. Credibility is a basic level of trust related to honesty and reliability. Communities that have a complete level of trust in a company think that the company will always behave according the interest of the community.

Consequently, both parties come to view project's success as a co-ownership arrangement. The

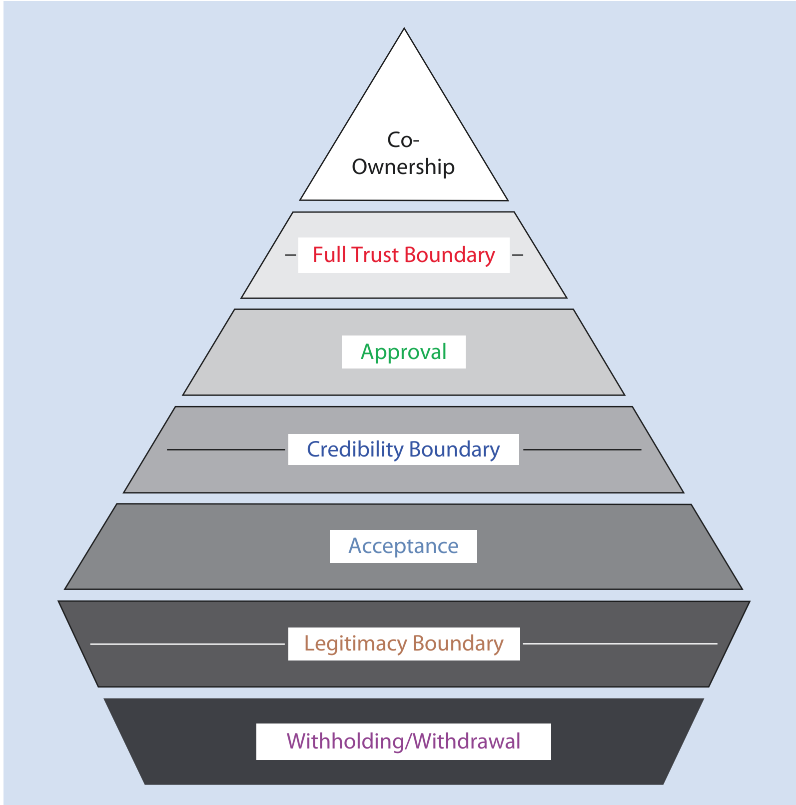

Fig. 7.9 Levels of social license with boundary criteria between them (Thomson and Boutilier 2011)

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limits of the responsibilities of each party are clear, as are ultimate decision criteria. At this co-ownership level of social license, the company becomes an insider in the community social network. Working closely together, the company and community often develop creative solutions to all types of challenges. If outside stakeholders, like the national government or an international non-governmental organization (NGO), move against the interests of the company, the community will mount a campaign in defense of the company. There have been cases where community members have traveled to foreign countries to challenge false information being promoted by NGO critics. Few mining companies have taken their community relations to the co-ownership level. Many have difficulty seeing beyond the immediate transactions to the much greater benefits of establishing strong collaborative relationships. Nonetheless, as awareness of the potential benefits grows, more companies are attempting to win a higher level of social license (Thomson and Boutilier 2011).

7.4 Potential Environmental Impacts and Their Management

7.4.1 Mining Project Phases and Environmental Impacts

Potential environmental issues associated with mining activity such as water use and quality, wastes, hazardous materials, biodiversity, noise and vibrations, and visual impacts (Fig. 7.10) can take place during all phases of the mining cycle, from exploration to closure and post-closure phases. The issue is that mining involves many stages that commonly begin from deposit prospecting and exploration stage, mine development and preparation phase, mine exploration stage, and treatment of the mineral itself with each of these phases involving specific environmental adverse effects.

Therefore, since there are different phases in a mining project, each phase of mining is associated

Fig. 7.10 Visual impact of mining

with different sets of environmental impacts. The adverse effects exploration stage on the natural environment are generally minimum, but it is worth initiating surveys of the present state of the environment before starting any activity that will impact the environment. Thus, activities at ground level commonly need the utilization of boreholes, pitting, and transect lines. For example, the drilling fluids utilized in diamond drilling can get into the water utilized to bring cuttings to surface; this water therefore must be adequately disposed so that it does not pollute the groundwater. The application of support equipment also affects the environment since prospecting vehicles request access tracks. For this reason, application of high-standard environmental management procedures in mineral prospection is critical to assure that such activities are adequately managed with the protection of environmentally delicate zones and community concerns efficiently addressed. Moreover, some countries request independent environmental evaluations for the exploration stage of a mining project since subsequent phases of mining cannot assure if prospection fails to find sufficient amounts of high-grade mineralization.

Once an ore body of sufficient grade has found, then the mining company can start for planning the development of the mine. This stage of the mining project has different components and possible environmental adverse effects. Thus, construction works and the greater amount of traffic originate noise and dust. Changes to the land surface raise the risk of soil erosion (Fig. 7.11) and surface runoff, and the further incremented waterborne loads of solid particles increase the turbidity of water bodies. Drainage water and runoff from a mine operation can also increment loads of metals and nitrogen in water bodies downstream if water is not adequately controlled. All of these changes in water body features as well as in vegetation can affect the conditions for organisms and generate significant changes in species biodiversity. On the other hand, if a mine operation is situated in a distant, undeveloped area, the mining company usually requests to start the operations by clearing land for the construction of staging areas that would house project personnel and equipment. Relative to other phases of activity, the design and construction phase is short. However, this intense

Fig. 7.11 Control of erosion using a mechanical erosionometer (Image courtesy of Freeport-McMoRan)

group of activities and related environmental implications can be destructive if not carefully managed.

After the mining company has developed access roads and staging areas, mining can begin. Surficial mining generally includes the extraction of vegetated zones and commonly also includes the generation of an open-pit that extends below the groundwater table. In this situation, groundwater must be pumped out of the open-pit to enable mining to develop. Thus, mining operations at this stage originate discharges to water that can represent the most important adverse effects to the environment. Outflows of water from the mine site can result in changes such as incremented turbidity, acidification, or salinization in water bodies downstream as well as incremented concentration of metals and nutrients. Regarding the mining projects that exclusively include removing of abandoned waste piles, they prevent the environmental adverse effects of surficial mining but still imply environmental impacts linked to concentration of minerals and/or metals from the waste piles.

Underground mining, although it is a less environmentally harmful method to extract the mineralization in an ore deposit, it is usually more expensive and implies greater safety risks than open-pit mining. Mineralization extraction, utilizing specializing heavy equipment such as loaders, haulers, and dump trucks, which transport the ore to concentration installations utilizing haul roads, can produce noise and generate a specific group of environmental adverse effects such as emissions of dust. Finally, disposal of overburden and waste rock is also a source of different environmental impacts, commonly associated to presence of harmful substances. These materials are often located on-site, either in piles on the surface or as backfilling in pits, or within underground operations. Therefore, environmental assessments for mining projects must carefully evaluate the management possibilities and related adverse effects of overburden disposal.

After the ore has been brought to surface, the process of getting the metal out can also create harmful substances. In beneficiation processes, grinding results in tailings that must be designed to avoid harmful components to reach the environment. For this reason, a treatment procedure for capturing chemically the by-products must be included in the design. Thus, they can be safely disposed individually from the principal portion of the mine tailings. For example, sulfides present in the waste rock have to be kept from creating acid runoff. Meanwhile, different concentration techniques create several types of waste, including waste rock dumps, another type of tailings, heap leach products (e.g., in gold treatment), and dump leach products (e.g., in copper leach beneficiation).

The concentration process uses plenty of water, and this water can contain small concentrations of various organic and inorganic reagents used in the concentration process. How this high volume of water and material is disposed is one of the central questions that will establish whether a suggested mining project is environmentally suitable. The essential long-term objective of tailings management is to avoid the release into the environment of toxic components of the tailings. For instance, wastes including sulfide minerals can produce acidic or neutral runoff with elevated concentrations of metals and sulfates as the sulfide minerals are acidified. If water is not adequately managed, this can decrease water quality in surface water and groundwater bodies. In some cases, water flowing from zones where tailings have been disposed can also include traces of chemicals utilized in mineral processing (e.g., flotation). Another issue of concern is the potential migration of pollutants through rock masses; in fracture rocks, the main portion of the pollutants can migrate through a system of joins, bedding planes, and faults producing contamination of soil and groundwater.

7.4.2 Waste Impacts and Their Management

At large mines, the mass of mineral waste generated can commonly be measured in tens of millions to billions of tons. Similarly, the surface area that must be disturbed for mineral waste disposal is often measured in tens to thousands of hectares and can account for the main disturbance. Waste-related perpetual water management and treatment can account for more than half of the total closure cost at some mines (Borden 2011). Public concerns during project permitting are commonly centered on potential exposure risks and water quality impacts from chemically reactive mineral wastes and can result in project delays and costly permitting requirements. Fortunately, significant advances in mineral waste characterization and management have been made over the past several decades. Proactive mineral waste management can significantly reduce the intensity, footprint, and duration of environmental impacts. Companies that practice proactive management can reduce their financial liability, improve their reputation, and become miners of choice, helping ensure access to new mining opportunities.

The environmental adverse effects of mine wastes are controlled by their type and compositions, which change significantly with the raw material being extracted, type of mineralization, and methods utilized to concentrate the ore. As a result, each mine needs its own waste profiling, prediction, monitoring, control, and treatment. Most mine wastes are environmentally harmless and can be utilized for landform reconstruction, vegetation covers, and road and dam construction. According to Rankin (2011), the main environmental impacts from waste disposal at mine sites can be separated into two categories: (a) the loss of productive land following its conversion to a waste storage area and (b) the introduction of sediment, acidity, and other contaminates into surrounding surface and groundwater from water running over exposed problematic or chemically reactive wastes.

Despite the recycling of many waste types at mines, the bulk of waste generated is still located into storage facilities, and the restoration and long-term control of these installations have become an essential part of modern mine development and closure. Governments and other types of regulators can request any waste storage structures to maintain stable at least for 100 or 200 years, which indicate they must withstand utmost events such as floods and earthquakes. Thus, technological improvements and variation in regulations have produced a meaningful enhancement in waste management procedures

Fig. 7.12 Overburden (top soil) used for landscape contouring (Image courtesy of Daytal Resources Spain S.L.)

over the last 10–20 years. Consequently, mine wastes at contemporary mines are usually better managed than they have been in the past. Moreover, governments of many countries request a specific waste management plan before they will issue mining permits. Guidelines on waste management and mine closure have been created at different levels (international, national, and regional) and offer an advisory framework for best practices in mine waste management. Correct control of tailings and waste rock is based on electing adequate waste storage placements and specific material description, including the precise forecasting of long-term chemical behavior. Structures such as waste and tailing dumps and containment facilities must be designed and treated such that geotechnical risks and environmental adverse effects are adequately evaluated and managed throughout the whole mine cycle.

Types of Waste

Solid wastes can be produced in any phase of the mining activity. Type, quantity, and features of solid mine wastes originated at diverse mines can change based on the raw material being extracted, beneficiation method utilized, and geology at the mine site. In general, the principal types of solid mine wastes are the following:

1. Overburden: cover of soil and rock that is extracted to obtain access to the mineralization at open-pit mines; overburden usually has a low potential for environmental contamination and is commonly utilized for landscape contouring and revegetation during mine closure (Fig. 7.12).
2. Waste rock: material that includes mineralization with low grade considered not interesting to be mined at a profit.
3. Tailings: the fine solid waste generated in the beneficiation process (e.g., froth flotation).

Waste rock is typically a poorly sorted mixture of clay, silt, sand, gravel, and boulder-sized material. Although waste rock can be utilized as backfill in earlier mined areas or translated off-site and utilized at construction projects, most of the waste material originated is placed in piles close to the mine site. The most common disposal method for waste rock is placement within dumps and stockpiles, although in-pit disposal is common in strip mines. Not all mine wastes are defined as harmful wastes since they can even be utilized as feedstock for cement and concrete. Such materials cannot be classified as wastes by definition because they really represent meaningful by-products of mining operations.

On the other hand, tailings are the fine-grained waste that remains after the minerals or elements of economic interest have been removed from the ore (◘ Fig. 6.91). Thus, tailings are composed of the gangue minerals in the ore and residual minerals of economic interest that were not recovered along with process water and any reagents that were added during the milling and concentration processes. Tailings commonly leave the process as slurry formed by 40–70% liquid and 30–60% solids; they are usually disposed of in the form of a water-based slurry in specially engineered repositories (on-site impoundments such as tailing ponds) that are capable of containing the fine-grained and often saturated tailings mass without risk of geotechnical failure.

When developing a waste characterization program, operations must identify and understand the physical and chemical characteristics and hazards of all mineral wastes that will be disturbed, exposed, produced, or imported over the life of the operation. The characterization program must be rigorous enough to provide reliable predictions of the long-term physical and chemical behavior of the waste. Ultimately, the program will be used to select appropriate management strategies that comply with pertinent regulations for each waste type, ensure that all repositories are physically and chemically safe and stable, and allow for successful rehabilitation and closure (Borden 2011). The presence of chemically reactive mineral waste can significantly increase the complexity and cost of waste management. Successful management of chemically reactive mineral waste requires thorough understanding of pertinent regulations, well-designed characterization programs, careful site selection, good facility design, and rigorous ongoing management and monitoring.

Potential Impacts

Mineral waste disposal can be responsible for much of the environmental impact caused by mining. Potential impacts that must be assessed, minimized, and mitigated during mine design, operation, and closure include the following:

1. Direct land disturbance: construction of out-of-pit mineral waste storage facilities will typically require burial of the pre-mining surface, its soils, and ecosystems beneath tens to hundreds of meters of waste.
2. Geotechnical instability: unless properly designed, thick waste piles can be prone to geotechnical instability and failure; instability can range from excessive surface erosion to large deep-seated slope failures; geotechnical risks are generally highest for tailings and other fine-grained waste materials that are saturated when deposited.
3. Erosion and sediment release: erosive wastes are prone to the formation of gullies and other erosion features; the release of sediment at much higher rates than surrounding natural landforms can have negative impacts on down-gradient surface water bodies and aquatic ecosystems.
4. Visual impacts: large-scale mineral waste transport and placement can significantly modify the landscape, creating landforms that are taller than the surrounding topography, truncating valleys and drainage lines, and creating unnatural uniform planar landforms that do not blend in with the surrounding natural topography; visual impacts are likely to be a particular concern near population centers and recreational or protected areas.
5. Direct exposure risks: chemically reactive mineral waste can pose direct chemical exposure risks to people, plants, and animals that live on or near the waste; the pH, salinity, or metal content of the waste can also inhibit vegetation establishment and prevent successful rehabilitation of waste surfaces.

6. Water quality degradation: chemically reactive mineral wastes can degrade the quality of water that runs off or seeps through the waste material; unless properly managed, this can cause degradation of surface and groundwater quality, impacts to aquatic ecosystems, and loss of the beneficial use of water resources far from the point of initial waste placement.
7. Dust release: wind erosion and dust release can degrade air quality because of increases in suspended particulate matter; if the dust is derived from chemically reactive waste, wind transport can disperse potential contaminants over a broad area (Borden 2011).

Site Selection

Waste disposal facilities should be located in areas that minimize environmental impacts and long-term environmental liabilities. The selection process should include a review of site regulatory requirements, baseline conditions and environmental considerations, environmental consequences, and direct surface impacts caused by disposal. In general, the following factors should be considered when selecting locations for waste disposal facilities:

1. Only place waste within legally permitted areas.
2. Where practicable, preferentially place waste within inactive open-pits, underground workings, or existing disturbed areas.
3. Avoid permanent disruption of drainage systems.
4. Tie waste repositories into the surrounding topography to maintain natural, free-draining landforms and to reduce visual impacts.
5. Avoid placement on land with high biodiversity or ecosystem services values.
6. Avoid placement in areas with significant archeological or social value.
7. Avoid placement in close proximity to local communities.
8. Preferentially place chemically reactive wastes in drainage basins that already contain reactive waste (thereby avoiding placement in undisturbed drainages).
9. Limit the footprint of chemically reactive mineral waste to the maximum extent practicable. (j) Avoid placement in areas with poor foundation conditions due to topography, underlying geology, or hydrology.
10. Avoid placement of chemically reactive mineral waste over significant aquifers or groundwater recharge zones.
11. Where the choice is available, such as in some mountainous terrains, preferentially place chemically reactive waste in areas with significantly dryer climates.
12. Balance economic considerations such as haul profiles, potential resource sterilization, and pumping costs with environmental, social, and closure considerations.
13. Avoid placement in or near perennial surface water bodies or in large ephemeral drainage lines where practicable, unless this represents the preferred environmental alternative (Borden 2011).

Waste Rock Dump Management

The overburden and waste rock is commonly arranged in engineered waste rock dumps. Controlling the dumps during the mine life cycle is essential to protect human health, safety, and environment. According to the International Finance Corporation from the World Bank Group (ICF 2007), the main recommendations for management of waste rock dumps are the following: (a) dumps should be planned with appropriate terrace and lift height specifications based on the nature of the material and local geotechnical considerations to minimize erosion and reduce safety risks; (b) management of potentially acid-generating wastes should be correctly undertaken; and (c) potential change of geotechnical properties in dumps due to chemical or biologically catalyzed weathering should be considered, reducing the dumped spoils significantly in grain size and mineralogy; design of new facilities has to provide for such potential deterioration of geotechnical properties with higher factors of safety.

Tailings Management

Tailings management strategies change based on the site constraints and the nature/type of the tailings. Because tailings are formed of fine particles (sand, silt, and clay-sized material), and usually including high water content, they have been especially problematic to manage. Thus, tailings management planning must take into account how tailings will be operated and placed, in addition to continued storage after decommissioning. Strategies should include the site topography,

Fig. 7.13 Dust control (Image courtesy of Anglo American plc.)

downstream receptors, and physical features of tailings (e.g., projected amount, grain-size distribution, density, and water content, among others). Critical considerations for leading practice tailings management are location of the tailings storage facility, geochemical characterization of the tailings, choice of the best tailings disposal technique, tailings delivery, water management, and dust control (Fig. 7.13). Leading practices utilizing paste tailings, good water management, and correct drainage and liners, where adequate, would result in completely consolidated tailings. For this reason, tailings management needs the involvement of competent professionals taking action in accordance with sound geotechnical and hydrological engineering principles.

The selection of appropriate management strategies typically begins by comparing the impacts as predicted by conceptual or numeric models to environmental compliance and performance objectives. If needed, a strategy is selected to reduce the potential impacts and ensure that all compliance and performance objectives will be met during start-up, operation, and closure. The selected strategy does not have to completely prevent any solute or contaminant release but must ensure that release rates meet regulatory requirements and are low enough to be assimilated by the receiving environment without causing harm to people, ecosystems, organisms, or resources (Borden 2011).

Monitoring

After a mineral waste management, strategy is selected, and waste storage facilities have been designed; they must be constructed and successfully managed on an ongoing, long-term basis. Monitoring data should be reviewed regularly, and historical trends should be examined so the longer-term chemical behavior of the mineral waste can be assessed. Time series of monitoring data should be maintained so that long-term changes in water quality, flow rate, or other key parameters can be tracked and significant changes can be identified. Monitoring is required to ensure successful implementation of the mineral waste management plan and to ensure that the strategy is leading to the intended results. Monitoring reports should be prepared annually and reporting should be accessible, easily understood, and transparent to stakeholders. Physical monitoring programs for waste disposal facilities will commonly include at a minimum (a) regular visual inspections of surface structures and facilities such as spillways (Fig. 7.14), piping, dykes, ditches, and other water management systems, (b) regular visual inspections for signs of excessive surface erosion and shallow or deep-seated failure on the outer slopes of waste repositories, and (c) monitoring of water levels and pore pressure within embankments and the waste (Borden 2011).

Spillways consist of primary spillways, which are designed to allow smaller flows out of the

Fig. 7.14 Aerial view of Marlin Spillway (Guatemala) (Image courtesy of Goldcorp Inc.)

impoundment, and emergency spillways, which are designed to pass a peak flow and to ensure the stability of the embankment. Most treatment-type reservoirs are designed with both a primary and an emergency spillway, so that treated water can be released on a regular basis while protecting the embankment. Programs to monitor the geochemical behavior of waste disposal facilities will include (a) periodic sampling of runoff water (Fig. 7.15) and water discharging from the facility's toe in order to monitor flow volumes, solute concentrations, and the solute mass that is being released from the waste, (b) periodic sampling of downgradient monitoring wells and surface water bodies to ensure that seepage from the waste is not adversely impacting receiving environment water quality, and (c) periodic assessment of revegetation success such as total cover, species composition, and plant health.

7.4.3 Water Management

Water is utilized in mining in a wide rank of operations such as beneficiation processes, dust elimination, slurry transportation, and employee requests. The water cycle of a mine is interlocked with the global hydrologic water cycle of a watershed (Fig. 7.16). The mining industry has made significant advances in the last decades in developing close-circuit considerations that maximize water preservation. In parallel, operations are commonly situated in zones where there are not only important municipal, agricultural, and industrial needs but also diverse opinions about the role of water. Moreover, the local environments of mine operations rank from very low to the highest rainfall zones in the world. Independently, liable management of water by mining enterprises is an essential component to assure that their contribu-

Fig. 7.15 Sampling of runoff water (Image courtesy of Glencore)

tion to sustainable development is clearly positive over the long term. In this sense, it is necessary to bear in mind that managing water is one of the most important environmental activities at operating mines. Moreover, water control is a collaboratively liability across the operations although collective management does not signify that liability for certain zones cannot be allocated. Global responsibility is best controlled if the operation has someone in charge of committees and processes (Commonwealth of Australia 2008).

The main sources of water on a mine site are from precipitation, dewatering of open-pits or underground workings, and pumping and removal of groundwater specifically around open-pit operations. Precipitation that falls on the mine site must be collected and cleaned prior it can be discharged to the environment. Water from the dewatered open-pit works or underground operations requests to be treated before being released. Regarding management of water utilized on the mine site (e.g., processing the ore or watering of roadways to keep dust down during dry periods), the water is usually recycled so only a small amount of new water is needed every day. For example, chemicals utilized in the concentration process are generally removed or diluted before tailings are sent to the tailings storage facility (Stevens 2010).

Planned water releases from mines into the environment are commonly closely controlled to assure observance with legislation and to diminish adverse effect to receiving waters. Release of process water is systematically managed and must acquire some quality standards and requests in terms of temperature, pH, and conductivity. Other discharges are produced due to normal run-off, utmost storm events, and release from surplus dewatering where water can be contained and discharged appropriately (ICMM 2012b). Surrounding surface and groundwater quality is controlled, and numerous treatment procedures can be utilized to assure mine water complies legislation standards previous to be released.

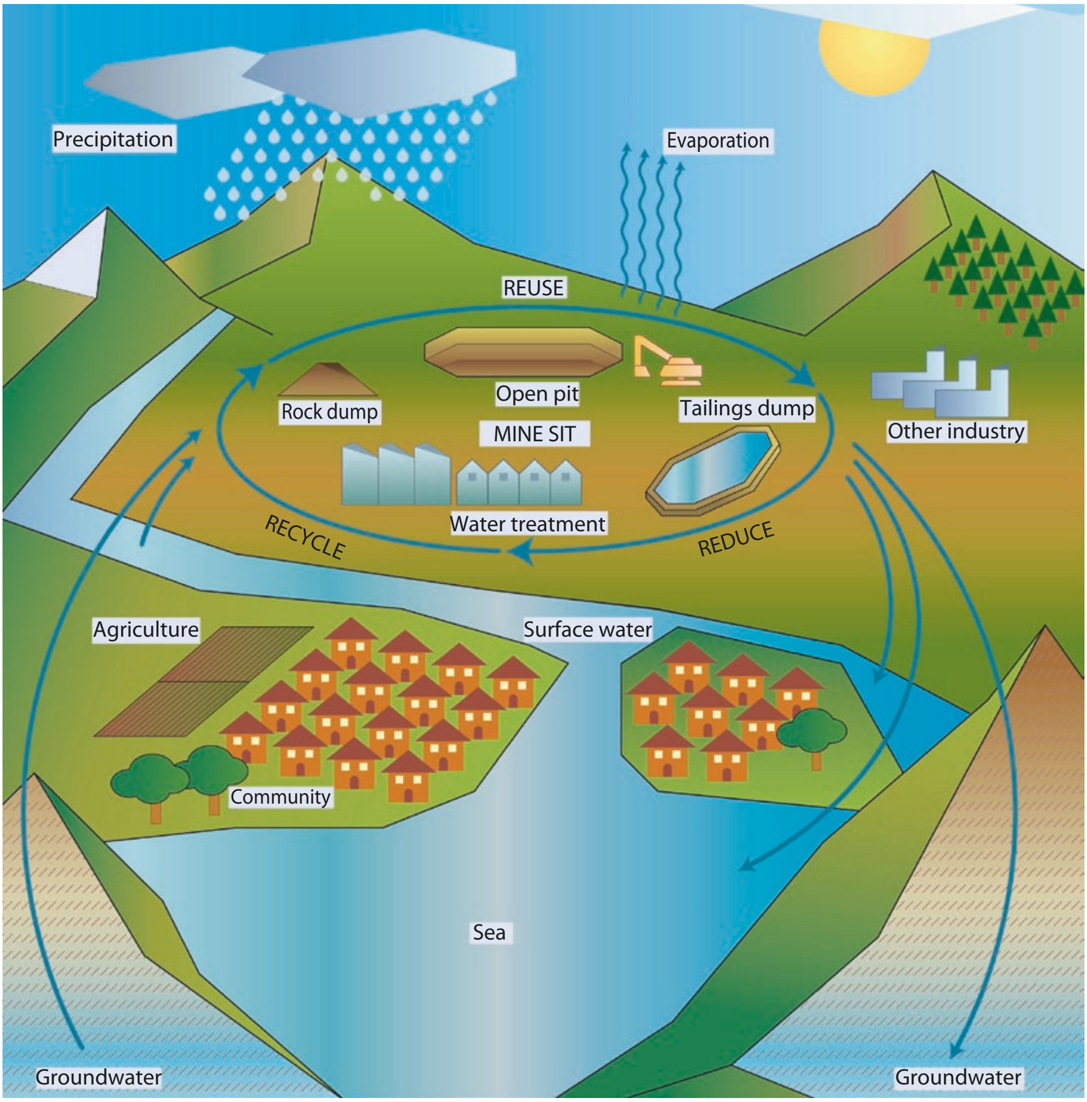

Fig. 7.16 Flows of water to and from a mine site (ICMM 2012b)

Potential Impacts

One of the principal issues that can be linked to mining operations is the release of contaminants to surface water since many activities of a mining operation can generate toxic and nontoxic components to surface water. Thus, open-pit, tailings pond, mineralization stockpile, waste rock dump, and heap and dump leach pile are all examples of possible important sources of toxic pollutants. The mobility of the contaminants from these origins is increased by exposure to rainfall. Seepage from tailing dump zones and groundwater generating from open-pit mines are another examples by which heavy metals can be mobilized and sometimes released to surface waters. Discharge of contaminants to surface waters can also take place indirectly via groundwater that has hydrological connecting to surface water. Some adverse effects to surface waters include the buildup of sediments that can be polluted with heavy metals, short- and long-term decreases in pH level (especially for lakes and reservoirs), degradation of aquatic habitat, and contamination of drinking water and other human health issues.

The impacts of the mining operations to the surrounding water resources and water-dependent ecosystems are by water withdrawal and dewatering impacts and the discharge of contaminated water. Surface and groundwater withdrawals to dewater the ore body or to supply operations can lower surrounding groundwater water tables, causing seeps, springs, and wells to dry up, harming groundwater-dependent vegetation and ecosystems, and reducing in-stream flow. If necessary, dewatering impacts must be predicted, monitored, and mitigated. Mitigation strategies can include (a) improving water efficiency through process and management improvements so that less water needs to be withdrawn, (b) intentional surface water discharge at key locations to maintain in-stream flow, (c) providing alternative water resources for impacted communities, (d) intentional recharge of groundwater to minimize drawdown impacts, and (e) construction of slurry walls and other subsurface flow barriers to minimize hydrogeologic connections (Borden 2011).

To reduce these issues, an adequate water management plan (WMP) is essential to leading practice water management. Its size and complexity is varied, depending on the nature of the mine, hydrology, and cultural and environmental sensibility of the surrounding area. The WMP defines all water management problems linked to development, operation, and decommissioning a project, integrating also water quantity and quality. The WMP records particular site water goals against which performance can be assessed; quantitative aims are better for an efficient auditing of performance. The WMP also includes any request for internal and external reporting of water performance. Finally, the WMP is dynamic and should be systematically updated and reviewed (Commonwealth of Australia 2008).

Practices for Water Management

Water treatment before discharge can be costly. At large mines with significant acid rock drainage flows, cumulative treatment costs can be measured in the tens to hundreds of millions of dollars. Implementation of internal proactive management strategies that reduce the volume of water that must be treated and/or reduce the solute load in the water can be cost-effective as well as ultimately more protective of the environment (Borden 2011).

Broad water management strategies and control techniques to decrease the potential for water pollution and diminish the amount of water needing treatment include the following: (a) water diversion: capture and diversion of clean surface and groundwater flows up-gradient of the operation can limit the volume of water that can be contaminated by contact with the operational footprint; (b) improved water use efficiency: improvements in water use efficiency can also reduce the volume of water that must be imported into the operation; (c) reagent management: process water quality can be improved by the efficient use of reagents and/or replacement of hazardous reagents with less hazardous but equally effective substitutes; (d) on-site evaporation: evaporative losses within the footprint of the operation will reduce the volume of water that must be discharged; and (e) installing liners and covers on waste rock and ore piles to reduce the potential for contact with precipitation and contamination of groundwater (Lottermoser 2012). Different combinations of strategies can be applied, and the selection of strategies is site-specific. For instance, the interception and diversion of surface water is a more prominent concern in environments with high rates of precipitation, whereas more emphasis is placed on water recycling in arid regions with little water availability.

For water treatment, there are numerous treatment methods forthcoming to clean contaminated water, being these technologies classified as active or passive. Active treatment methods need input of energy and chemicals, while passive technologies use only natural procedures such as gravity, microorganisms, and/or plants in a system, anyone of which requests uncommon but regular maintenance (Younger et al. 2002). In general, the treatment methodology utilized at a mine is based on how contaminated the water is, what chemicals products require to be extracted, how much water needs processing, and the needed release water quality standards. Active water treatments are the most usual manner of water processing at working mines (Fig. 7.17). Thus, mine waters are almost always acidic and need the addition of lime or caustic soda to increase the pH. Once pH has been incremented, dissolved metals can precipitate out of solution and sink to the bottom of settling or sedimentation ponds where they can be extracted. Chemicals called coagulants or flocculants can be added with the aim of converting smaller particles into larger

Fig. 7.17 Active water treatment plant near a coal mine

clumps that settle out of the water more quickly (Brown 2002). Regarding the passive water treatments, they are commonly combined with water monitoring programs and advantage of natural physical, chemical, and biological processes that remove water contaminates without additional physical or chemical inputs. Examples of these procedures are bacteria-controlled metal precipitation, contamination uptake by plants, and filtration through soil and sediments.

Acid Mine Drainage

The term acid mine drainage (AMD) or acid rock drainage (ARD) is used to describe the drainage resulting from the natural oxidation of sulfide minerals that occur in mine rock or waste exposed to air and water. It is important to remember that it is a natural process, not something specifically generated by mining (Box 7.4: Chemistry of Acid Mine Drainage). AMD can incorporate acidity and dissolved metals into water, which is usually very harmful to aquatic life.

Acid mine drainage is responsible for problems of water pollution in major coal and metal mining areas around the world.

Once AMD develops, it can be hard to control and stop. If acid mine drainage is not controlled, it can pose a serious threat to the environment because acid generation can lead to elevated levels of heavy metals and sulfate in the water, which obviously have a detrimental effect on its quality. Stopping AMD development can be very complex since it is a process that, when left unrestrained, will advance, and can accelerate, until some of the chemical components (sulfide minerals, oxygen, and water) are depleted or removed from reaction (Verbug 2011). Thus, the development of ARD is time dependent and sometimes can evolve over a period of decades or even centuries after mining has ceased.

Managing acid rock drainage is a preoccupation at mine workings and after mine closure. Furthermore, AMD is also a major concern for mining companies since nowadays mining operations tend to increment the quantity of rocks

Box 7.4

Chemistry of Acid Mine Drainage

Sulfide minerals in ore deposits are former under reducing conditions in the absence of oxygen. When exposed to atmospheric oxygen or oxygenated waters due to mining, mineral processing, excavation, or other earthmoving processes, sulfide minerals can become unstable and oxidize. Thus, the generation of acid (H^+) occurs typically where iron sulfide minerals are exposed to both oxygen (from air) and water. This process can occur both abiotically or biotically (e.g., microorganisms). In the latter case, bacteria such as *Acidithiobacillus ferrooxidans*, which derive their metabolic energy from oxidizing ferrous to ferric ion, can accelerate the oxidation reaction rate by many orders of magnitude relative to abiotic rates. Sulfide oxidation produces sulfuric acid and an orange precipitate, ferric hydroxide ($Fe(OH)_3$), as summarized in Reaction 1.

Reaction 1

$$FeS_2 + 3.75O_2 + 3.5\,H_2O \Leftrightarrow Fe(OH)_{3(s)} + 2SO_4^{\,2-} + 4\,H^+$$

$$\text{Iron sulfide} + \text{Oxygen} + \text{Water} \Leftrightarrow \text{Ferric hydroxide} + \text{Sulfate} + \text{Acid (orange precipitate)}$$

There are two key processes involved in the generation of acid (H^+) from iron sulfide: (a) oxidation of sulfide ($S_2^{\,2-}$) to sulfate ($SO_4^{\,2-}$) and (b) oxidation of ferrous iron (Fe^{2+}) to ferric iron (Fe^{3+}) and subsequent precipitation of ferric hydroxide. These can be represented in the following three reactions (these reactions, when combined, are equivalent to Reaction 1):

Reaction 1a

$$FeS_{2(s)} + 3\,O_{2(g)} + H_2O_{(aq)} \Leftrightarrow Fe^{2+}{}_{(aq)} + 2\,SO_4^{\,2-}{}_{(aq)} + 2\,H^+{}_{(aq)}$$

$$\text{Iron sulfide} + \text{Oxygen} + \text{Water} \Leftrightarrow \text{Ferrous iron} + \text{Sulfate} + \text{Acid}$$

Reaction 1b

$$Fe^{2+}{}_{(aq)} + O_{2(g)} + H^+{}_{(aq)} \Leftrightarrow Fe3^+{}_{(aq)} + H_2O_{(aq)}$$

$$\text{Ferrous iron} + \text{Oxygen} + \text{Acid} \Leftrightarrow \text{Ferric iron} + \text{Water}$$

Reaction 1c

$$Fe3^+{}_{(aq)} + H_2O_{(aq)} \Leftrightarrow Fe(OH)_{3(s)} + 2\,H^+{}_{(aq)}$$

$$\text{Ferric iron} + \text{Water} \Leftrightarrow \text{Ferric hydroxide} + \text{Acid (orange precipitate)}$$

Once sulfides have been oxidized to sulfates, it is difficult to avoid oxidation of aqueous ferrous iron to ferric iron and subsequent iron hydroxide precipitation. This precipitation stage is acid-generating (Reaction 1c).

The interaction between dissolved ferric iron (Fe^{3+}) and fresh iron sulfide minerals can also lead to significant acceleration of the acid generation process, as represented in the following reaction.

Reaction 2

$$FeS_{2(s)} + 14\,Fe^{3+}{}_{(aq)} + 8\,HO_{(aq)} \Leftrightarrow 15\,Fe^{2+} + 2\,SO_4^{\,2} + 16\,H^+$$

$$\text{Pyrite} + \text{Ferric iron} + \text{Water} \Leftrightarrow \text{Ferrous iron} + \text{Sulfate} + \text{Acid}$$

Under the majority of circumstances, atmospheric oxygen acts as the oxidant. However, aqueous ferric iron can oxidize pyrite as well. This reaction is considerably faster (two to three orders of magnitude) than the reaction with oxygen and generates substantially more acidity per mole of pyrite oxidized. However, this reaction is limited to conditions in which significant amounts of dissolved ferric iron occur (i.e., acidic conditions – pH 4.5 and lower). Oxidation of ferrous iron by oxygen is required to generate and replenish ferric iron, and acidic conditions are required for the latter to remain in solution and participate in the ARD production process.

exposed to air and water, and many metal mineralization and coal deposits are rich in sulfide minerals. In this sense, mining companies are upwardly requested to assess the ARD potential at future mine operations and propose comprehensive planning to prevent or avoid ARD at all stages of mining cycle as part of the environmental impact assessment (EIA) procedure.

AMD Formation

The process of sulfide oxidation and development of AMD is not easy to understand and includes numerous chemical and biological processes that can change importantly in accordance with environmental, geological, and climate characteristics (Nordstrom and Alpers 1999). In unaffected natural situations, acid development is a moderately

Fig. 7.18 Acid mine waters (*red waters*) formed in an old metallic mining area (Image courtesy of María de los Ángeles Bustillo)

slow process considering geological time. But mine works and concentration of mineralization and materials incorporating metal sulfides hugely increase the acid-generating process because it rapidly exposes those substances to oxidizing conditions.

The most common acid-generating sulfide minerals are pyrite (FeS_2), pyrrhotite (FeS), marcasite (FeS_2), chalcopyrite ($CuFeS_2$), and arsenopyrite ($FeAsS$). It is clear that not all sulfide minerals originate acidity when being oxidized since sphalerite and galena tend not to generate acidity when oxygen is the oxidant. But it is also very evident that all sulfide minerals are capable of generating acidity if aqueous ferric iron is the oxidant. In this sense, the presence of microorganisms such as *Thiobacillus ferrooxidans* may accelerate the reaction by its enhancement of the rate of reduced sulfur oxidation. If conditions are not favorable, the bacterial influence on acid generation will be minimal. As aforementioned, ARD is a natural process and has been produced in a natural manner over millions of years. Thus, the names of rivers such as the Rio Tinto in Spain, the Norwegian Raubekken, and the Iron Creek in Colorado reflect the historical nature of AMD.

In general, ARD can show the following chemical features: (a) low pH ranging from 1.5 to 4, (b) high-soluble metal concentrations, (c) high (sulfate) salinity, (d) low quantities of dissolved oxygen, and (e) low turbidity or total suspended solids. On the other hand, according to Commonwealth of Australia (2007a), essential indicators of AMD presence include «red colored (Fig. 7.18) or unnaturally clear water, orange-brown iron oxide precipitates in drainage lines, death of fish or other aquatic organisms, precipitate formation on mixing of AMD and background (receiving) water, poor productivity of revegetated areas (e.g., waste rock pile covers), vegetation dieback (e.g., bare areas), and corrosion of concrete or steel structures.» For instance, the most common and very noticeable manifestation of ARD from a dump is the reddish brown staining associated with the effluent and which consists of precipitates of principally ferric salts. These salts are a source of turbidity, but they do not represent an environmental issue.

Locations susceptible to develop acid rock drainage since sulfides can be routinely exposed to air and water are waste rock pile, ore stockpile, tailings storage facility (Fig. 7.19), underground mine, and heap and dump leach pile. However, ARD will not occur if the sulfide minerals are nonreactive or if the rock contains sufficient alkaline material to neutralize the acidity. In the latter instance, pH value of the water may be near neutral, but it may carry elevated salt loads, especially of calcium sulfate. In other words, the acid-generating capability of sulfide minerals is countered by acid-neutralizing minerals. Most carbonate minerals are capable of dissolving

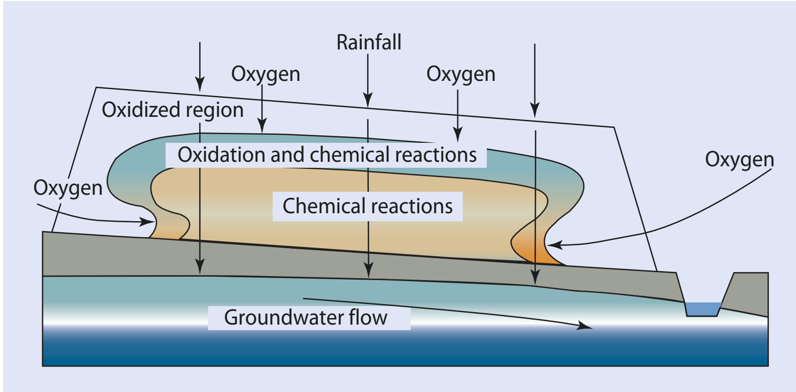

Fig. 7.19 Schematic representation of AMD generation and pollutant migration from a waste rock (Ritchie 1994)

quickly, making them efficient acid consumers. In some cases, calcium-magnesium silicates can buffer mine effluents at neutral pH. In cases of near neutral pH, the levels of major ions such as calcium, magnesium, and sulfate are unacceptably high from an environmental viewpoint.

However, the neutralization of acid generally increases the amount of toxic metal concentrations in the resulting drainage. While increases in pH are desirable, the consequent increase in toxic metal concentrations is not. At most mining sites, there is not sufficient natural neutralizing materials to increase the pH of drainage to near neutral values. Thus, acid mine drainage characterized by low pH and high toxic metal concentrations is the most usual manner of AMD undergone at mine operations (Commonwealth of Australia 2007a).

Lottermoser (2012) affirms that the rate of AMD generation depends on a number of factors such as:

1. Surface area of sulfide minerals exposed: increasing the surface area to air and water increases sulfide oxidation and AMD formation.
2. Type of minerals present: not all sulfide minerals are oxidized at the same rate, and neutralization by other minerals present can occur, which would slow the production of AMD.
3. Amount of oxygen present: sulfide minerals oxidize more quickly where there is more oxygen available; as a result, AMD formation rates are higher where the sulfides are exposed to air than where they are buried under soil or water.
4. Amount of water available: cycles of wetting and drying accelerate AMD formation by dissolving and removing oxidation products, leaving a fresh mineral surface for oxidation; in addition, greater volumes of AMD are often produced in wetter areas where there is more water available for reaction.
5. Temperature: pyrite oxidation occurs most quickly at a temperature around 30 °C.
6. Microorganisms present: some microorganisms are able to accelerate AMD production.

Important progresses in the knowledge of AMD have been carried out in the last decades with advancements also in mine water quality forecast and utilization of preventing methods. However, mine water quality forecast can be complex due to the broad range of the chemical reactions included and potentially very long periods over which these reactions develop. In spite of the uncertainty, quantitative forecasting generated by utilizing realistic scenarios has demonstrated to be of significant value for identifying AMD management options and evaluation of potential environmental adverse effects. Thus, prediction of mine water quality generally is based on one of more of the following procedures: (a) test leachability of waste materials in the laboratory; (b) test leachability of waste materials under field conditions; (c) geological, hydrological, chemical, and mineralogical characterization of waste materials; (d) geochemical and other modeling (INAP 2009).

AMD Impacts

AMD is one of the most sensible and visible environmental problems facing the mining industry because it is often the method of transport for a rank of contaminants, which can affect on-site and off-site water resources, and associated

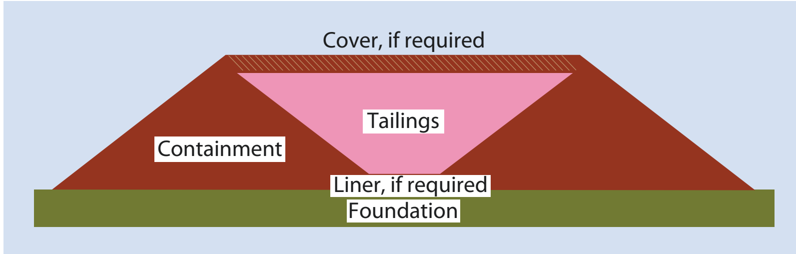

Fig. 7.20 Encapsulation of reactive tailings to minimize sulfide reaction rates (Commonwealth of Australia 2007a)

human and ecological receptors. The impacts of AMD on near and distant water resources and receptors can also be long term and persist after mine closure. Therefore, AMD prevention, mitigation, and treatment are important components of overall mine water management over the entire life of a mining operation (Verbug 2011).

The environmental adverse effects of AMD depend on the size and sensibility of the water body concerned and the quantity of neutralization and dilution. For instance, the same amount of ARD would have greater adverse effect on the water quality of a small lake than it would have in the ocean, as the ocean has a higher dilution capability and salt water has stronger acid-buffering capacity than freshwater. The dissolved metals associated with AMD are commonly more toxic to fish and aquatic organisms than is the acidity.

AMD Prediction and Mitigation

One of the most important studies that must be carried out in a mining environmental assessment is to evaluate the potential developing of AMD processes. Thus, an accurate prediction of acid mine drainage is required in order to determine how to bring it under control. The objective of AMD control is to satisfy environmental requirements using the most cost-effective techniques. The options available for the control of contaminated drainage are greater at proposed rather than at existing operations, as control measures at working mines are limited by site-specific and waste disposal conditions. The length of time over which the control measurements are requested to be efficient is a factor which requires to be determined previous to the design of a system to control ARD. The prediction of the potential for acid generation involves the collection of available data and the performance of static and kinetic tests. Both tests provide data that can be used in different models to predict the effect of acid generation and control processes.

On the other hand, a risk-based planning and design forms the basis for prevention and mitigation of AMD. The main goal of the risk-based procedure is to quantify the long-term adverse effects of alternatives and to utilize this knowledge to elect the option that has the most convenient combination of attributes. Including the prevention and mitigation effort into the mine operation is an essential factor for successful AMD management (INAP 2009). Therefore, the most cost-effective and low-risk AMD management approach is to prevent AMD development through prediction and mine planning. Prevention of AMD must begin at exploration stage and continue throughout all the mine cycle, being the keystone to avoid costly mitigation. The first aim is to use techniques that minimize sulfide reactions, metal leaching, and further migration of weathering products originated from sulfide oxidation (Fig. 7.20).

Where sulfide mineral extraction is inevitable, a number of AMD prevention strategies have been carried out such as locating waste rock underwater, flooding and sealing underground mines, mixing acid-producing materials with acid-buffering materials, covering waste rock, and treating of sulfide wastes chemically. In the latter, organic chemicals have been used to sulfide wastes with the aim to decrease the rate of AMD; however, there is concern that some of these chemicals can reduce beneficial microorganisms in the environment, thus being pollutants themselves (Price and Errington 1998; Johnson and Hallberg 2005). In this sense, it is far more efficient and usually far less costly in the long term to control acid mine drainage during its early phases.

Prevention and control of AMD is undertaken using primary, secondary, and tertiary control mechanisms. Primary control measures are those that prevent AMD from developing. They commonly include segregating potentially acid origination waste rock or tailings from non-potentially acid-generating rocks and locating it underwater or underground. Secondary control measures are those that do not stop AMD from developing but prevent or decrease the migration of AMD waters. In some cases, secondary control measures can be applied until primary methods can be developed. Finally, tertiary control measures involve the long-term collection and treatment of AMD waters to decrease acidity and remove dissolved metals. This is an unacceptable solution for a new mine and is only utilized for old or closed mines that did not consider AMD-mitigation at the time of the operation or were not planned effectively. This type of measures is costly and can go on indefinitely (Stevens 2010). Obviously, where the entire prevention of AMD process is ineffective, acid mine waters must be trapped and treated utilizing a number of water treatment processes.

AMD Management

The management of AMD and the evaluation of its efficiency are generally considered within the site environmental management planning or in a site-specific ARD management report. The requirement for a formal AMD management planning is commonly motivated by the results obtained in AMD characterization and prediction reports or the results of site monitoring. It is important to note that the development, evaluation, and constant enhancement of an AMD management planning are a continuum throughout the life of a mine (INAP 2009). The principal objective of the management planning must be to minimize or, wherever possible, remove the footprint of potentially acid-forming materials. The AMD management planning detects materials that need special management. To be efficient, the AMD management planning must be completely integrated with the mine plan. Finally, accountability to implement the management planning is verified to assure that those responsible are meeting the requests stipulated in the plan (Verbug 2011).

Strategies to manage ARD can be classified in three main types: minimization of oxidation and transport of oxidation materials, control to decrease contaminants, and/or active or passive treatment to enable water reuse. From a sustainability point of view, minimization is favored to control, and the latter is preferred over treatment. Election of the best minimization and control management strategies depend on climate, topography, mining method, material type, soil/rock types, mineralogy, and available neutralization resources as well as interrelationships between these. The control of acid mine drainage can request different approaches, depending on the severity of potential acid generation, the longevity of the source of exposure, and the sensitivity of the receiving waters. Regarding treatment of waters, there are two phases involved with the design of a system for the treatment of ACM, one during mine operation and another after closure. In any case, conventional active treatment of mine waters needs the installation of a treatment plant, continuous operation, and maintenance, which result in high capital and operational costs. Alternatively, passive methods try to minimize the inputs of energy, materials, and manpower and so decrease operational costs.

7.4.4 Hazardous Materials Management

Hazardous substances are materials that can have adverse effect on human health due to their physical, chemical, and biological properties. Common hazardous industrial wastes include solvents, used oil, oily debris, spent reagents, coolants, greases, batteries, and used paints. Usually these wastes are sent to off-site recycling, treatment, or disposal facilities. Taken into account the previous definition, some materials found in mining and processing operations can be hazardous to human health and the environment. Naturally occurring materials that can be classed as hazardous when exposed by mining include asbestiform minerals, silica, metals, and radioactive minerals. Chemical substances utilized in mining (e.g., explosives and flotation reagents) are hazardous as well. Wastes and by-products of mining operations, such as dusts and acid-generating sulfides, can also be hazardous. The actual risks posed by the handling of these materials depend on their innate hazards, volumes that are present, potential receiving environments, and transport pathways that could connect the point of release with potential receptors (Borden 2011).

Asbestiform Minerals

Where asbestiform minerals are found (◘ Table 7.1), airborne asbestos fibers can be present as minor/trace contaminants in the dust produced during blasting, crushing, and further handling and processing. Concern about the effect on

Table 7.1 Asbestiform minerals

Asbestiform variety	Chemical composition
Serpentine group	
Chrysotile (white asbestos)	$Mg_3(Si_2O_5)(OH)_4$
Amphibole group	
Crocidolite (blue asbestos)	$Na_2Fe_3Fe_2(Si_8O_{22})(OH,F)_2$
Amosite (grunerite) (brown asbestos)	$(Mg,Fe)_7(Si_8O_{22})(OH)_2$
Anthophyllite	$(Mg,Fe)_7(Si_8O_{22})(OH,F)_2$
Tremolite	$Ca_2Mg_5(Si_8O_{22})(OH,F)_2$
Actinolite	$Ca_2(Mg,Fe)_5(Si_8O_{22})(OH,F)_2$

health from long-term, low-level exposure to asbestos needs that adequate procedures be used wherever asbestiform minerals are encountered. The aim is to assure that exposure is as low as is acceptably suitable. To minimize the potential risks from asbestiform material, a competent person (such as a geologist or mineralogist) should analyze exposed rock during the initial studies into the ore body to determine the presence and extent of asbestos. An asbestos management planning can then be prepared for the risk zones determined through asbestos exposure monitoring (Commonwealth of Australia 2009a).

Silica Minerals

Silica minerals make up the matrix or occur linked to the targeted mineral in mineralization. They include quartz, which is a common gangue component of the ores and a very common rock-forming mineral in most igneous and metamorphic rocks. The same natural process that results in sulfide ore bodies often concentrates silica minerals. They are stable until ground or blasted into a dust. Crystalline silica dust is termed as a Group 1 carcinogen by the International Agency for Research on Cancer, being the dust irritant to lungs.

Metals

Metal concentrations increment in waters at low pH values. Thus, dissolved metals can move from mining facilities to local ground and surface water. Once released, metals will continue in the environment. While AMD can improve pollution mobility by fostering leaching from wastes and mine infrastructures, liberations can also take place under neutral pH values. First sources of metals in solution from mining works cover underground and surface mine operations, overburden and waste rock piles, tailings piles, discharges from beneficiation processes, leach piles and processing facilities, chemical disposed areas, and restoration activities. Thus, depending on the local geology, the mineralization and the waste rock and overburden can contain trace levels of numerous elements such as arsenic, cadmium, chromium, copper, iron, lead, mercury, nickel, silver, zinc, and many others as well as naturally occurring radioactive materials.

The presence of certain metals, their liberation potential, and the linked risks are very dependent on facility-specific features such as design and operation of mining and mineral processing operations, waste controlling methods, treatment/mitigation measures, environmental characteristics (e.g., climate, hydrogeology, or mineralization composition, and geochemistry), and nature of and vicinity to human and environmental receptor. To prevent the unintended presence of these metals, dissolved metal concentrations in water can be decreased through physical removing (sorption, precipitation, and biological uptake) (Smith 2007).

Radioactive Minerals

All minerals contain radionuclides that are members of the naturally occurring radioactive decay chains. The impact of these radionuclides needs to be considered in certain types of mining. Radionuclides such as uranium, thorium, radium, and radon can pose exposure risks because of toxicity and/or radiological hazards. Igneous and certain metamorphic rocks are more radioactive than most sedimentary rocks. The release of uranium and its daughter products are an issue at uranium mines. However, radionuclides can also pose hazards at heavy mineral sands, rock phosphate, coal, rare earth ore bodies, and ore bodies associated with granitic rocks. Exposure to elevated radioactivity levels can also occur during rare earth production, bauxite production, and oil and gas extraction, among many examples. The level of possible hazard from radioactive minerals relies on the type of radioactivity and its half-life period.

One of the major radiological risks in mining is associated with inhalation of radon (a radioactive gas with a short half-life) and its short-lived radioactive decay products. Radon is produced by the radioactive decay of radium. Radon exposure can be a particular concern at some underground uranium mines and needs to be carefully considered. The control of radon at underground uranium mines should commence with the process of selecting the mining method, controlling water inflows, and designing a flexible ventilation system. In addition, each mine has to establish safety operating procedures specific for each operating mine. The latter is extremely important, as even the best ventilation system can malfunction because of a power outage, human error, or other unforeseen circumstances. When designing ventilation systems for underground uranium mines, deposits can be divided into two groups: low-grade deposits, usually ranging from 0.1% to 2% U_3O_8, and high-grade deposits, where the grade can exceed 20% U_3O_8 (Apel and Hashisho 2011). In the case of high-grade deposits, the radon emanation rate from the ore would make it practically impossible to dilute the radon daughters using flush-through ventilation, and in these cases, the ore is mined using remote mining methods (e.g., raise boring or mining using water jets).

Regarding management of hazardous materials, it starts with their adequate identification during pre-feasibility studies followed by characterization of the mineralization, waste rock, overburden, mine process residues, and natural soil under the mine installation. If harmful naturally occurring minerals are found during mining, activities should finish until hazard has been adequately assessed and corrective actions have been organized.

Other Hazardous Substances

Other hazardous substances utilized and produced on mine and mineral processing sites can include the following:

1. Acids (sulfuric, hydrochloric): contact with strong acid liquids or fumes is a human health hazard and can also cause structural damage in a facility.
2. Sodium cyanide for gold recovery in large operations: the risk of cyanide poisoning arises from ingestion and exposure to workplace vapors and solutions.
3. Mercury for gold recovery in small/artisanal operations.
4. Metals as ions or complexes from Cu, Pb, Zn, Ni, Fe, As, Hg, and Cd sludges or solutions.
5. Thiosulfates and polythionates, also resulting from acid mine water or processing solutions.
6. Process reagents (acids, alkalis, frothers and collectors, modifiers, flocculants, and coagulants) that contain aluminum and iron salts and organic polymers.
7. Nitrogen compounds from blasting materials: best practice consists of adequate ventilation and monitoring of the workplace atmosphere rather than the use of personal protective equipment.
8. Oil and fuel used for engines, power plants, and lubrication.
9. Solvents used in extraction plants (Commonwealth of Australia 2009a).

7.4.5 Mining and Biodiversity

The protection and conservation of biodiversity is crucial to sustainable development. The United Nations Convention on Biological Diversity defines biodiversity as «the variability among living organisms from all sources including inter alia, terrestrial, marine and other aquatic ecosystems and the ecological complexes of which they are part; this includes diversity within species, between species and of ecosystems.» Thus, biodiversity is commonly defined at three separate levels: genetic diversity, species diversity, and ecosystem diversity. It is crucial that all partners constituting the mining industry admit that biodiversity has significant environmental, social, and cultural value.

Mining can affect biodiversity throughout the life cycle of a project, both directly and indirectly. Direct or early adverse effects from mining can be produced from any activity that includes land clearance (e.g., access road or tailings dumps construction) or direct discharging to water bodies (e.g., riverine tailings disposal) or the air (e.g., dusts or smelter emissions). This type of adverse effects is commonly easy to identify. Indirect or secondary impacts can be generated from social or environmental variations produced by mining operations and are usually very difficult to identify quickly (ICMM 2006). At the same time, the mining industry has offered considerable effort to the knowledge of biodiversity management. It is essential that the mining industry admits that it not only has a liability to control its impacts on biodiversity but also has the possibility to carry out a decisive contribution to biodiversity conservation through the production of knowledge and the implementation of actions in cooperation with others partners.

Since mining will often have unavoidable negative impacts on biodiversity, it is possible to offset impacts by creating benefits elsewhere to produce an overall conservation outcome that maintains the biodiversity assets of a region. Such offsets can be direct through acquiring comparable land and managing it for biodiversity conservation. This process is sometimes referred to as biobanking (CSIRO 2014). Another form of a direct offset is through funding the implementation of regional conservation plans. Biodiversity offsets can also be indirect such as by conducting relevant research for improved conservation management or through education and training that increases regional capacity for biodiversity management.

7

The risks and impacts to business of the failure to correctly manage biodiversity problems can include (a) increased regulation and liability to prosecution; (b) increased rehabilitation, remediation, and closure costs; (c) social risks and pressure from surrounding communities, civil society, and stakeholders; (d) restricted access to raw materials, including access to land, both at the initial stages of project development and for ongoing exploration to extend the lifetime of existing projects; and (e) restricted access to finance and insurance (Commonwealth of Australia 2007b).

Thus, it is very interesting for mining companies to address biodiversity for many different sound business reasons. Consequently, most mining companies have established an ever more complex perspective to managing biodiversity as part of their compromises to achieve and maintain a social license to operate.

Taking responsible decisions regarding to biodiversity management is upwardly considered as very important with respect to (a) reputation, which links to the license to operate, an intangible but significant benefit to business; it can profoundly influence the perceptions of communities, NGOs, and other stakeholders of existing or proposed mining operations; and (b) access to capital, particularly where project finance is to be obtained from one of the investment banks that are signatories to the Equator Principles, which apply the Biodiversity Performance Standard of the International Finance Corporation (IFC) to all investments in excess of US $10 million, recognizing that strengthened commitments to biodiversity assessment and management are likely to be adopted (ICMM 2006). The conceptual approach adopted for a good practice guidance is illustrated in ◘ Fig. 7.21, showing how integrate biodiversity into the mining project cycle (ICMM 2006).

Biodiversity Management

Habitat alteration is one of the most significant potential threats to biodiversity associated with mine operations. Although this alteration can take place at any stage of the mine cycle, there is no doubt that the greatest potential for temporary or permanent alteration of terrestrial and aquatic habitats occurs during construction and operational activities. To integrate conservation requests and development priority in a manner that meets the land utilization requirements of local communities is generally a critical problem for mining projects.

Recommended strategies to solve these issues from the International Finance Corporation (World Bank Group) include consideration of the following (IFC 2007):

1. Whether any critical natural habitats will be adversely impacted or critically endangered or endangered species reduced
2. Whether the project is likely to impact any protected areas
3. The potential for biodiversity offset projects (e.g., proactive management of alternative high-biodiversity areas in cases where losses have occurred on the main site due to the mining development) or other mitigative measures
4. Whether the project or its associated infrastructure will encourage in-migration, which could adversely impact biodiversity and local communities
5. Consideration of partnerships with internationally accredited scientific organizations to, for example, undertake biodiversity assessments, conduct ongoing monitoring, and manage biodiversity programs
6. Consultation with key stakeholders (e.g., government, civil society, and potentially affected communities) to understand any conflicting land use demands and the communities dependency on natural resources and/or conservation requirements that can exist in the area

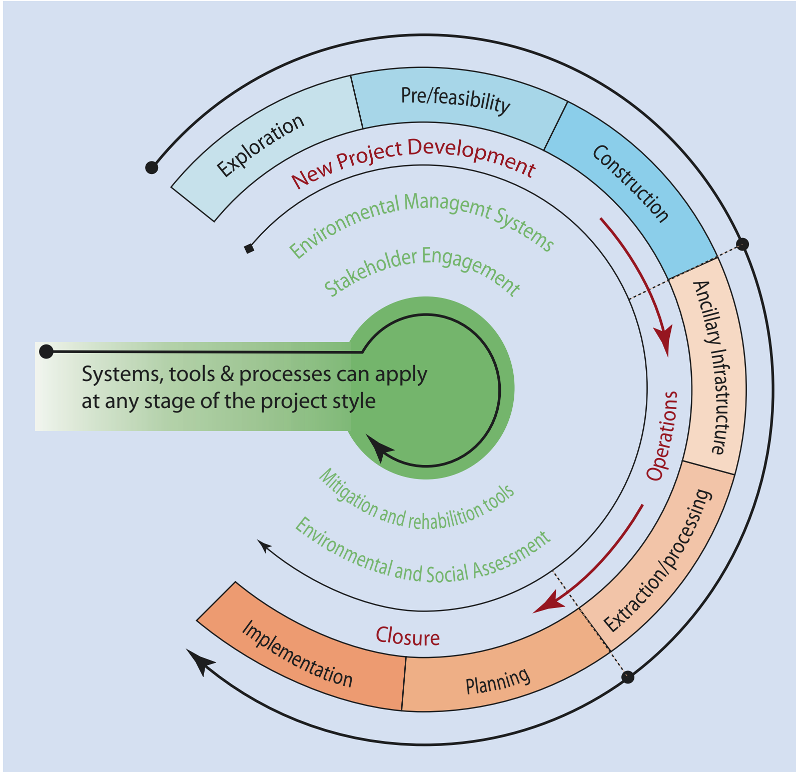

Fig. 7.21 Integrating biodiversity into the mining project cycle (ICMM 2006)

Regarding terrestrial habitat alterations, they must be diminished as much as possible and be consistent with the request to preserve critical habitats. Some controlling strategies include siting access roads in places that prevent adverse effects to critical terrestrial habitat, diminishing disruption to vegetation and soils, and implementing mitigation techniques adequate for the type of habitat. Other strategies are preventing the generation of barriers to wildlife movement and offering alternative migration routes if the generation of barriers cannot be avoided and manage vegetation growth along access roads and at continued above-ground facilities (IFC 2007).

Aquatic habitats are affected through variations in surface water and groundwater flows and generating incremented pressures on fish and wildlife communities. In particular, aquatic habitats in marine environments can be affected by several methods to extract resources such as dredge mining and deep sea mining or other activities such as offshore loading activities, port development, and tailings disposal. Assessment and control of adverse effects for marine environments must be in accordance with suitable host-country obligations to international decisions such as the United Nations Convention on the Law of the Sea (IFC 2007). To an adequate management of potential affections in aquatic habitats, it is essential to maintain water body catchment zones equal or comparable to predevelopment features, preventing stream channel stability by restricting in-stream and bank disturbance and constructing, maintaining, and reclaiming watercourse crossings that are stable and safe for the intended utilization and that decrease erosion, mass wasting, and degradation of the channel or lake bed (IFC 2007).

7.4.6 Airborne Contaminants, Noise, and Vibration Management

Airborne Contaminants

The provision of an adequate air environment to promote the health, safety, and comfort of people has always been and will continue to be an essential requisite for successful mining operations. Airborne emissions can take place during all stages of the mine cycle but specifically during exploration, development, construction, and operation activities. The main sources of these contaminants

Fig. 7.22 Air quality monitoring at a dust collection point near the mine (Image courtesy of Rio Tinto)

are dust from blasting, crushing ore, exposed surfaces such as tailings facilities, stockpiles, waste dumps, haul roads and infrastructure, and, to a lesser extent, gases from combustion of fuels in equipment (Fig. 7.22). Therefore, although dust is the principal emission associated with mines, a rank of gaseous and particle emissions are linked to mining and other on-site processing operations. The adverse effects of air emissions depend on the type of pollutant, its release features, and the nature of the receiving environment. The pollutants can be present in solid, liquid, and gaseous forms. Gaseous emissions generated by fuel combustion or mineral processing include pollutants such as sulfur dioxide and nitrogen dioxide that have well-defined harmful effects and are tightly controlled in the ambient environment and workplace (Commonwealth of Australia 2009b). Since management of air quality at mine operations is essential at all phases of the mine cycle, dust emissions from the dry surfaces of tailings facilities, waste dumps, stockpiles, and other exposed areas should be always minimized.

The sequence of dust control techniques are (a) prevention of generation of dust and its suspension in air, (b) suppression of airborne dust on-site, (c) collection of dust that could not be suppressed, and (d) dilution with auxiliary and main ventilation.

The control strategy for these environmental impacts depends on the type of contaminants, their sources, and rates of emission. It can range from simple dilution with ventilation air to complex procedures for removal of the contaminant prior to mixing with the mine air or suppression/elimination at the source. Thus, the International Finance Corporation (IFC 2007) recommended the following air pollution management strategies:

1. Dust suppression in roads and work areas, optimization of traffic patterns, and reduction of travel speeds.
2. Exposed soils and other erodible materials should be revegetated or covered promptly.
3. New areas should be cleared and opened up only where absolutely necessary.
4. Surfaces should be revegetated or otherwise rendered non-dust forming when inactive.
5. Storage for dusty materials should be enclosed or operated with efficient dust suppressing measures.
6. Loading, transfer, and discharge of materials should take place with a minimum height of fall and be shielded against the wind.
7. Conveyor systems for dusty materials should be covered and equipped with measures for cleaning return belts.
8. Chemical treatment at haul roads.
9. Selection of superquality mine explosives.
10. Installation of dust/gas extraction systems at crushers.
11. Spraying waste rock piles with sealants.
12. Storing crushed ore that is waiting to be processed in the mill in enclosed structures.

Fig. 7.23 Noise control (Image courtesy of Eldorado Gold Corporation)

Noise and Vibration

Noise is an inherent health hazard in mining industry. Raw material exploration, extraction, and processing can generate important levels of noise that can affect the surrounding environment. This is because the mining process is highly mechanized, from the earliest ore removal to final processing, and heavy equipment is essential at virtually every stage of operation. Thus, exposure to noise is a concern for workers who drive mechanized equipment as well as those who operate or work near stationary equipment such as haulage belts or crushing equipment. Moreover, communities can suffer noise and vibration adverse effects from mining operations in many forms, not only from the mine site: noise can take place at all stages of the logistics chain, including rail and truck haulage and activities at ports (Commonwealth of Australia 2009b). Exposure to noise levels above regulatory or recommended limits can result in hearing loss. It is important to emphasize that most hearing loss is preventable. Prevention can be achieved by eliminating noise sources, substituting quieter equipment, installing appropriate engineering controls, implementing administrative controls, using personal protective equipment, and adopting effective hearing conservation programs (Walter 2011).

Good practices in management of the noise sources must be defined based on the prevailing land utilization and the vicinity of noise receptors such as communities or community use areas. Where necessary, noise emissions must be managed (Fig. 7.23) through the application of methods that can include (a) implementation of enclosure and cladding of processing plants; (b) installation of proper sound barriers and/or noise containments, with enclosures and curtains at or near the source equipment (e.g., crushers, grinders, and screens); (c) installation of natural barriers at facility boundaries (e.g., vegetation curtains or soil berms); and (d) optimization of internal traffic routing, particularly to minimize vehicle reversing needs (reducing noise from reversing alarm) and to maximize distances to the closest sensitive receptors (IFC 2007).

The most significant vibrations are usually associated with blasting activities. In this sense, the increasing size and depth of open-pit mines and large diameter long-hole blast in underground mines further aggravate the vibration (Haldar 2013). However, vibrations can also be generated by many types of equipment. Measurement and control of vibration serves two purposes: (1) prevention of premature wear and failure due to structural damage and (2) reduction of noise levels. Measurement of vibration requires specialized equipment and experience in data interpretation.

7.4.7 Other Potential Environmental Impacts

Subsidence

Subsidence of the ground surface can be considered as ground movement caused by the extraction from underground of any resource, whether it be solid, liquid, or gas. It is commonly an inevitable consequence of such activities and reflects the movements that occur in the area so affected. The problems associated with subsidence have been recognized since antiquity. Agricola's De Re Metallica of 1556 talks about «a mountain or hill… subsiding by its weight» as a result of mining. The subsidence effects of mining raw materials are controlled by the type of mineral deposit, the geological features, specifically the nature and structure of the overlying rock or soil, and the mining method applied in the extraction process. In addition, time when subsidence occurs depends upon the type of mining, as does the reliability of subsidence prediction. Thus, the major objectives of subsidence engineering are prediction of ground movements, determining the effects of such movements on structures and renewable resources and minimizing damage. The surface displacements and deformations characteristic of subsidence will affect any use made of the ground surface. Consequently, subsidence can generate serious effects on surface structures, buildings, and communications and can affect agricultural land through the disturbance of drainage and alteration of gradient.

The creation of any subsurface opening produces deformations and displacements of the material, and these changes can cause the rock around a mine excavation to collapse into the mined void. The ground movements associated with such collapse tend to propagate to the ground surface, with the deformations and displacements experienced there being termed subsidence. Surface subsidence generally entails both vertical and lateral movements and can be discontinuous (steps, cracks, or cavities form at the surface) or continuous (the surface deforms smoothly). Discontinuous subsidence is generally of limited areal extent and is characterized by large vertical displacements. It occurs where material overlying an extraction zone collapses into the void, and its form depends on the mining method, the geometry of the extraction zone, and the geomechanical properties of the rock above the extraction zone (Harrison 2011). The presence of weak structural features (e.g., faults or boundaries between different geological materials) can lead to plug subsidence in which a large plug of material falls suddenly and instantly downward into the mine void; the speed and suddenness of the process means this is particularly dangerous. Mining methods such as block caving and sublevel caving also lead to discontinuous subsidence, but in these operations, use of an access to the surface area affected by the subsidence is generally prohibited. In the case of continuous subsidence above laterally extensive extraction zones such as longwall coal mining operations, observations of subsidence profiles or troughs above the mined areas have shown that they can be characterized on the basis of shape, in particular the absence or presence of an essentially horizontal central region.

Regarding the factors affecting mine subsidence, experience has revealed that many geological and mining parameters besides the width of the extraction zone can affect the magnitude of subsidence. The number and interrelation of these factors means that predicting in an accurate, quantitative manner the magnitude and time to subsidence onset is generally not straightforward. The main factors are the following:

1. Extraction thickness: the thicker the material mined, the larger the quantity of possible surface subsidence.
2. Mining depth: magnitude and time to onset of subsidence are dependent on depth.
3. Inclination of extraction horizon: asymmetric subsidence occurs where the zone being mined is inclined.
4. Degree of extraction: reducing the amount of material extracted will reduce the amount of subsidence.
5. Mined area: the critical width of a mined void must be exceeded in all directions if maximum subsidence is to develop.
6. Method of working: the amount of subsidence is largely controlled by the degree of caving induced by the mining method (e.g., complete subsidence for block caving and longwall mining and zero for room and pillar) together with the amount of support offered by any backfilling.
7. Competence of surrounding materials: because subsidence propagates from the mine level, the mechanical behavior of the rock adjacent to the mined void directly affects the initiation of subsidence.
8. Geological discontinuities: the existence of faults can increase and localize subsidence potential so strongly that in areas of adverse geological conditions the effects of the other parameters can be discounted.

9. Near-surface geology: the nature of any near-surface soils and unconsolidated rocks affects subsidence development, with both the thickness and mechanical characteristics of these materials being important.
10. Hydrogeology: the increased groundwater pressure can reduce the effective stress, thereby inducing shear on faults.
11. Elapsed time: subsidence does not occur instantaneously but over a period of time (Harrison 2011).

Measures that can be implemented to control and minimize subsidence damage fall into the categories of adoption of particular mining methods, post-mining stabilization, architectural and structural design, and comprehensive planning. In adopting a particular mining technique, the principal measures to consider are partial mining, changes to the mine layout, harmonic mining, backfilling, and changing the extraction rate. For post-mining stabilization, stabilization of complete mine sites extending over many hectares can be achieved by backfilling (as previously outlined), grouting, or, in the case of shallow voids beneath derelict or unused land, complete excavation and backfilling. Concerning architectural and structural considerations, where structures are to be built in areas of known or future mining activity, designs should be adopted that will tolerate the anticipated ground movements. Many design techniques are available to produce structures tolerant of subsidence.

Visual Impact

Mining activities, specifically surface operations, can generate negative visual adverse effects to resources linked to other landscape utilizations such as recreation or tourism. Potential contributors to visual impacts are roads and highways, erosion, changes in water color, haul roads, waste dumps, slurry ponds, abandoned mining installations (◘ Fig. 7.2), garbage and refuse dumps, open-pits, and deforestation. Regarding color changes, in areas where the color of the rock matches with the natural color of the terrain, visual impacts will be less than with sharp color contrasts.

The impact on landscape by surface mining depends on various factors; location, size, extracted volume, and mining methods can influence the impact of mining activities on the visual appearance of the land (Haney G 2010). Restored lands must conform to the visual features of the surrounding landscape. The reclamation planning should consider the vicinity to public viewpoints and the visual effect within the context of the viewing distance. Alleviation methods can incorporate specific location of screening materials including trees and utilization of adequate types of plants in the reclamation stage as well as changes in the location of ancillary installations and access roads. In this sense, visual absorption capability is described using three physical factors: slope, vegetation (including landscape texture), and geology (landform dissection). Visual absorption capability classifies the relative ability of a landscape to accept human alterations without a loss of landscape character or scenic quality. A typical example of visual impact is that produced from mine waste dumps and leach pads. This is because this adverse effect is a major concern for mines located in the proximity of populated areas or where the facilities are clearly visible from roads and highways.

Landscape alteration can generate an adverse opinion among potential observers and compromise the possible development of the surrounding territory. In fact, the evaluation of landscape and visual impact often is based more on the subjective perception of the observers, which includes cultural and social issues, individual opinions, aesthetic tastes and visual comprehension, and less on the real features of the visible alteration (Nicholson 1995). For instance, Las Médulas Roman Mine (◘ Fig. 1.7) was one of the most important visual impacts of mining two millennium ago, and at present UNESCO includes Las Médulas Cultural Landscape in the list of the World Heritage Sites. However, several aspects of landscape modification require to be objectively assessed to estimate the magnitude of change and offer an objective evaluation of the adverse effects originated by pre-existing mines or to be generated by new mining operations involving surface excavation. Landscaping can be undertaken about mineral workings to reduce their visual impact. For instance, a mine can be screened from view to some extent by the construction of embankments around it that are subsequently planted with grass and trees (Bell and Donnelly 2006).

To prevent visual impact, a Visual Resource Management (VRM) should be carried out. It was

Fig. 7.24 Revegetated waste rock areas (Image courtesy of Eldorado Gold Corporation)

originally created by the Bureau of Land Management (US Department of Interior), and the main goal is to manage public land in a manner that protects the scenic values of the lands. Thus, VRM includes inventorying scenic values and determining management aims for those values through the management planning procedure and then assessing suggested activities to establish whether they conform to the management purposes. The VRM system is split in two parts. The first step is the identification of visual values to determine the appropriate level of management. This step, called VRM inventory, has three components: scenic quality evaluation, sensitivity analysis, and distance zone measures. The second part of the VRM system is the analysis stage. It includes establishing whether the potential visual adverse effects from suggested surface-disturbing activities will meet the management goals defined for the area or whether design adjustments will be requested.

Fire and Explosions

Fires and explosions have the potential to kill people in addition to causing an environmental impact. Presence of methane is probably the most characteristic source for this issue although flammable and combustible liquids are often stored underground in most mines and pose a special fire hazard (WorkSafe New Zealand 2016). The content of methane is specific to underground mines where operations are focused on the exploration or extraction of coal or metalliferous mines and tunnels where methane is present at levels greater than 0.25%. For this reason, it is essential to develop fire and explosion risk assessments and to identify the measures required to prevent, manage, and mitigate those risks.

7.4.8 Revegetation

Reclamation management must consider soil structure and fertility, microbe populations, top soil developing, and nutrient cycling with the objective to convert the ecosystem as closely as possible to its early conditions (Sheoran et al. 2010). Thus, establishing vegetation is essential in reclaiming mined lands (Fig. 7.24). The establishment of vegetation can reduce erosion, significantly increase evapotranspiration, and reduce the amount of water that infiltrates the underlying waste material. Direct revegetation may allow many of the benefits of a store-and-release cover to be realized without the need to import large volumes of cover material (Borden 2011). For instance, revegetation of tailings impoundments can be particularly important to prevent dust generation from inactive tailings surfaces as they dry out. In this sense, some waste surfaces may be directly revegetated after minor physical or chemical modification, such as ripping to reduce compaction, addition of alkaline materials to increase the pH to near neutral, or the addition of organic matter. Thus, revegetation tests for mineral wastes may progress from nutrient analyses and grain-size distribution, to greenhouse trials, and to field revegetation test plots and plant tissue sampling (to determine metals uptake) (Borden 2011).

Revegetation fosters soil development, generates aesthetically landscapes, and facilitates post-mining land use. Thus, revegetation is the most broadly admitted and helpful manner of restoration of mine works with the objectives to decrease erosion and protect soils against degradation. The revegetation must be established with the plants elected in accordance with their capability to subsist and regenerate in the particular environment and on their capability to stabilize the soil framework. In this sense, numerous factors must be taken into account in an efficient mined-land revegetation procedure such as soil features, time of seeding, species seeded, and soil amendment application rates.

Revegetation in a zone impacted by mining works, once the final landform has been developed and an adequate growing medium generated, includes five main steps: (a) mine soil selection and placement procedures, (b) species selection, (c) planting, (d) seed collection and purchase, and (e) seedbed preparation. In general, the optimum moment to establish vegetation is defined by the seasonal pattern and reliability of rainfall. All the previous works must be finished prior the time when seeds are most likely to experience the conditions they need to germinate (Minerals Council of Australia 1998).

A plan for revegetation includes, but not limited to, descriptions of the revegetation schedule; species and amounts per square meter of seeds and seedlings to be used; methods to be used in planting and seeding, mulching techniques, and irrigation (if appropriate); pest and disease control measures, if any; measures proposed to be used to determine the success of revegetation; soil testing plan for evaluation of the topsoil results; and handling and reclamation procedures related to revegetation (Nelson 2011). Several immediate revegetation establishment options exist, including drill seeding, hydroseeding, broadcast seeding, and transplanting entire live plants or plant cuttings. In addition, the placement of mulch can increase soil moisture, provide a temporary cover to reduce erosion risk, moderate soil temperature, and increase the likelihood of seed establishment.

Mine Soil Selection and Placement Procedures

Correct revegetation processes of active open-pit mines start well early of fertilization and seeding. Thus, the most significant stage in surface mine revegetation takes place where the soil is chosen and located on the land surface. With the objective to obtain an optimum plant growth, the soil must be elected to offer physical and chemical features proper for the aimed post-mining land use. Fertilization and, in some instances, liming are significant elements of revegetation processes. The most efficient manner to attain a correct combination among soil characteristics, species, and post-mining land use is to choose and place surface-soil materials to generate a soil that is beneficial to vegetation congruent with the post-mining land use declared in the mining permission. Election and arrangement of surface spoils will have a crucial impact over vegetation success in post-mining land use. Lime, fertilizer, and organic component additions can be added to remediate issues of low soil fertility and/or moderate acidity.

Species Selection

Frequent issues associated with revegetation defeat are the inadequate election of plant species and their unsuitable mixtures. Sometimes, the chosen species are either not adjusted to the site characteristics or to the suggested land utilization. The species for establishment will be selected based on the future land utilization of the zone, soil characteristics, and weather conditions. Many rehabilitation processes are directed toward the reestablishment of native species. If the main goal is to restore the pre-mining conditions, then the species must be preset (◘ Fig. 7.25). However, a decision must be taken whether to utilize only local origin of the native species or to utilize a broad rank of sources. This decision requires to be made on a site-by-site basis, usually depending firstly on the degree of similarity between the pre- and post-mining environmental features.

Where the aim is the reestablishing of a diverse and permanent cover of local species, the

Fig. 7.25 Predetermined species for revegetation (Image courtesy of Daytal Resources Spain S.L.)

following methods of determining suitable species for the post-mining conditions should be followed:

1. Observe plant species growing naturally on any old disturbed areas near the rehabilitation site so that the effective colonizing species can be identified.
2. Observe the soil and drainage conditions to which the different local species are adapted, and match them with the conditions on the mine site.
3. Identify plant species that produce sufficient viable seed to harvest economically.
4. Consider habitat requirements where return of wildlife to the area is a significant element of post-mining land use.
5. Consider planting local legume species as they are often good colonizers and will improve soil fertility (Minerals Council of Australia 1998).

Three main types of plants are used for revegetation of mine sites: grasses, forbs, and trees. Grasses are the most generally seeded plants in revegetation procedures. They have fibrous roots that maintain soil in place to control erosion. Forbs are commonly utilized in mine revegetation combined with grasses, while trees are the final plant type; they are applied where forested or wildlife habitat land use is selected after mining. Where agriculture is the desired land use, legumes must always be taken into account for their capability to enhance soil fertility. Legumes are significant for revegetating mine sites since they transfer the «fixed» nitrogen to other elements of the plant/soil system. A population of legumes is crucial to an adequate revegetation, mainly on sites where topsoil replacement is not sufficient.

Planting

The planting techniques elected will be based on the size and nature of the mine sites and the species to establish. Direct seeding is potentially a costly effective and reliable technique to establish species that generate sufficient numbers of easily collected, viable seed with high germination, and seedling survival rates. Advantages include low cost, random distribution of plants, and no check on growth rates through planting out. Disadvantages include higher risk of failure through adverse climate conditions, competition from weeds, loss of seed by insect predation, and low seed germination and survival rates. For planting seedlings, a reliable supplier of seedlings or the establishment of an on-site nursery is obligatory (Fig. 7.26). Advantages are the effective utilization of forthcoming seed, control over species mixture and location, and less limitation on the species considered in the revegetation program. Disadvantages include higher costs for planting and/or nursery operation or purchase of seedlings, check in growth rate at planting, need to preorder or sow several months previous to anticipated utilization, longer planting time needed, and seedlings can deteriorate if planting is delayed.

Another option to planting is transplanting (Fig. 7.27). Transplanting of trees and ground covers is adequate for certain sites or amenity planting. Advantages include immediate solution and incorporation of species not amenable to other means of propagation. Main disadvantage is high risk of expensive defeats. Where individual mature trees are needed for the rehabilitation process, transplanting should be ended while suitable earthmoving and lifting equipment is on-site (Minerals Council of Australia 1998). In some

Fig. 7.26 On-site nursery (Image courtesy of Goldcorp Inc.)

Fig. 7.27 Transplanting of mature tree (Image courtesy of Eldorado Gold Corporation)

Fig. 7.28 Spreading mulch (Image courtesy of Tronox)

cases, the most usual method to seed and apply amendments is using a hydroseeder. Fertilizer, lime, mulch (Fig. 7.28), and seed are commonly mixed with water in the hydroseeder tank.

Seed Collection and Purchase

A consistent supply of adequate seed is crucial for the success of revegetation. Seed can require to be obtained from different zones with the aim to match site characteristics since there are many issues inherent in collecting native seeds. Seed of several species needs pre-sowing treatment. Thus, germination of most native legumes and a number of other species is improved by heat treatment. Most companies utilize seed mixtures including at least two or three perennial grasses, two or three legumes, and either a warm-season annual or a cool-season annual for quick cover. Thus, a broad variety of species is suitable for utilization in mine sites rehabilitation.

Seedbed Preparation

Methods selected for the preparation of the seedbed will be based on topography of the site, the required land use, the extent of soil amelioration and fertilizer utilization, and the sowing or planting method suggested. The objective in creating a seedbed is to place the seed in an adequate location for germination. For this purpose, points to consider include:

1. Prevent compaction, crusting, and subsequent erosion by avoiding disturbance to soils when wet and sticky or dry and powdery.
2. Timing of seedbed preparation and sowing is often critical for successful establishment of vegetation.
3. Where the topsoil contains significant quantities of seed of desirable species, care must be taken not to disturb the soil after these seeds have started to germinate, as this will cause a substantial reduction in plant establishment.

4. Where hand planting of seeds or seedlings is proposed, site preparation can best be limited to deep ripping (Minerals Council of Australia 1998).

Biosolids

Biosolids are «the dark, organic, and nutrient-rich materials produced as byproduct of current wastewater treatment practices» (EPA 2001). An increasing option to traditional waste disposal is the land application of biosolids. Since they include many nutrients and metals necessary for plant life, biosolids are capable to serve as fertilizers and as a mine reclamation alternative. Thus, biosolids have been utilized successfully at mine sites to establish vegetation. Not only do the organic matter and nutrients in the biosolids decrease the availability of toxic components commonly encountered in disturbed mine soils, they also build a healthy soil layer where little soil has been left. They can also be applied for treating acid mine drainage from abandoned mines. Biosolids are able to efficiently establish a vegetative cover on contaminated lands and limit the movement of metals through erosion, leaching, and wind. Depending on the amendments added, biosolids can serve many purposes, including pH control, metal control, and fertilization. Moreover, their adaptability enables them to conform to the specific features of any reclamation site.

7.5 Potential Social Impacts

A community is usually a diverse group of people with some common bonds. Diversity can come in the form of gender, ethnicity, religion, race, age, economic or social status, wealth, education, language, class, or caste. As a result, individuals of any community are likely to hold diverse perceptions about a mining operation and its activities as well as most other subjects. Individuals within a community will have different and sometimes overlapping associations with the mine as neighbors, employees, suppliers, and so on. It is not uncommon for disagreement and sometimes conflict to develop between different sections of a community in relation to mining operations (Evans and Kemp 2011). More recently, the term stakeholder has become a common term that is related to but distinct from community. A common definition of stakeholders is those who have concern in a specific choice, either as individuals or representatives of a group. This covers people who influence a decision, or can influence it, as well as those affected by it (MCMPR 2005). Thus, this term can include local community members, NGOs, governments, shareholders, and employees.

The social impacts of mining projects have received increasing attention in recent years. Although it has been commented that mining can be a crucial economic impeller for developing countries because it can facilitate industrialization along with the promises of wealth and jobs, mining can also be a source of social discontent. In fact, the social cost of mining interacts with other cultural and environmental issues that call for concerted efforts in addressing them. Thus, unmitigated negative social impacts have the potential to result in negative publicity, incremented litigation processes, and reputational damage or to delay, prevent, or close down mining in existing and prospective areas because of community concerns. In this sense, it is also interesting to introduce the concept of social risk. A social risk is the potential for an existing or planned project to have an impact on individuals or groups or, conversely, to be impacted by them. Like impacts, social risks are both positive and negative because of the potential for mining to generate social and economic opportunities, such as economic and community development and employment (Franks 2011).

Many factors can have a significant impact on the interactions and relationships between mining operations and communities, including various social and political aspects, as well as the stage of the mining life cycle involved. Mining is a truly global activity, involving many different types of organizations and communities in settings that range from arid mountains in parts of the Andes or remote areas within the Arctic Circle to established agricultural regions in developed countries and to tropical rainforest settings in developing economies in Asia (Evans and Kemp 2011). In this sense, political and legal frameworks within a country will have a significant impact on the scale and nature of the mining industry and can also often be the subject of intense community focus. Government capacity to regulate the minerals industry and manage the benefits of mining for the local communities has been identified as a crucial aspect by recent studies and has been the subject of recent World Bank projects in several developing countries.

In the nineteenth century and most of the twentieth century, all involved entities such as governments and mining companies paid little interest to the adverse impact of mining on indigenous people. Consequently, it has become almost impossible for different indigenous communities to commit successfully with contemporary issues that impact on their communities such as resource development propositions (Commonwealth of Australia 2007c). Based on the above, the social impacts of mining activities and projects have received increasing attention in recent years.

A social impact is considered as «something that is experienced or felt (real or perceived) by an individual, social group, or economic unit; social impacts are the effect of an action (or lack of action) and can be both positive and negative» (Franks 2011). Obviously, social impacts can vary in type and intensity and over space and time. Moreover, many times an environmental impact induces a social impact because mining activities can originate changes to community amenities, health, or accessibility and quality of water and land. Though it has been argued that mining can be a vital economic propellant for most countries, especially the developing ones, sometimes it can also be a source of social discontent. In fact, the social cost of mining interacts with other cultural and environmental issues that call for concerted efforts in addressing them.

If communities think that they are being unjustly treated or improperly compensated, mining projects can originate social tension and violent conflict (ELAW 2010). Communities feel especially vulnerable where links with different sectors of the society are weak or where environmental impacts of mining affect the subsistence and livelihood of local people. Thus, the main impacts of mining projects on social values can include (a) human displacement and resettlement and migration, (b) lost access to clean water, (c) impacts on livelihoods and public health, and (d) impacts to cultural and aesthetic resources (ELAW 2010).

However, it could be stated that well-managed mineral projects can deliver abroad range of long- and short-term profits. Thus, many countries have benefited from foreign exchange earnings, incorporation of new technologies, enhanced investment opportunities, construction of infrastructure, and education of mine workers and their families (Anderson 1997). Moreover, in some cases mine works form the most significant economic resource. In this sense, the closure of mine can have a strong unfavorable socioeconomic impact. The social issues originated by the closure of a mine can be partially mitigated through the retraining of the workers to newer employment possibilities and newer companies (Aswathanarayana 2005).

In spite of the social impacts and concerns, literature reveals that efforts at mitigating the impacts of mining have only focused on the environmental impacts and have been wrongly assumed that dealing with the environmental impacts alone would inevitably reduce the social impacts. The fact that policy initiative responses are usually geared toward environmental impact assessment implies that social impacts are necessarily not considered (Opoku-Ware 2010). Thus, social impacts are commonly mentioned exclusively in the context of environmental impact studies, alluding to impacts that affect communities causing changes in their welfare. Many companies have concentrated much effort on employment especially for indigenous people and have created programs to support them in their shift from welfare to work (Jantunen and Kauppila 2015).

Gender is obviously an essential aspect to understand the concept of community. Mining is usually a male-dominated industry, but women play significant roles in communities as workers, as family members, and as individuals and are generally very active forming groups in the community. In some situations, «special effort can be needed to ensure that women's perspectives are sought and that women are proactively included in community engagement and development programs because women are deprived of the access to the benefits of mining developments, especially money and employment» (Commonwealth of Australia 2006b).

7.6 Environmental Impact Assessment (EIA)

Including the environment into development planning is the most essential tool in accomplishing sustainable development. Because of the increased concern over the impact of human activity on the environment, most countries have adopted legislation requiring that the potential effects of new projects should be assessed. Consequently, environmental protection and economic development must be carried out in an integrated way. For this objective, the environmental impact assessment (EIA) process is essential to provide an anticipatory and foreseeing procedure for environmental management and protection in any development. EIA is a complex study that must be developed and approved by the government authorities where industrial operations are permitted. In other words, the process of establishing potential environmental effects of a proposed project is known as environmental impact assessment, and it must enable the best environmental option to be determined and adequate mitigation to be involved. Nowadays, environmental impact assessment and utilizing the required measures for industrial and mining projects are crucial to prevent and control environmental problems.

The environmental impact assessment procedure is an interdisciplinary and multistage process to assure that environmental characteristics are taken into account in decisions related to projects that can affect the environment. In a simple manner, the EIA process assists to detect the potential environmental effects of a suggested action and how those impacts can be alleviated. Thus, the principal aim of the EIA procedure is to inform decision-makers and the public of the environmental results of implementing a suggested project. The EIA process also helps as a decisive procedural role in the global decision-making procedure by fostering transparency and public involvement. It is important to bear in mind that the EIA procedure does not guarantee that a project will be changed or rejected if the process shows that there will be intense environmental footprints. In other words, the EIA process assures a documented decision but not indispensably an environmentally beneficial resolution (ELAW 2010). At the international level, lending banks and bilateral aid agencies have EIA processes that implement to borrowing and recipient countries (Ogola 2007).

7.6.1 Origin of EIA

Before the First World War, quick industrialization in developed countries generated a rapid decrease of natural resources. This process maintained to the period after the Second World War, originating important issues related to pollution, quality of life, and environmental stress. In early 1960s, investors notice that the projects they were developing were affecting the environment, including people. For this reason, pressure groups constituted with the objective of getting a tool that can be utilized to protect the environment. Consequently, several developed countries such as Australia, Japan, Sweden, or the USA decided to respond to these problems and established different environmental protection laws. For instance, Sweden published the Environmental Protection Act in 1969, Australia the same document in 1974, and the USA developed in 1969 the National Environmental Policy.

In those years, these documents were the first documented as official tools to be utilized to safeguard the environment. Regarding these documents, complications can take place where there is overlapping among regulation at national, regional, and local level. This can be the case in large countries such as the USA or to member states of the European Community. Furthermore, industries working on a global scale may be subjected to a great variety of EIA requirements, specific to each country of operation. However, although EIA legislation changes in complexity from one country to the next, there is a clear underlying theme: potential impacts of certain projects must be assessed and documented during the planning stage.

Likewise, the United Nations Conference on the Environment in Stockholm in 1972 and further conferences formalized EIA. Nowadays, all developed countries and many developing countries have environmental laws to restrict the environmental impacts generated by the industry. Principle 17 of Rio Declaration on Environment and Development in 1992 claim for utilizing EIA as a decision-making component to be applied in evaluating whether suggested activities are likely to have important adverse effects on the environment. Thus, EIA is carried out within the legal and/or institutional frameworks defined by countries and international agencies (Ogola 2007).

7.6.2 EIA Phases

The early stage of an EIA is termed the «Initial Environmental Examination (IEE),» and the second is the «Environmental Impact Studies (EIS)» or merely detailed EIA. IEE is developed to establish whether possible unfavorable environmental effects are important or whether mitigation measurements can be adopted to decrease or even remove the adverse results. The IEE includes a short statement of main environmental problems obtained using forthcoming information, and it is utilized in the first stage of project planning. The IEE also decides if further in-depth studies are required. Where an IEE allows offering a final solution to environmental issues of a project, an EIA is not needed.

EIS or detailed EIA is a process utilized to study the environmental effects, both positive and negative, of a proposed project and to assure that these consequences are considered in project design. Consequently, the EIS is based on predictions. The adverse effects can include all significant items of the natural, social, economic, and human environment. The study needs a multidisciplinary focus and must be carried out very early at the feasibility stage of a project. In other words, a project should be assessed for its environmental feasibility. Thus, EIS should be established an integral part of the project planning procedure.

Finally, the analyses of alternatives are carried out to define the preferred or most environmentally sound, financially viable, and benevolent possibility for accomplishing project goals. The World Bank directives request systematic comparison of suggested investment designs. For each alternative, the environmental cost is estimated as far as possible and economic data enclosed where feasible and the selected alternative stated. The analysis of alternatives must always incorporate the so-called no project alternative.

7.6.3 Impact Analysis and Prediction

Predicting the extent of impacts and estimating their significance are essential in environmental impact assessment processes. Prediction should be based on the available environmental baseline of the project area, being these predictions described in quantitative or qualitative manner. According to Ogola (2007), the considerations in impact prediction must include:

1. Magnitude of impact: this is defined by the severity of each potential impact and indicates whether the impact is irreversible or reversible and estimated potential rate of recovery; the magnitude of an impact cannot be considered high if a major adverse impact can be mitigated.
2. Extent of impact: the spatial extent or the zone of influence of the impact should always be determined; an impact can be site-specific or limited to the project area.
3. Duration of impact: environmental impacts have a temporal dimension and need to be considered in an EIA; an impact that generally lasts for only 3–9 years after project completion can be classified as short term; an impact that continues for 10–20 years can be defined as medium term, and impacts that last beyond 20 years are considered as long term.
4. Significance of the impact: this refers to the value or amount of the impact; once an impact has been predicted, its significance must be evaluated using an appropriate choice of criteria.

7.6.4 Methods for Identification of Effects and Impacts

There are three main methods for assessing environmental impacts: checklists, flow diagrams, and matrices (Sorensen and Moss 1973). Checklists are complete registers of environmental effects and impact gauges established to encourage the analyst to think widely about potential consequences of contemplated actions. However, this strength can also be a weakness because it can lead the analyst to ignore factors that are not on the lists. In any form, checklists are included in almost all EIA methods. In some cases, flow diagrams are utilized to look for action-effect-impact relationships. They allow the technician to visualize the connecting between action and impact. This method is most suitable to single-project assessments, not being recommended for large regional actions. Regarding matrix method, it is probably the most used in the EIA (Box 7.5: Matrix Method in the EIA). The matrix method in environmental impact assessment studies can be very helpful due to its simplicity and understandability of its algorithm.

7.6.5 EIA for Mining Projects

The EIA is the accepted method for evaluating proposed mining projects to obtain regulatory approval and to help companies plan for responsible development. From its early beginnings to its development over the past three or four decades, the EIA has become increasingly exacting, paralleling the development and expansion of international and national standards. Generally accompanied by environmental and social management plans, the EIA has undeniably become the essential regulatory document required of new mines by governments worldwide (Mitchell 2012). Thus, before any mining project can be carried out, it must undergo an environmental assessment as legislated by local or national governments. In this sense, each jurisdiction has different regulations governing environmental review, and some are more stringent than others (Stevens 2010).

Initially, an EIA was only requested in highly regulated circumstances. Nowadays, it is impossible to find a major mining project anywhere in the world that is not requested, either by legislation or corporate standards, to undertake an EIA. In general, an EIA for a mining project must include (a) assessment of the current state of the environment; (b) definitions of various project alternatives, assessments of their environmental impacts, and a comprehensive picture of the impacts of the project and its implementation alternatives, presented together with assessments of the scale and significance of such impacts; (c) plans for the mitigation of detrimental impacts; and (d) the publication of an accurate and coherent EIA report (Jantunen and Kauppila 2015).

Box 7.5

Matrix Method in the EIA

The matrix method was initially developed by Dr. Luna Leopold and others of the US Geological Survey (Leopold et al. 1971) in response to the Environmental Policy Act of 1969. As Gillette previously stated (Gillette 1971) «the law's instructions for preparing an impact report apparently are not specific enough to insure that an agency will fully or even usefully, examine the environmental effects of the projects it plans.» This method consists of a matrix that is primarily a check list designed to show possible interactions between development activities and a set of environmental characteristics. Combining these lists as horizontal and vertical axes for a matrix allows the identification of cause-effect relationships between specific activities and impacts. This matrix has (1) on the horizontal axis the actions that cause environmental impact and (2) on the vertical axis the existing environmental conditions that can be affected by those actions. This provides a format for comprehensive review of the interactions between proposed (anthropogenic) actions and environmental factors (characteristics and conditions). The entries in the cell of the matrix can be either qualitative estimates or quantitative estimates of these cause-effect relationships. The latter are in many cases combined into a weighted scheme leading to a total «impact score.» The original Leopold system was an open-cell matrix containing 100 project actions along the horizontal axis and 88 environmental «characteristics» and «conditions» along the vertical axis. This provides a total of 8800 interactions. However, in practice only a few of the interactions would be likely to involve impacts of such magnitude and importance to warrant detailed treatment.

Matrix methods identify interactions between various project actions and environmental parameters and components. They incorporate a list of project activities with a checklist of environmental components that might be affected by these activities. They should preferably cover both the construction and the operation phases of the project because sometimes the former causes greater impacts than the latter. Simple matrices are useful: (1) early in EIA processes for scoping the assessment, (2) for identifying areas that require further research, and (3) for identifying interactions between project activities and specific environmental components. Matrix method is probably the most used in the identification of effects and impacts. However, it also has their disadvantages since it does not explicitly represent spatial or temporal considerations and does not adequately address synergistic impacts. Figure 7.29 shows an example of an environmental impact matrix.

POTENTIAL IMPACTS	MINING ACTIVITIES	**Exploration and construction**	Early stages of exploration	Exploration drilling	Access road construction	Land clearance [for construction, etc.]	Obtaining construction materials	Construction related infrastructure	**Construction of ancillary infrastructure**	Roads, rail & export infrastructure	Pipelines for slurries or concentrates	Energy/power & transmission lines	Water sources, wastewater treatment	Transport of hazardous materials
Impacts on terrestrial biodiversity														
Loss of ecosystems and habitats				●	●	●	●			●	●	●		●
Loss of rare and endangered species				●	●	●	●	●		●	●	●		●
Effects on sensitive or migratory species				●	●	●	●	●		●	●	●		●
Effects of induced development on biodiversity					●	●		●		●				●
Aquatic biodiversity & impacts of discharges														
Altered hydrologic regimes					●	●	●	●		●		●	●	●
Altered hydrogeologic regimes				●			●							
Increased heavy metals, acidity or pollution				●		●	●	●		●		●	●	●
Increased turbidity [suspended solids]				●	●	●	●	●		●	●	●	●	●
Risk of groundwated contamination				●			●	●		●	●		●	●
Air quality related impacts on biodiversity														
Increased ambient particulates [TSP]				●	●	●	●	●		●		●		●
Increased ambient sulfur dioxide [SO_2]								●				●		●
Increased ambient oxides of nitrogen [NO_x]								●				●		●
Increased ambient heavy metals												●		
Social interfaces with biodiversity														
Loss of access to fisheries						●	●			●	●	●		
Loss of access to fruit trees, medicinal plants						●	●	●		●	●	●		
Loss of access to forage crops or grazing					●	●	●	●		●	●	●		
Restricted access to biodiversity resources						●	●			●	●	●		
Increased hunting pressures				●	●	●	●	●		●		●		●
Induced development impacts on biodiversity					●	●	●	●		●		●		●

Fig. 7.29 Example of an environmental impact matrix

EIA of mining projects request an approach of the entire life cycle of a mine, from exploration to mine closure and reclamation. Mining companies have realized that this is the most cost-effective method to planning and managing a mine and, particularly, to managing environmental effects (Weaver and Caldwell 1999). In summary, EIA can help to reduce costs and unscheduled project delays and minimize future economic and environmental liabilities. As aforementioned, a credible approach to EIA by the proponent company can serve to support the reputation of both the company and the mining industry generally as participants in planning for the sustainable development of the world's resources. In this sense, key aspects of the EIA procedure of mining projects must include (a) broad participation; (b) the public availability of documents prepared during the EIA procedure (EIA program, EIA report, and the statements and opinions of the competent authority and other parties), (c) review of the various project alternatives, (d) broad definition of the environmental impacts of the project, and (e) assessment of the environmental impacts that will occur during the various stages of the project (planning, construction and commissioning, operation, and closure) (Jantunen and Kauppila 2015).

In all the EIA procedure, consultations are an essential component of the environmental revision process. Consultations enable experts, government, communities, and indigenous people a possibility to discuss the adverse effects of the mining project, occurring at different phases in the review process. In this sense, mining companies that adhere to the principles of sustainable development commonly include consultations since the early prospection stage of the mineral deposit. As a result, the company will likely have addressed any significant concern with the project before it officially begins the review process.

Depending on the EIA method, liability for generating a mining EIA will be allocated to one of the following: the government agency or the project proponent. For proponents, a correctly coordinated EIA of a suggested mining project can help substantially to efficient planning. If EIA laws permit, either party can opt to recruit a consultant to carry out the EIA or handle certain parts of the EIA procedure. In this sense, some EIA laws accept conflict of interest generated where a mining company or other project proponent recruits an external consultant to draft an EIA. Utilizing a consultant carries the risk that the paper will be influenced in favor of developing the mining project. For this reason, some laws request consultants to be registered with the government and/or a professionally accredited organization in EIA preparation. In some cases, a consultant can be requested to file a statement disclosing any financial or other interest in the result of the project (ELAW 2010).

Stages of the EIA Process

EIA must be a procedure that proceeds throughout the life cycle of a mining project with results that become ever more accurate. ◘ Figure 7.30 shows the EIA process in connection to the commented life cycle of a mine (Jantunen and Kauppila 2015). Pre-feasibility reports usually offer an adequate basis for carrying out an EIA because they classically approach to the geology of the property, types of ore deposits, resource estimations, mining and mineral processing techniques, management of mining wastes, requirement for infrastructure, water and energy consumption, and labor and transportation costs. At this phase, estimates of these factors cannot be awaited to be especially accurate (margins of error can range between 20% and 30%). This is because the information about the project is still clearly imprecise.

To generate a suitable EIA, the planning of the project must be so advanced that its adverse effects can be evaluated accurately and reliably enough. For instance, it is essential to have precise knowledge of the technical solutions that will be utilized in the project to allow accurate quantitative and qualitative assessment of emissions. However, the EIA process cannot be left too late because it must be finished before a mining project can obtain the necessary permits. It is a sound practice to start the permit procedure for a project only after the EIA process has been finished. Thereafter, the EIA document and the qualified authority's statement on the report must be enclosed to the permit applications for the mining project. Thus, the EIA process is commonly formed by a group of procedural stages culminating in a written impact assessment document that will report the decision-maker whether to approve or reject a proposed mining project (ELAW 2010).

The first stage includes the identification and definition of the project or activity. Although this stage can be comparatively easy, definition of a project for the purpose of an EIA can be very difficult and even controversial if a mining project is large and has several phases, or multiple sites must be covered. The aim of this phase is to define the project with sufficient specificity to accurately establish the area of potential adverse effects and to incorporate activities that are strictly linked with the proposition, so that the entire scope of environmental impacts is assessed. In this step, the screening process establishes whether a certain project warrants preparation of an EIA. In some instances, especially if the potential impacts of a project are not understood, a previous environmental evaluation will be outlined to establish whether the project warrants an EIA.

The next step, scoping, commonly involves the interested parties that identify the key environmental problems that should be addressed in an EIA. This phase offers one of the earliest opportunities for members of the public to learn about a suggested project and to voice their opinions. Scoping can also show connected activities that can be occurring near a project or identify issues that request to be mitigated or that can originate the project to be canceled. In this procedure, the terms of reference serve as a roadmap for EIA preparation and should ideally embrace the adverse effects that have been identified during the scoping. A draft «terms of reference» can be

7

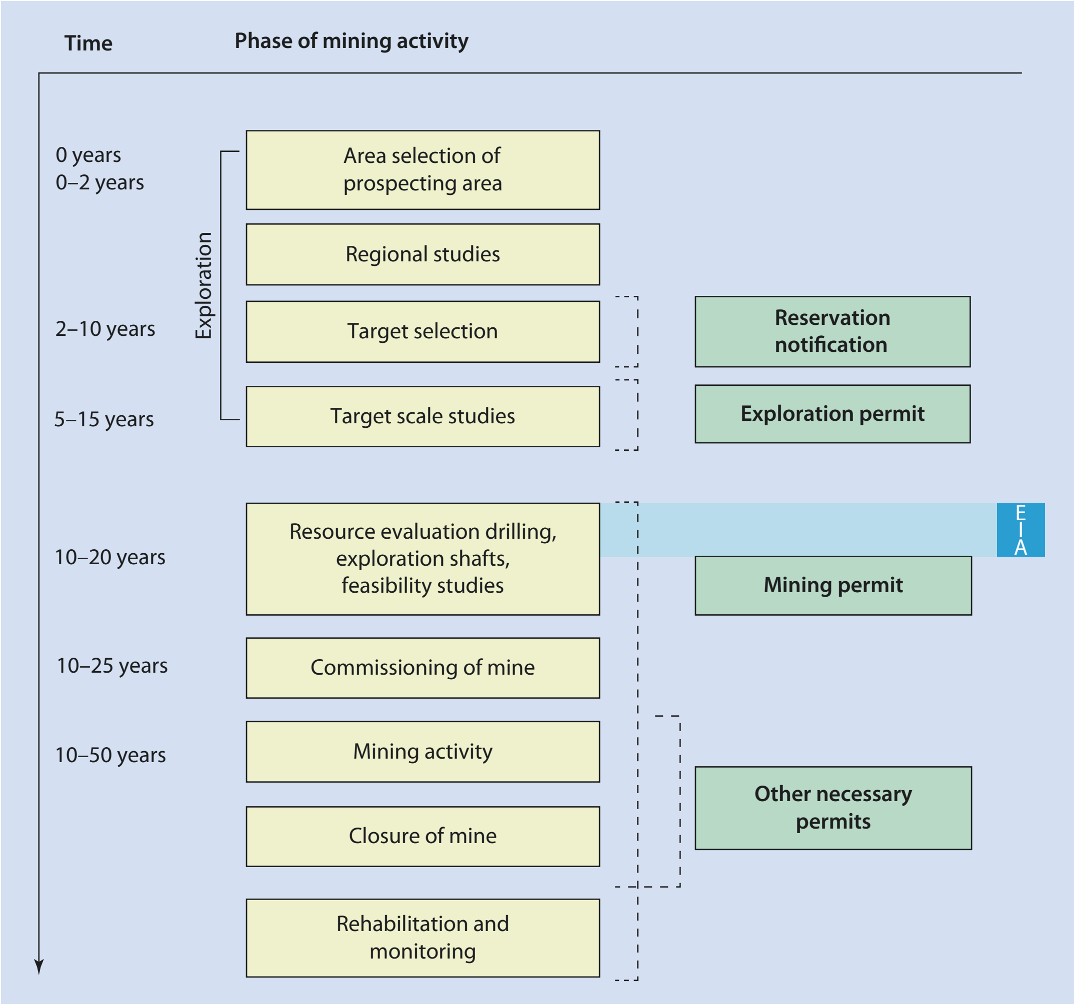

Fig. 7.30 EIA procedure in relation to the life cycle of a mine (Jantunen and Kauppila 2015)

made forthcoming for public revision and comment. Public revision at this first phase of the process originates a good opportunity to assure that the EIA is adequately framed and will address issues of community concern.

Then, a draft EIA is developed according to the terms of reference and/or the rank of problems identified during the scoping procedure. The draft EIA can also meet the content requirements of the global EIA regulations. This phase will ideally take part a broad range of technical specialists to assess baseline conditions, forecast the likely adverse effects of the project, and establish mitigation measurements. Regarding baseline studies, they identify the present status of the physical, social, and economic environment before the project starts, and technical studies define the features of the project. The technical details of the project will be obtained from pre-feasibility or feasibility studies that commonly are finished previous to beginning of the environmental assessment process. Baseline studies generally take several years to complete and, in some cases, commonly start at the beginning of the exploration stage (Stevens 2010).

The next stage generates a final impact assessment document that tackles the points of view and comments of the parties that revised the draft EIA. These comments can promote revisions or additions to the report of the draft EIA. In some instances, this final EIA will include an appendix synthesizing all of the commentaries received from the public and interested institutions and supply responses to those comments. A decision to approve or reject a mining project is commonly based on the information provided for the final EIA, but, in some cases, an environmental clearance can be just one stage in the mine permitting procedure. Once the mine is permitted, monitoring procedure is an important tool of project implementation. According to ELAW (2010), monitoring serves three purposes: (1) ensuring that required mitigation measures are being implemented, (2) evaluating whether mitigation measures are working effectively, and (3) validating the accuracy of models or projections that were used during the impact assessment process.

7.7 Social Impact Assessment (SIA)

Social impact assessment (SIA) can be defined as «the process of managing the social issues of projects.» To be efficient, the management of social issues requires to begin from the moment a project is early planned right through to further closure. Corporations can carry out SIA as part of their liability to address their social impacts and their wish to obtain a social license to operate. The origin of this type of study was in the 1970s, and the main goal of SIA has varied from early concerned about the adverse impacts of a project to being more concerned about how a project can be improved. This is with the aim of increasing the profits to communities so that both communities and companies can benefit from the project. These studies will have more importance in the near future, and its request will continue to increase for several reasons, including the incrementing investment in developing countries. In this sense, a combined action of weak institutions and decreasing land accessibility generates potential for disagreement between companies and communities, mainly if the risks are not early defined and mitigation planning is not implemented or not carried out in cooperation with the impacted peoples themselves.

In addressing the social aspects of sustainable development, social impact assessment early emerged as a component within environmental impact assessment (EIA) used to evaluate, moderate, and invariantly mitigate the impact of planned interventions (Esteves et al. 2012; Mahmoudi et al. 2013). The impact of projects and policies on the social welfare of communities is clearly a topic of increasing concern, which justifies the increased development and practice of SIAs in the last years (e.g., Vanclay and Esteves 2011). Moreover, SIA is a common requirement of regulatory approval processes at the project approvals phase for mining and processing stages in many jurisdictions.

The good practice of SIA accepts that social, economic, and environmental issues are inherently interconnected. Thus, change in any of these fields immediately generates changes in the other domains. According to Esteves et al. (2012): there is consensus on what good SIA practice is: (a) it is participatory; (b) it supports affected peoples, proponents, and regulatory agencies; (c) it increases understanding of change and capacities to respond to change; (d) it seeks to avoid and mitigate negative impacts and to enhance positive benefits across the life cycle of developments; and (e) it emphasizes enhancing the lives of vulnerable and disadvantaged people.

Social impact assessment has been early included within the field of Sociology and related sub-areas (Environmental Sociology, Human Geography, etc.), but different professionals from many disciplines have developed experience in the field. It is essential to consider the SIA in context with the other parts of the project, specifically the environmental impact assessment that must be also submitted with the application for an exploitation license (BMP 2009). Social impact assessment and management are the responsibility of community relations practitioners at most mining operations. However, there is a need for mining engineering professionals to be familiar with such perspectives because efficient management needs integration across all aspects of the operation. SIA and impact management are most effective where carried out in all the life cycle of mining, including all of the activities from exploration, construction, extraction, and processing, through to post-closure, as well as also incorporating recycling and waste management. The diverse social impacts across the mine life cycle stages and the extraction and resource processing phases require a complete rank of approaches to assessment and management.

7.7.1 General Overview of SIA

Social impact assessment involves «the processes of analyzing, monitoring, and managing the social consequences, both positive and negative, of planned interventions (policies, programs, plans, projects) and any social change processes invoked by those interventions.» Its first goal is to achieve a more sustainable and equitable biophysical and human environment (Vanclay 2003). Contemporary SIA arguably began along with EIA in the early 1970s in response to the formal requirements of the National Environmental Policy Act (NEPA) 1969 of the USA. The first SIA document was the publication in 1994 of the «Guidelines and Principles for Social Impact Assessment» by the US Interorganizational Committee.

A milestone case in the establishment of SIA was the inquiry at 1974 by Chief Justice Thomas Berger into the suggested Mackenzie Valley gas pipeline from the Beaufort Sea to Edmonton (Alberta, Canada). It was the first occasion that social impacts had been formally taken into account in project decision-making. The SIA finally recommended that the project be postponed for at least 10 years to enable sufficient time for land claims to be settled and for new programs to help the native population. «The findings were, at the time, unprecedented and marked the start of a huge growth in SIA» (Joyce and MacFarlane 2002). Today, many institutions and national governments consider SIA as a mandatory activity for project proposals.

Other procedures related to social impact assessment are health impact assessment (HIA) and strategic environmental assessment (SEA). In most EIAs, HIA is usually included under SIA. HIA is a wide concept that implies an interest in the safeguarding and improvement of human health. Regarding SEA, it is carried out much earlier in the decision-making process than EIA, being thus a key tool for sustainable development. SEA aims to include environmental and sustainability aspects into strategic decision-making procedures such as the formulation of policies, plans, and programs.

7.7.2 SIA for Mining Projects

Large-scale mining projects can generate different and intense social impacts. They can differ significantly based on the duration of the project, the position of populated areas related to the project area, and the potential mine expansion planning. Most EIA guidelines require social impact analysis. This implies that specialists in several fields are involved in planning, implementation, and monitoring throughout the mining project life. In this sense, it is essential to take into account the social impacts of mining on the surrounding environment and affected communities and to include social impact assessment into the operational activities of a mine as a management tool.

The social impact assessment should consider baseline information related to at least the four following areas: (1) changes in access to and power over local resources (land, water); (2) changes in the characteristics of a population (size, composition, traditions, productive activities); (3) divergent perceptions between decision-makers, the mining company, and local people about the distribution of economic benefits and social/environmental costs of a large mining operation; and (4) land property and use (ELAW 2010). For instance, relocation of a population is a vital social problem. To resolve this issue, environmental impact assessment must incorporate detailed information about compensation, relocation planning, and information about consideration to guarantee people similar quality of life.

Phases of a Mining Social Impact Assessment

A list of sequential steps should be followed in the SIA process, drawn primarily from the environmental impact assessment (EIA) steps (Arce-Gomez et al. 2015). Thus, Franks (2011) affirmed that mining social impact assessment can include a number of distinct but iterative phases within an adaptive management process (◘ Fig. 7.31): (1) scoping and formulation of alternatives, (2) profiling and baseline studies, (3) predictive assessment and revision of alternatives, (4) management strategies to avoid and mitigate negative social impacts and enhance positive impacts, (5) monitoring and reporting, and (6) evaluation and review.

The scoping stage establishes the criteria for the further stages of assessment and management by determining the scale, timing, and focus of the assessment, establishing who is likely to be impacted and detecting the actions that are likely to result in impacts. In this stage, alternative possibilities must be defined for further studies and a first evaluation of the impacts of these alternatives carried out. The output of this phase can be to consider the aim, scope, scale, priority issues, and terms of reference for the following phases of assessment and management.

The second stage includes understanding the communities and stakeholders potentially affected

Fig. 7.31 Phases of social impact assessment within an iterative adaptive management process (Franks 2011)

by the activity through social and economic research. Profiling includes studies of the social and economic features of an area at a given point of time. In turn, baselines are an evaluation of the state of a community before a mining activity occurs. Thus, baseline information must generate a clear description of present social conditions in the area potentially impacted by the project before it is realized.

Regarding predictive assessment and revision of alternatives, the outcomes of predictive assessment are generally prioritized by their scale and level of significance. They are utilized to offer feedback to stakeholders and project developers with the aim of modifying and revising the project. They allow them to make the decision to which suggested project alternative best accomplish the goals of the project while still improving social outcomes and preventing negative impacts. Different scenarios for the project design might be significant to describe apart from describing the zero alternative where the possible consequences are explained if the mining project is not finally developed.

The monitoring and reporting stage includes collection, analysis, and dissemination of information through time. As a rule, a well-defined monitoring plan shall include:

1. Outline of the monitoring methodologies to be applied to measure progress.
2. Baseline information on which progress can be measured.
3. Well-defined indicators for each program and identified impacts in the SIA: the indicators can be quantitative or qualitative, and they shall be of scientific quality.
4. Frequency: while baseline information provides a picture of the present situation, explicit and verifiable parameters are needed in order to assess the progress made (BMP 2009).

The final phase, evaluation and review, evaluates and reviews both assessment and management processes. The reconciliation of impacts estimated in the assessment stage with the actual impacts undergone during implementation will contribute to refine and enhance future perspectives. A well-defined evaluation plan shall include an outline of the evaluation methodologies to be applied and a plan of action for handling the outcome of the evaluations.

7.8 Reclamation Case Studies

Sanquelim Iron Ore Mine Reclamation (Goa, India): Courtesy of Vedanta

The Sanquelim group of mines is located in the North Goa District of Goa State (India) covering an

Fig. 7.32 Afforestation using cinnamon (Image courtesy of Vedanta)

area of 203 Ha. Sanquelim group of mines were operated since 1956–1957 for production of iron ore and subgrade ore. The ore deposit was broken in six pits. The mines have been reclaimed since the 1980s with environmental considerations and community infrastructure requirements. Where major mining operations were discontinued in late 1990s, there were no legislations in place for a systematic mine closure planning. However, the company proactively carried out systematic and scientific mine closure plan. The reclamation activities mainly comprised three main aspects: extensive afforestation, conversion of some parts of the pits into water bodies to harvest rainwater, and utilization of existing building infrastructure for benefit of community.

Afforestation

The total area of mine leases is 203 Ha, out of which *about* 105 Ha has been efficiently restored by afforestation. The open-pits were consistently backfilled by constituting benches making it viable for carrying out plantation. Company has planted more than 750,000 saplings on the Sanquelim iron ore mine. Initially, most of the areas were covered by planting fast growing plants like *Acacia auriculiformis* and *Casuarina equisetifolia*. These species were mainly planted as nurse crop to prevent erosion on dumps and stabilize the dumps. The company also tried growing cashew plants based on their experience at Orasso Dongor mine (Sesa's First mine).

After the dumps were stabilized, company selected one of the reclaimed mine pits to experience with diverse afforestation methods utilizing native horticulture and forest species. Most of the horticulture crops growing in Goa (e.g., mango, banana, guava, pineapple, cinnamon; Fig. 7.32) were planted with success. Use of leguminous cover crop like plumeria seeds was used to sow in the areas under acacia and eucalyptus plantation. The plumeria creeper grew luxuriantly over the trees and over the time killed the acacia and eucalyptus plantations naturally, thus making the stabilized land available for plantation of native species.

Pisciculture

Along with afforestation, a major part of mine pits were also retained or converted into water bodies by harvesting rainwater. In order to add value to the water bodies, the company approached the National Institute of Oceanography to establish the option of cultivating freshwater fishes in the

Fig. 7.33 Butterfly park (Image courtesy of Vedanta)

open-pit filled with rainwater. The pisciculture project was taken up in 1990, and, as a result, the cultivation of fish was successfully carried out, and the pit was abundant with freshwater fish like rohu, katla, common carp, etc. The project has also resulted in increase in bird and butterfly activity in the area.

Reclaiming the Old Building Infrastructure

The reclamation was not restricted to growing of plants but also for the old infrastructure like buildings and workshops that were put to productive use for communities. The mine workshop was converted into a technical school imparting education to local youth, and the residential quarters were converted into the football academy to cater to the needs of the local community.

Biodiversity Management Plan

After the results of previous reclamation methods used in one of the mine pits, it was decided to bring the other areas within the Sanquelim mine also under biodiversity plantations, and hence a Sanquelim Mine Management Plan was prepared. In consultation with forest department, the mature acacia plantations were proposed and to plant various native species to improve the biodiversity of the areas. Under this plan, various projects such as a medicinal garden, a butterfly park, and a bamboo serum were conceptualized and implemented.

Two medicinal gardens, namely, Nakshatra Vatika and Charak Vatika based on constellation, zodiac signs, and Ayurveda, respectively, were developed. The idea behind it was to spread awareness among the locals and school students about medicinal plants growing in our surroundings and its benefits. Each plant is identified and provided with other details such as medicinal value. Regarding the butterfly park (Fig. 7.33), in its life cycle, the butterflies require two types of plants to survive, namely, host plant and nectar plant. If both types are available, they naturally attract the butterflies. Various plants identified as host and nectar plants (flowering plants) were planted on the mine site with the aim to attract butterflies. This has added the beauty of the area increasing also the biodiversity.

Bamboo is one of the frontrunners of environmental rejuvenation, being the quickest growing species among all other woody plants. Its capability to rejuvenate itself without being planted is most advantageous because cutting bamboo

Fig. 7.34 Bamboo Pavilion (Image courtesy of Vedanta)

encourages the growth of new ones; they grow to its fullest height in 60 days compared to the woody plants which take 60 years. With an aim to promote bamboo cultivation, various species of bamboo were collected from across India. More than 25 varieties of bamboos have been grown and each of them was identified. Further to support the cause of bamboo promotion, a huge structure made out of locally available bamboo was constructed. This Bamboo Pavilion (Fig. 7.34) is used as an exhibition hall cum training center for self-help groups.

In order to assess the reclamation status, it is very important to regularly carry out various biodiversity studies. In a recent study, it was observed that there are different species of mammals, birds, butterflies, insects, reptiles, and amphibians in the restored mine zone. This shows that the biodiversity of the area has increased significantly.

Cooljarloo Heavy Mineral Sand Mine Reclamation (Perth, Australia): Courtesy of Tronox Ltd.

The Cooljarloo heavy mineral deposit that lies within the Perth Basin in Australia contains ilmenite, rutile, and zircon, which were produced from igneous and metamorphic rocks in the adjacent Archaean shield, separated in near-shore sediments through different stages of weathering. As the mineralization is extracted, overburden and sands with a small content of valuable minerals are backfilled into the void, clay waste is pumped to solar drying cells, and the surface is recontoured to look like the original landscape, previous to re-spreading topsoil and seeding for reclamation.

Thus, the objective of rehabilitation at Cooljarloo is to establish safe and stable landform (Fig. 7.35) capable of supporting a sustainable native ecosystem similar to that existing in adjacent areas of unallocated Crown land. To meet this objective, rehabilitation standards have been established that indicate how each component of the rehabilitation cycle will be implemented. The standards have been developed over many years of trial and error and now seem to be producing quality results in the field. Outlined below is a broad overview of some of the aspects contained within the standards.

Subsoil Reconstruction

The upper soil profile is formed with a layer of coarse to medium-grained sands, referred to as Class 1 material, which provide suitable conditions

Fig. 7.35 Tailings landforming (Image courtesy of Tronox Ltd.)

for vegetation establishment. The lower soil profile is usually formed with sands containing a higher clay content to assist in water retention within the root zone (referred to as Class 2 materials). Watercourses are constructed during landforming to ensure appropriate surface water flow across the site is maintained. Infiltration embankments are constructed to increase water retention and minimize water erosion on slopes.

Topsoil Placement

Topsoil is stripped ahead of mine path at the time of mining and stockpiled for use where the area is rehabilitated. Topsoil contains a good source of seed that becomes less viable as the stockpile ages. A portion (10%) of freshly stripped topsoil with older stockpiled topsoil is currently blended on all rehabilitation areas, which increases the amount of viable seed that is distributed at the time of topsoil placement.

Topsoil Stabilization

The topsoil must be stabilized to prevent wind and water erosion. This is done by spreading native mulch and sowing a cover crop of oats. Native mulch is a blend of freshly cut native vegetation harvested on mine path and tub ground woody material that is windrowed during clearing (Fig. 7.36). As fresh harvested mulch is becoming depleted, resource alternatives are being trialed, including the use of Terolas, an emulsion that binds the topsoil together while allowing infiltration of water and germination of seedlings. This product will be used in conjunction with tub ground and fresh harvested material over areas on Dam 8.

Native Seed Distribution

Native seed picked by the Billinue Aboriginal Community (Fig. 7.37) and from seed suppliers is spread over all native rehabilitation areas. Several vegetation groups are established into the rehabilitation at Cooljarloo. Each group corresponds with a particular landform characteristic and requires a particular mix of seed. All seed purchased is of local provenance ensuring similar species are grown as the surrounding UCL. The 2010 rehabilitation season will involve the reintroduction of the DRF *Andersonia gracilis* at Site 16 within the Falcon tenement via the return of fresh topsoil stripped from the original plant populations.

Rehabilitation Monitoring

Rehabilitation monitoring is conducted annually to assess the quality of rehabilitation against a set of performance targets, known as Completion Criteria, for various aspects for rehabilitation (e.g., species richness or landform stability). Reporting

Fig. 7.36 Tub grinding (Image courtesy of Tronox)

Fig. 7.37 Billinue seed collection (Image courtesy of Tronox Ltd.)

performance against Completion Criteria gives confidence that the rehabilitation will meet the objectives and not be a lingering liability for the company or the state.

Jabiluka Uranium Mine Reclamation (Jabiru, Australia): Courtesy of Energy Resources of Australia

Jabiluka exploitation is situated within the Alligator Rivers area and is about 230 km east of Darwin and 20 km north of Jabiru, being this area a major uranium-bearing zone. The mine site of Jabiluka and associated facilities are within the Jabiluka Lease surrounded by the Kakadu National Park. Principal land utilization for this region includes national park, fishing, Aboriginal traditional uses, and mining. Work began on rehabilitating the disturbed land on the Jabiluka Mineral Lease in 2003 where the surface and subsurface facilities were dismantled and the open-pit and decline were backfilled.

Revegetation

Revegetation is an essential part of the progressive reclamation activities that have been taking place at Jabiluka. Revegetation of the disturbed areas at the Jabiluka footprint took place in three stages over a decade. The Energy Resources of Australia (ERA) formed a strategic partnership with a local indigenous supplier Kakadu Native Plants, which raised saplings from seeds collected within the lease area. Traditional owners consulted on native species, density, and landforms, and several company indigenous trainees and workers were incorporated in the planting of saplings. Revegetation of disturbed areas began in 2005 with the planting of 7560 local native seedlings, being the seeds from native species collected and germinated. As of February 2014, 36,000 individual tube stocks had been planted within the Jabiluka mine site footprint with survival of 48% noted during the June 2014 routine periodic inspection. In 2015 the final phase of the revegetation project at Jabiluka was completed.

Water Management Pond Rehabilitation

In 2013, ERA committed to rehabilitating the Interim Water Management Pond. The program of work included removal of the pond liner and concrete spillway, relocation of waste rock stockpiles, regrading of fill surfaces, and excavation work. The land was reshaped and recontoured so that it was similar to the previous landform. Erosion matting and rock drainages were used to monitor erosion. The removal of the Interim Water Management Pond was completed in October 2013. Works go ahead to revegetate the primer area of the Management Pond. Thus, during 2014 a further 4678 native tube stock trees were planted at the landformed pond site. ◘ Figure 7.38 shows the evolution of the pond from 2011 to 2016, after complete rehabilitation.

Jabiluka is actually under long-term care and maintenance. Current weed, fire, and water quality management is in place at Jabiluka, incorporating monitoring in the care and maintenance stage that is regularly reviewed. Regarding weed control, in 2015 ERA embraced a qualitative estimation to weed management with the aim of evaluating trends in weed management zones, backed by regular on-ground observations. Thus, weed management operations are controlled by land management approach that addresses priority species such as annual *Pennisetum*, mission grass, and rattlepod. In-field weed monitoring shows that a progressive reduction of weed had been produced.

Ekati Diamond Mine Reclamation (Northwest Territories, Canada): Courtesy of Dominion Diamond Corporation

The Ekati mine site is situated in the Lac de Gras region of the Northwest Territories, about 250 km northeast of Yellowknife (Canada). The Ekati Diamond mine (named after the Tlicho word meaning «fat lake») (◘ Fig. 5.29) is the first surface and underground diamond mine in Canada. It is a remote mine accessible only by air and by winter road for 2–3 months of the year. The company understands the importance of reclaiming the Ekati mine site so that it can be returned to a viable northern environment at the end of operations. The goal of reclamation is to keep the site safe for human and wildlife use. This involves arranging rocks and plant life in a variety of patterns to determine which pattern allows the vegetation to grow best and which offers protection from erosion. Thus, the use of vegetation in the final cover system will enable a more economical cover design and also blends itself into the natural tundra landscape. Similar to the sections of a garden, the test areas have been seeded with various configurations of native grasses to develop an initial ground cover.

2011 2013 2016

Fig. 7.38 Evolution of Water Management Pond rehabilitation from 2011 to 2016

Fine processed kimberlite is discharged as a slurry to Long Lake Containment Facility (LLCF) and Beartooth open-pit. The overall reclamation goal for the LLCF is the design and construction of a long-term cover that will physically stabilize the processed kimberlite with a landscape that will be safe for human and wildlife use. For this purpose, it is essential to define a combination of vegetation and rock cover system to physically stabilize the processed kimberlite. Vegetation is planned to be the main stabilization component. Rock placement is intended to promote a localized environment for vegetation growth and provide larger-scale wind and water erosion protection (Fig. 7.39). Short-term focus of reclamation research has been to establish and evaluate vegetation growth directly within processed kimberlite including natural colonization, vegetation/rock plots, annual cover crop trials, plant species trials, soil amendments, and plant tissue analysis.

Since 2004, native northern alkali goose grass has been naturally colonizing on the east side of the LLCF. Rate of vegetation colonization is annually analyzed. Historical natural colonization and establishment in research areas indicates high potential for goose grass. Regarding annual cover crop trials, temporary ground cover until permanent vegetation is established. Control erosion provides microniches for colonizing plants and adds organic matter to the soil environment. In addition, there have been some investigations into the plant species best adapted for revegetation of the processed kimberlite in the LLCF. Preliminary investigations indicate that revegetation with grass has been effective and initial monitoring has indicated specific species that are better adapted to the conditions of LLCF.

Recent soil chemistry test results on new kimberlite in LLCF have indicated elevated sodium concentrations and pH when compared to older processed kimberlite test results. Elevated sodium results in increase in the sodium adsorption ratio (SAR). Thus, elevated SAR values can make it difficult for plants to obtain other essential nutrients due to competition from the excessive sodium. For this reason, in 2013 small field-scale trials were constructed to evaluate the potential of lowering the SAR through the additions of chemical amendments such as alfalfa pellets, gypsum, and/or calcium nitrate.

In summary, the main 2013 to 2014 LLCF reclamation research and monitoring was the following.

Fig. 7.39 2013 goose grass in the boulder field (July 2014) (Image courtesy of Dominion Diamond Corporation)

Vegetation/Rock Plots

Two areas totaling approximately 7 ha were seeded in the fall of 2013. In the winter of 2014, rock was placed in four configurations within the seeded areas. In 2014, vegetation monitoring of the rock/vegetation plots consisted of determining the survival rate of the tussock cotton grass seedlings and collection of field observations by walking through and photographing the site. Additionally, baseline soil chemistry data were collected within the rock plots. Samples were obtained from 0 to 15 cm and 15 to 30 cm from upper, middle, and lower slope locations in each of the three rock pattern areas. The vegetation rock plots were observed in June 2014 following freshet to gain insight into surface water flow. The data suggest that the most important factor affecting tussock cotton grass seedling survival is competition from other plants and is not directly related to the rock pattern, except perhaps indirectly through its influence on vegetation growth.

Plant Species Trials

Historically, the primary role of vegetation in mitigating the effects of disturbances was to provide a readily established ground cover to control surface erosion. To achieve that end, the use of commercially available agronomic grasses was common and widespread. With this objective in mind, plant species trials using native grass cultivars, locally harvested native plant propagules, and combinations thereof have been an important component of reclamation research at Ekati. In 2013, a trial involving eight grass species and one native legume was established. The plant species being investigated were slender wheatgrass, slough grass, tufted hair grass, fall rye, reflexed locoweed, bluejoint reed grass, creeping red fescue, spike trisetum, and Canada wild rye. Each species was hand seeded into a 6 m-long row and one row consisted of a mix of the grass species. With the exception of slough grass, all species have established and are doing reasonably well. Also in 2013, 40 mountain cranberry seedlings, grown from seed collected on site, were planted in the area colonized naturally by goose grass.

In 2014, 563 seedlings grown from seeds collected on the mine site were planted. Of those 563 seedlings, 210 were tall water sedge, 60 were short water sedge, 180 were tussock cotton grass, and 113 were nodding cotton grass. Various numbers of those seedlings were planted at 11 different

Fig. 7.40 Typical wet tundra species planting scenarios in LLCF: 30 tall water sedge and 35 nodding cotton grass seedlings at location EK1 as planted June 28, 2014 (Image courtesy of Dominion Diamond Corporation)

locations. Each of those species has its own habitat requirements, but in general all would normally be found growing on moist to wet tundra or in shallow water (Fig. 7.40). Where seedlings were counted, their height was measured, and percent ground cover was estimated along each row in the plot. In 2014, only seedlings in each row were counted (selective grazing by sik siks had affected many of the plants, rendering height and percent ground cover inapplicable). The cranberry seedlings planted in 2013 were monitored by determining the size of every living shrub.

Annual Cover Crop Trials

In 2013, eight grass species, one native legume, and 40 mountain cranberry seedlings were planted. In 2014, transects were established in each of the treatment areas, percent ground cover was estimated, and plant stems were counted. In this year, a total of 563 seedlings grown from seed collected on the mine site were planted in various locations across LLCF. The seeding operation was conducted in stages. First, the surface of the area to be seeded was loosened by pulling a weighted chain harrow across it with a track-mounted side-by-side ATV. Seed was then broadcast at 100 kg/ha using a large tire spreader pulled behind the ATV. A final pass with the chain harrow and a roller was conducted to incorporate the seed and improve seed to soil contact.

In 2014 and 2015 (Fig. 7.41), annual cover crops were planted over 18 ha in part of the LLCF. These hectares were seeded with barley and fall rye cover crops. Species trials include test growth of native grass cultivars and native plant seed and/or seedlings directly within processed kimberlite. In 2015, 15 new species of seedlings and seeds were planted LLCF. Monitoring results suggest barley is better adapted than fall rye.

Soil Amendment Trials

A small trial plot area was constructed in 2013 to test the effectiveness of various soil amendments in modifying the elevated sodium levels in PK. Gypsum and calcium nitrate were applied at 1.5 tons/ha, alfalfa pellets at 10 tons/ha. In 2013 and 2014, soil chemistry from the samples col-

Fig. 7.41 Seeding annual cover crops (July 2014): harrow and roller following broadcast seeding (Image courtesy of Dominion Diamond Corporation)

lected at 0–20 cm depth was analyzed, and vegetation was monitored by measuring percent ground cover, counting seedlings, and measuring average plant heights. Barley and alkali grass performed similarly in all treatments in 2013, except the two with calcium nitrate, and growth was most successful in the untreated PK. In 2014, vegetation growth in the unamended PK was poorer than in the treated areas.

Glacial Till Topdressing

Another possible solution to address the new PK chemical properties is the use of glacial till as a topdressing over the PK. The objectives of the glacial till topdressing study are to assess the suitability of till as a capping material over PK, thereby providing a better plant growth medium than the uncapped material. The suitability of glacial till as a reclamation substrate has been assessed at several locations on the mine site. In 2013, two small trail test pad areas were constructed using till as a topdressing material. One test pad was with till over PK, and one was with till over Coarse Kimberlite Rejects (CKR). Over the long term, plant growth on till has been satisfactory, but establishment is impeded by the hard surface crust that develops upon its drying. That condition can be ameliorated, however, by roughening or ridging the surface by deep ripping.

Natural Colonization

Over the long term, a key measure of revegetation success at Ekati will be the proportion of ground cover comprising indigenous native plant species. Therefore, creating conditions that encourage colonization of disturbed areas by local native plants is an important objective. Conversely, the unlikely spreading of non-native annual crops would not be desirable. In 2014, site investigations were completed for any evidence of vegetation colonization. Baseline vegetation growth was established by analyzing satellite imagery data using the normalized difference vegetation index (NDVI). The output data were consolidated into three categories with the following results: (a) 81% is PK and does not have any overlying vegetation growth; (b) 12% is covered with vegetation that has been classified as lower biomass vegetation; the majority of this 12% is attributed to the «goose grass» that surrounds the tundra and the 2013 seeded alkali grasses; and (c) 7% is covered with vegetation that has been classified as higher biomass vegetation;

Fig. 7.42 Geese grazing on the barley annual crop (Image courtesy of Dominion Diamond Corporation)

the majority of this vegetation is attributed to uncovered natural tundra that was not covered by PK during deposition activities.

Wildlife Observations

One closure objective for the final LLCF cover is ensuring its safety for wildlife use. In the short term, introduction of wildlife into the reclamation research areas has the potential to lead to positive benefits by initiating the nutrient cycle. Initial wildlife observations were collected during 2014 site visits. Large numbers of geese were attracted to the 2014 annual cover crop trials with substantial evidence of grazing (Fig. 7.42). Arctic hares and sik siks were also frequently seen eating young plants. Observation of grazing by wildlife was evident for all the research areas, most notably for the annual crops.

True North Gold Mine Reclamation (Fairbanks, Alaska, USA): Courtesy of Kinross Gold Corporation

The True North Gold Mine is within the Chatanika River watershed, about 26 miles northwest of Fairbanks Alaska. The region is vegetated with black spruce and surface moss that cover the north and east facing slopes. Because of the climatic conditions, reclamation commonly is carried out at summer months. The True North Mine reclamation process is planned to return the land disturbed by mining works to a stabilized, near-natural condition that will assure the long-term protection of land and water resources. Additional goals include minimizing or eliminating long-term management requests and matching state and federal regulatory requests. In this sense, True North offers a habitat to a wide range of Alaskan species such as moose, wolves, bear, and birds of prey. In 2010, Kinross set out to utilize indigenous seedlings. After establishing that no seedlings are present, the company created a greenhouse to observe if indigenous seedlings could be grown. The selected species were black spruce, white spruce, birch, and alder.

Reclamation has occurred in the following phases, with some overlap: (a) interim reclamation to stabilize and maintain viability of topsoil and growth media stockpiles were completed during and directly after construction; (b) previously disturbed areas including historic exploration trenches, abandoned roads, and exploration drill pads that were not affected by current mining operations were concurrently reclaimed; (c) final contouring occurred upon final cessation of mining operations; and (d) vegetation, slope stability, and water quality monitoring will continue until all reclamation performance standards are achieved.

General Reclamation Procedures

The primary reclamation components of the True North Reclamation Plan included grading and recontouring, storm water conveyance channel construction, growth media placement, seedbed preparation, fertilizing, seeding, and monitoring. Waste rock dumps required major grading, contouring, and possible growth media application. Other disturbed areas were revegetated and some required regrading. Growth media were be applied on all waste rock dumps and areas that require it to successfully achieve a 70% cover. Waste rock dumps were configured to establish drainage and avoid swales and depressions.

Storm water drainage channels were constructed where deemed necessary during recontouring to minimize potential soil erosion while vegetation is reestablishing. Temporary control devices were removed where the site-specific potential for erosion had been minimized through earthwork or revegetation. There were sound reasons to continue maintenance of some control structures depending on final recreational use and other types of use. Growth media were stockpiled at True North in anticipation of future reclamation needs. Approximately 6 inches of growth media were applied generally to those sites requiring additional growth media to be revegetated or to promote natural reinvasion by native plant species. However, application depth varied depending upon the facility. Roads, trails, stock pads, and building sites required little, if any, growth media. Once the implementation of growth media is possible, the specific site was designed for seeding by ripping on the contour to roughen the surface. The goal of preparing the seedbed in this fashion promoted revegetation and enhance evapotranspiration. Prepared seedbeds were fertilized prior to, after, or during the seeding operation. Final fertilizer and application rates considered information acquired from previous reclamation efforts.

Regarding the grass seed mix used, the first aim of this seed mix was to obtain fast vegetative cover that helped to diminish soil erosion and promote succession back to climax vegetation. The seed mix may change over time in response to such factors as internal and external research results, changes in technology, changes in land management philosophy, and commercial availability. Native species were the preferred mix. In some instances, mulch was to be found useful in conserving moisture, moderating soil temperatures, and improving erosion control. The practice of scarifying the seedbed on the contour prior to seeding minimized the potential for erosion. Mulch was evaluated if seed germination becomes a limiting factor in the reestablishment of vegetation. Seeding was conducted as soon as possible following seedbed preparation. Generally, seeding was implemented after spring break up until mid-July. Such seeding allowed the seed to take advantage of the summer moisture period. However, if a seeding was unsuccessful for any reason, the area was reseeded the following year.

Waste Rock Dumps

During the summer of 2005, the identified disturbance in dumps was regraded and ripped with dozers, seeded, and fertilized, including this work different dumps. Seed and fertilizer were used on all restored disturbance utilizing either a broadcaster mounted on a dozer or by aerial application with a fixed wing aircraft (Fig. 7.43). Reclamation consisted of scarifying or ripping of the graded surface on contours apart that created a broken, roughened surface to trap moisture, reduce wind shear, and minimize surface erosion by increasing infiltration of the top surface of the soil, which in turn created micro-habitats conducive to seed germination and development. Figure 7.44a shows North Shepard Waste Rock Dump in 2005, prior reclamation, and Fig. 7.44b the same Waste Rock Dump after reclamation in 2013.

Subsequent to this restoration work, a part of the North Shepard Dump slumped and needed further earthworks and reseeding/fertilizing. The slump area encompassed approximately 7 acres. This problem was corrected in early 2010 by picking up the slide material and backfilling the North Central Pit along the south and west edge of the pit. The material to backfilling was pushed at a 3:1 slope for final grading. The slump area was excavated to a 2.5:1 or shallower slope or until natural ground was found.

Pits

Pits developed during mining have been backfilled. The three remaining pits account for 125 acres of disturbance. The North Central Pit was partially backfilled in 2007 to decrease surface water pooling. The Hindenburg Pit floor was graded and scarified to decrease potential runoff to the North Central Pit. Both of these pits

Fig. 7.43 Aerial seeding via fixed wing aircraft (Image courtesy of Kinross Gold Corporation)

received seed and fertilizer in 2007 to promote revegetation to reduce surface water flow. Due to work within the pits and continued water flow, the North Central Pit and Hindenburg Pit were regraded and scarified once again in August 2010. To prevent water flow, the diversion ditch above the pit was cleaned out and reconstructed to divert water toward the Shepard Road.

Ambatovy Nickel Mine Reclamation (Antananarivo, Madagascar): Courtesy of Sherritt International Corporation

Ambatovy is a large-scale nickel and cobalt mining situated 80 km east of Antananarivo (the capital of Madagascar) near the town of Moramanga. The mine operates since 2010 as an open-pit mining and a processing plant. From the mine, the slurried laterite mineralization is sent via pipeline of approximately 220 km in length to a preparation plant and refinery situated south of the Port of Toamasina. The estimated life of the operation is approximately 29 years. Since ultramafic rocks present in Ambatovy mine are highly unstable in a tropical weathering environment, the mine presents a deep weathering alteration, with a complete lateritic profile capped by a ferruginous duricrust. Thus, the ore deposit is a typical nickel laterite in which enrichment has occurred in the residual soils formed by tropical weathering of ultramafic bedrock. Prolonged weathering has produced a thick mature laterite profile in which the nickel grades have been enriched from the levels seen in the underlying bedrock.

The Ambatovy mine (Fig. 7.45) lies in a high-biodiversity area at the southern tip of a large section of remnant eastern rainforest corridor. From a reclamation viewpoint, it is essential to bear in mind that Madagascar is a global hotspot for biodiversity, with very high-degrees of endemism, and, at the same time, a high-level of threat. The most important impacts on biodiversity will carry out at the mine site and along the upper section of the pipeline.

Environmental management at the property is adaptive and consists of applying the mitigation hierarchy, which includes impact avoidance, minimization, and, where necessary, compensation or offsetting by regular monitoring of the physical and biological environment. Physical environmental monitoring includes water quality (total suspended solids and other parameters), air quality

■ **Fig. 7.44** **a** North Shepard Waste Rock Dump in 2005 prior reclamation (Image courtesy of Kinross Gold Corporation); **b** North Shepard Waste Rock Dump after reclamation in 2013 (Image courtesy of Kinross Gold Corporation)

Fig. 7.45 Ambatovy mine and rainforest (Image courtesy of Sherritt International Corporation)

(dust and other parameters), and meteorological monitoring. Biological monitoring includes monitoring the populations and health of affected lemurs, small mammals, birds, reptiles, amphibians, and fish. Biological management actions of the mitigation hierarchy include defining clearly and minimizing the mine footprint, slow directional clearing of forest (accompanied by the salvage and relocation of plant species of concern and the less mobile vertebrate animals), and establishment and management of conservation zones or offsets. The offsets include about 3300 ha of forest surrounding the footprint, two set-aside parcels of azonal forest amounting to approximately 300 ha growing over part of the ore body and active support to regional conservation initiatives.

Mine Site

As a consequence of the estimated high residual impacts to biodiversity and the significance of azonal and transitional areas to supporting rare plants and fauna within the mine area, a comprehensive on- and off-site mitigation plan to preserve key habitat elements is proposed. At the mine site, sedimentation dams are constructed to prevent release of sediments from the mining area into local watercourses. A total of seven dams will be built, three of which have been completed. The dams are equipped with spillways as discharge structures.

The progressive mine site reclamation will be carried out through erosion monitoring, reforestation with certain species, and facilitated secondary successions. Several test plots have already been launched to establish the optimal floral species succession composition for the soil matrix once mining works are finished. The main goal is to generate a rehabilitation process for the mine site. It is necessary to comment that the process will incorporate specific ecological aspects such as the selection of the flora species that can favor species recolonization of the reclaimed pit zones. Throughout 2012, Ambatovy worked to conserve forests around the mine footprint and to prepare for reclamation of the footprint itself. Construction of a research and production nursery was completed, with a capacity to produce over 250,000 plants annually and equipped with a poly-tunnel and other experimental facilities to determine the optimal cultivation methods and conditions for successful plant production.

Fig. 7.46 Aerial view of slurry pipeline route in tavy zone (Image courtesy of Sherritt International Corporation)

Pipeline

The pipeline is mainly buried and the elected route made significant deviations, including tunneling, to prevent affecting forest fragments, cultural sites, and local habitations. The dominant vegetation type along the route is tavy (85%) (Fig. 7.46), areas disturbed by the traditional slash and burn technique used to clear brush and forest for crop production and comprising cleared forest and scattered shrubby vegetation or trees. The second most typical vegetation class is degraded primary forest (4%) comprising either heavily logged forest or very small forest patches that have been invaded by exotic (alien) plant species. For most of its length, the 220 km pipeline was buried using standard «cut and cover» construction at an average depth of 1,5 m. In areas of unspoiled forest and important rivers, Ambatovy drilled horizontally below the surface, leaving stretches of forest intact and allowing the pipeline to pass safely below the river courses.

Regarding the rehabilitation of the pipeline servitude, the implementation of erosion control structures commenced during 2008 with about one million linear meters of fascines (comprise scrub material collected selectively from surrounding areas, rolled into bundles, and tied with banana leaves) and vetiver grass hedgerows positioned on and below the fill embankment slopes of the pipeline servitude and right-of-way, as additional sediment control measures. This work was carried out over the full extent of the fill slopes of the servitude or right-of-way (ROW). In 2010, 550 ha of sparsely covered areas of the platform were hydroseeded and fill slope areas between the fascines and vetiver hedgerows. The seed was sourced from advance seed, and the seed mix was approved for use in the hydroseeding process which was done at 50 kg/ha along the servitude. The mix was selected to provide temporary cover until native species are able to establish over time. Lime and organics were added to the hydroseeding mixture.

Queen Copper Mine Reclamation (Bisbee, USA): Courtesy of Freeport-McMoRan

The porphyry copper deposits of Arizona are located in what is known as the «basin and range» physiographic province. This region is characterized by a series of fault bound blocks that have risen

Fig. 7.47 The former Crawford mill and diesel power plant area after reclamation (Image courtesy of FreePort-McMoRan)

and fallen creating a distinctive valley-mountain range topography. Critically as this landscape formed, they tilted exposing the lower crustal levels where porphyry copper deposits form. The deposits are all copper dominant with subsidiary molybdenum mineralization and unusually with very little precious metals. They are Late Cretaceous to Early Tertiary in age. Mineralization is high-grade copper sulfides (chalcopyrite and bornite) with minor lead and zinc carbonates in irregular replacement ore bodies. Ore control was nearby dykes and sills with associated brecciation. Alteration was gossan with Mn and Fe oxides induced by hydrothermal metamorphism.

At Bisbee, underground mining started in 1880 and kept to until all activities ceased in 1975. In the early 1900s, it was the most productive copper mine in Arizona. Once open-pit mining started in 1954, rock stockpiles were constructed, and tailings from diverse milling works were constituted. Since 2006, several restoration projects have been carried out to mitigate zones impacted by mining activities. These projects encompass approximately 500 ha. Part of this effort includes restoration studies at different stockpiles in the Bisbee area. These stockpiles were always sources of acid rock drainage during summer and winter storm events. The project included grading, capping, and replanting vegetation with the aim of enhancing visual impact, removing acid drainage, and creating wildlife habitat. About three millions of cubic meters of material were translated to recontour and cover the South Bisbee stockpile.

In 2011, restoration works started on the Bisbee area tailings dams and adjacent installations. The tailings program reclamation comprises the North and South tailings dams (Fig. 7.47), the Crawford mill concrete sub-structures and diesel power plant (Fig. 7.48), and the Horseshoe Basin. As part of that reclamation procedure, the company regraded the side and top surfaces of both dams and covered them with about 70 cm of clean material to efficiently control storm water and assure that it is discharged in a manner that originates replacement of the local watershed.

In this sense, fast liberation of these large storm water runoff flows is critical for restored tailings dams to decrease the volume of water that

Fig. 7.48 Reclamation of North (*right*) and South (*left*) tailings impoundments as of 2013 (Image courtesy of FreePort-MacMoRan)

can infiltrate into the tailings material becoming long-term seepage that must be gathered. In order to match runoff and infiltration objectives, past tailings restoration programs needed important quantities of new material to be placed with the aim of providing positive drainage from the top surface toward off-site conveyance.

The approach for the Bisbee tailings dam restoration plan was to prepare design ideas that would reduce regrading of the top surfaces with imported material while decreasing the necessity for conveyance structures needed to control peak storm water runoff flows. The Bisbee concept uses the present grading of the dam and the big top surface area to capture storm water runoff from certain areas of the dam and decrease the peak flow as it is conveyed off the top of the dam into an off-site conveyance. This new concept attenuated peak flows by nearly ten times. Moreover, basins are created to generate low net infiltration of precipitation. The basins are planned not only to decrease large flows but also to capture low flows. These attenuation basins work in essence as «engineered playas.»

Mina Fe Uranium Mine Reclamation (Salamanca, Spain): Courtesy of ENUSA INDUSTRIAS AVANZADAS S.A.

Uranium was discovered in Salamanca during the 1950s. Mine production started in 1974 at ENUSA's Mina Fe mine, an open-pit mine that increases to become the biggest uranium exploitation in Spain that produced over 4000 t of uranium. The mine closed in 2000 because of the low prices of the metal. A full decommissioning plan began in 2001, and the mining areas, including one large pit, three small ones, and four waste dumps, have since been restored. Also, a small uranium plant and heap leaching have been dismantled. The Mina Fe uranium-ore deposit is located about 10 km northeast of Ciudad Rodrigo (Salamanca, Spain). Regarding the geological setting, the rocks consist, mainly, of low metamorphosed carbonaceous pelitic and fine-grained psammitic rocks, in which sedimentary textures are commonly preserved, interlayered with carbonate metric beds. The igneous rocks (granodiorites) intrude the stratigraphic sequence generating a contact metamorphism. The Tertiary Alpine orogeny produced further fractures as well as the rejuvenation of older ones, forming some time important cataclastic breccia zones. Primary uranium deposits (uraninite + carbonates + pyrite + adularia) occur in fault-related rocks.

At Mina Fe, the Elefante plant was mainly a bacterial heap leach installation that was substituted by the Quercus mill in 1993. This plant utilized a combination of heap and dynamic leach until 2000. Quercus metallurgical process was based in a grading of the crushed mineral, in a humid atmosphere, to obtain three fractions of differing size and grade: the coarsest was classified as waste, intermediate was heap-leached (static lixiviation with sulfuric acid), and finest was leached in mixing tanks (dynamic lixiviation), also with sulfuric acid. After a back washing in thickener classifiers and a clarification of the fertile liquids obtained from previous stages, recuperation, concentration, and purification of the uranium contained were carried out. It was performed through a process of extraction with an organic dissolvent and re-extraction of pH controlled with ammonium sulfate. The following steps were precipitation of the uranium contained in the watery extract in ammoniac diuranate, drying the humid concentrate, and packaging the product in drums. The process terminated with the neutralization of the sterile effluents, sending these solutions and the neutralized pulps to the waste dam, and conditioning of effluents.

7

Reclamation Process

ENUSA began in 2001 to reclaim the forming uranium mining operations and to dismantle the uranium plants. The objective of this restoration (one of the most important in Europe) was to recuperate the affected natural space with environmental and radiological conditions as similar as possible to those existing before mining works. Actually, these environmental reclamation activities are focused to control the dismantled radioactive installations and the restored mining works, and on chemical processing of polluted waters, which is required until the adequate quality for discharge to public waterways is carried out. The main criteria used to achieve the aforementioned objective were:

1. Ensuring the containment and stability of the contaminated structures for long term
2. Creating new structures without active maintenance, integrated in the environment
3. Protecting water resources (surficial and groundwater)
4. Limiting dust and radon emissions according to future land uses
5. Applying ALARA acronym for (As Low As Reasonably Achievable) criteria (a principle for radiological protection, minimizing radiation doses, and releases of radioactive materials by employing all reasonable methods)

The main works of reclamation of Mina Fe and ancillary installations were located in: (a) open pit mines, 15 Mm^3 (with about 2.7 Mm^3 of acid mine waters); (b) waste rock piles (schists), 35 Mm^3; (c) spent ore piles, > 4 Mm^3; (d) tailings dams, > 1 Mm^3; and (e) metallurgical plants: Elefante and Quercus plants.

The reclamation process was developed in different projects due to their different nature and structures involved.

Uranium Plants (Elefante and Quercus) Decommissioning

The main activities in dismantling operations of Elefante plant were carried out between 2001 and 2004 and included:

1. In situ stabilization by leveling tops and slopes and extending beds of spent ore piles from heap leaching: 7.2 Mt (60 ha).
2. Dismantling of industrial plant: wastes were stored in a containment enclosure, built under reconfigured spent ore piles and capping beds.
3. Capping of a multilayer cover for land restoration (Fig. 7.49), formed from bottom to top by (1) 0.9 m of clayey arkoses to minimize water infiltration and to attenuate radon gas emission, (2) 0.9 m of rip-rap (selected rock waste of low grade) to prevent erosion of the clayey layer, and (3) 0.5 m of top soil to allow planting of vegetation and reinforce the action of preceding layers.
4. Technical and radiological controls.

Concerning the evolution of the reclamation process, first the spent ore piles (more than 7 Mt, in the shape of a truncated pyramid with slopes near 75%) were reconfigured to a final structure having new slopes around 20%. The original surface 24 ha was converted to 56 ha of reclaimed terrain, being located the containment enclosure and the tailings dams, from the plant processing, under the restored ore piles. The whole set then was covered with the aforementioned 2.3 m multilayer cover, formed by almost 3 Mt of the

Fig. 7.49 Multilayer cover (Image courtesy of ENUSA INDUSTRIAS AVANZADAS S.A.)

cited materials. There is a monitoring and control program since 2006 to verify the compliance with limits for decommissioning.

Regarding Quercus processing plant, it is pending of decommissioning approval. Due to problems related to acid mine drainages (AMD), it is necessary to maintain some structures, as the tailings dam and big ponds, to collect acid waters as well as the plant for chemical treatment (neutralization process), working until the water quality allows its discharge directly to the river. For this reason, the process will be carried out in phases. Dismantling of the plant will be undertaken firstly, including the associated spent ore piles. The plan will include the building of a containment enclosure and a multilayer cover, as in the case of Elefante plant. At present, there is a maintenance and control program to the beginning of the dismantling process.

Open-Pit Mines and Waste Rock Pile Reclamation

Open-pit mines and associated waste rock dumps affected 250 ha. The main activities in open-pit mine reclamation from 2004 to 2008 were:

1. Geomorphological restoration by filling of open-pits with waste rock from dumps and/or in situ stabilization (Fig. 7.50).
2. Capping of multilayer cover for land restoration; it was formed from the bottom by (1) 0.3 m of clayey arkoses to minimize water infiltration and to attenuate radon gas emission and radiation, (2) 0.3 m of rip-rap (selected rock waste) to prevent erosion of the lower layer, and (3) 0.3 m of top soil to allow planting of vegetation and reinforce the action of preceding layers.
3. Land revegetation.
4. Water management plan, including collection, treatment, drainage, discharge, analytic and radiological controls, etc.
5. Technical and radiological controls. Since 2014, there is a monitoring and control program, including groundwater management and stability of created structures, to check the compliance of the restoration objectives.

The first activity was the most important from a visual impact viewpoint. Total amount of removed

Fig. 7.50 Open-pit at 2004 **a** and 2016 **b** (Images courtesy of ENUSA INDUSTRIAS AVANZADAS S.A.)

waste rock was about 20 Mm^3. Regarding the multilayer cover for environmental and radiological protection, it was prepared in a similar way for Elefante plant. The only difference was the total thickness, since the cover was globally 1 m less than in Elefante plant, because of the different radiological natures of the rocks to cover, mine waste rock instead of spent ore rock, and, therefore, different radiological activities. Revegetation process includes seeding and planting native species, such as herbaceous and

Fig. 7.51 Evaporation units (Image courtesy of ENUSA INDUSTRIAS AVANZADAS S.A.)

bush plants, but not trees to prevent their roots drill the clayey layer, covering more than 250 ha. Water management was carried out creating new channels for best drainage of waters as well as some minor dams for water temporary storage.

Acid Mine Drainage

To control acid mine drainage from the mine, it is necessary to collect approximately 500,000 m^3 of water per year for chemical neutralization process in two neutralization plants. This is in order to guarantee appropriate quality, according to required parameters, before the controlled discharge of the water to the river. In order to try to minimize this important problem, different actions were carried out, including (a) sugar beet carbonate foam amendments; (b) improvement of revegetation; (c) waterproof of filtering dams; (d) in situ stabilization actions, protecting gullies, repairing multilayer covers, etc.; and (e) incorporating new enhanced evaporation units (Fig. 7.51).

Their effectiveness has been only partial or temporary. This is the reason that a new plan for AMD remediation, based in artificial soils («tecnosoles») application technics, is being tested, including edaphic studies, chemical and radiological determinations, etc. They are obtained with inert residues, not toxic or hazardous, and are designed, made, and used à la carte. Artificial soils can solve or reduce the specific problems of affected mining exploitations and/or meet the needs for the restoration of contaminated soils. Previous successful results in the recovery of contaminated major sites were located in mining sites, such as the coal mine of As Pontes (La Coruña); waste dumps, open-pits, and waters in the sulfide mine of Touro (La Coruña); soils of the Guadiamar River Valley (Sevilla) contaminated by the failure of the Aznalcóllar sulfide tailings dam and the sludge discharge; and others. The artificial soils are spread in thin beds on the ground as surficial deposit or in areas with backwaters to create a reactive wetland.

7.9 Question

Short Questions

- Explain the final step in the operation of a mine.
- Define the concept of community used in minerals industry.
- Explain the concept of rehabilitation in mining reclamation.
- What is an Environmental Management System?

- Define sustainable development.
- What are the Equator Principles?
- List the main types of solid mine waste.
- What is biodiversity? Explain the relationship between mining and biodiversity.
- What are the main measures that can be implemented to control and minimize subsidence damage?
- List the potential contributors to visual impacts in mining.
- What are the main advantages of revegetation in reclaiming mined lands?
- What biosolids means?
- What is a social impact?
- Explain briefly the environmental impact assessment process.
- What is the main method for the identification of effects and impacts in an environmental impact assessment?

Long Questions
- Describe in detail the definition and importance of a «social license to operate.»
- Explain the acid mine drainage formation.

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7

Mining Software

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Summary

The use of computers to process geological data as well as to design the mine has become a boom in mineral industry. After an initial introduction about inexpensive software, ► Chap. 8 considers some examples of commercial mining software: RockWorks and Datamine. RockWorks is a comprehensive software with modeling of spatial and subsurface data as the main tool. It was developed by RockWare at the beginning of 1990s and is probably the most important software worldwide for geological utilities. On the other hand, Datamine provides a range of integrated mining solutions for the all the processes involved in mining development.

8.1 Introduction

Computers are an essential tool to conduct mineral inventory studies and a significant amount of software exists that has been developed specifically. Thus, the use of computers to process geological data as well as to design the mine has become a boom in mineral industry. The continued progress over the last 25 years allowed the mining software to run from the simplest to the most complicated options. The software is continuously updating to fulfill the needs of the customers and users. To carry out mineral inventory studies without the use of these computing facilities is actually a huge mistake since software procedures form part of modern mineral inventory practice, being impossible to obtain similar results making calculations by hand.

Most resource calculations carried out in exploration, in pre-feasibility or feasibility studies, or in grade control and scheduling utilize a specific software package that handle 3-D data. At the early stages of exploration, the main features of the software will be to input borehole information and link this information to the surface features. Where a resource or reserve is being estimated, the capability of the package to model the shape of geological units and calculate volumes and tonnages becomes essential. It is essential that the user is very specialized about the subject, data input, and the possible result expected at the end of the processing.

Mining software is used in a wide variety of applications: (1) databases to record geological data coming from diverse sources such as surveys, drillholes, geochemical analysis, geological structures, rock mass behavior, processing and production costs, etc.; (2) statistical analysis to manipulate and summarize the very large and complex data sets that are typical of exploration and mining operations; (3) a wide variety of estimation tools for creating spatially continuous models of geological structures; (4) computer-aided design (CAD) facilities that are specialized for accurate and efficient modelling of geological structures and mine openings; and (5) an array of specialized algorithms that serve as aid for engineering design (Smith 1999).

8.2 Types of Mining Software

Available software can be classified as public domain and commercial. As a rule, public domain software is free and incorporate full source code, but it does not usually include guarantees or technical support. This type of software is commonly forthcoming from universities, government organizations, textbooks, technical literature, and diverse user groups accessible through Internet. On the contrary, commercial software, although more expensive, generally comes with an up-to-date user's manual and also including a technical support system from the company. However, the source code is often generally proprietary. The software packages are typically integrated, that is, the different capabilities are available in modules that can be purchased separately. Obviously, the package selected must depend on the finance accessible and the needs of the users at any given moment since there are modules specifically devoted to open pit mining, underground mining, and so on.

8.2.1 Inexpensive Software

Inexpensive software includes from freeware or public domain software to software included in a book. An example of the first is GEOEAS tools. Regarding the software programs included in a book, CSMine software is delivered with the book entitled *Open Pit Mine: Planning and Design*, third edition (2013).

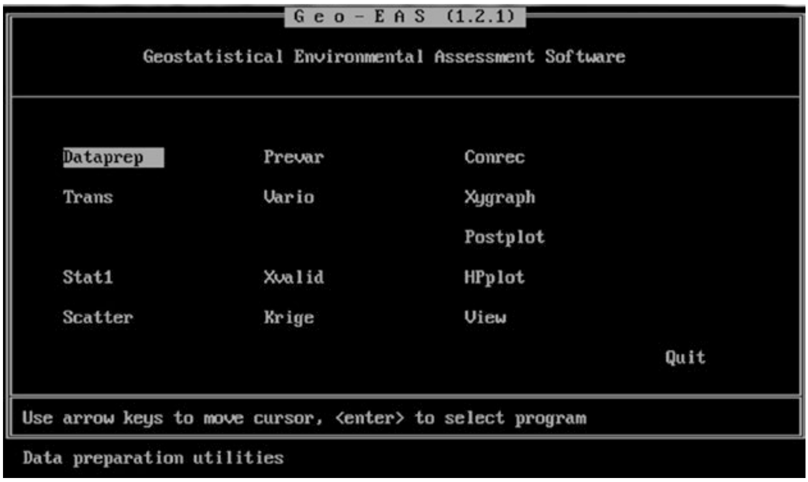

Fig. 8.1 Main program window including the different options (GEOEAS)

GEOEAS

Geostatistical Environmental Assessment Software (GEOEAS) was developed by the Environmental Protection Agency at 1991, so this system was designed to run under DOS and consequently works badly in modern equipment, most of them using 64-bit architecture. However, it is possible to run properly the program using emulation software such as DoxBox. GEOEAS is probably the most famous public domain geostatistical software in the history, being a suite of tools for carrying out two-dimensional geostatistical analysis of spatially distributed data. GEOEAS can produce data maps, univariate statistics, scatter plots/linear regression, and variogram computation and model fitting (Fig. 8.1), but the main function of the package is the generation of grids and contour maps estimates obtained by interpolation using kriging methods.

Once the drillhole database has been selected, GEOEAS provides a Stat1 program for univariate exploratory data analysis. It is essential to have a look at the histogram of the data and to maintain it in mind during the different processes since variogram modeling and kriging estimation are very susceptible to the presence of extreme values. Then, the Vario program is selected. The main goal of Vario is to develop a mathematical model of the spatial continuity of a variable (variogram) adjusting the model to the experimental semivariogram. Later, Xvalid control the goodness-of-fit of the model selected. Finally, Krige program carry out 2-D estimation using kriging.

CSMine

The main emphasis of the program is on open pit mine planning because the book in which the program is included is mainly devoted to this topic. The program is designed to take raw drillhole data through the block modeling process to the generation of the final economic pit limits. The major features of the program include (1) graphical displays of drillhole data: drillhole plan maps and drillhole section or profile maps; (2) compositing of raw drillhole data to regularly spaced samples for processing by the block model; (3) block modeling by the inverse distance squared method or kriging; it includes graphical presentation of the block model data (block plots and contour plots through any bench or section); (4) assigning of economic values to the blocks and their graphical presentation; and (5) final pit limit generation including geometric pit limits defined by the surface topography and pit slope constraints and economic pit limits defined by a three-dimensional floating cone algorithm. Thus, the program is divided into three modules to deal with the three types of data. These are (a) the drillhole mode, used to read in and display the raw drillhole data; (b) the composite mode, used to regularize the raw drillhole data into composites of equal length; and (c) the block mode, used to create and display the block model, to assign economic values to the blocks, and to generate the final economic pit limits.

Table 8.1 The main mining software vendors, including their programs and web pages, arranged alphabetically

Company	Software	Web page
Aranz Geo Limited	LeapFrog	► www.aranzgeo.com
Constellation Software Inc.	Datamine	► www.dataminesoftware.com
GEOVIA	Minex, Whittle, Surpac, Gems, and others	► www.geovia.com
Hexagon	MineSight	► www.hexagon.com
Maptek Pty Ltd.	Vulcan, Eureka, and others	► www.maptek.com
Micromine Pty Ltd.	Micromine, PitRam, and others	► www.micromine.com
MineMap Pty Ltd.	MineMap IMS	► www.minemap.com.au
Minemax Pty Ltd.	Minemax	► www.minemax.com
MineRP	Mine2-4D, CADSMine, and others	► www.minerp.com
Phinar Software	X10-Geo	► www.phinarsoftware.com
RungePincockMinarco	Xpac, Coal seam AGG, and others	► www.rpmglobal.com
ThreeDify Inc.	Geomodeler, FlowPit, and others	► www.threedify.com

8.2.2 Commercial Software

There is a number of software packages of this type in the market. They can be classified into two main groups: (1) software with applications in mining and (2) specific mining software. The former are programs with mining applications such as Surfer or RockWorks, although RockWorks is more specifically delivered as geological utilities software. Below are very brief descriptions of an example of each group: RockWorks and Datamine, respectively.

Regarding specific mining software, there are several packages available in the market (Table 8.1). Many of the companies were founded in the late 1980s, but a range of acquisitions in the following decades change dramatically the market. For instance, three of the most popular softwares in those times such as Whittle, Gemcom, and Surpac are now property of GEOVIA. On the other hand, companies such as Micromine or Maptek remain the same. Whittle is an exciting example of this acquisitions trend (Box 8.1: Whittle Software History).

Box 8.1

Whittle Software History

Jeff Whittle became involved with the mining industry in 1979. In 1984, Jeff Whittle and his wife Ruth Whittle established the Whittle Programming Company in Melbourne (Australia), focusing on mine optimization software and its application to strategic mine planning. At that moment, Jeff Whittle developed the Whittle 3D pit optimization product based on the Lerchs and Grossman algorithm (see ► Chap. 5). Thus, Whittle 3D was released in 1985 revolutionizing the optimization of surface mines around the world. Consequently, Jeff Whittle is the man who has made an essential impact on the mining industry in pioneering strategic mine planning. His innovative thinking has made an impact on the majority of mining companies and mining professionals involved in the evaluation of mining deposits and the planning of mining operations.

Whittle 3D estimated using the cited algorithm the optimal shape for an open pit mine. Successively, in 1987 Whittle 4D was released incorporating time, risk, and optimizing with net present value, which allowed sensitivity analysis for long-term planning. Whittle 4D estimated a nested set of 40–100 optimal pits based on potential economic scenarios. Thus, the user could select the best option based on a great variety of cutoff, scheduling, scaling, and timing options within a financial analysis framework. The program was very good on sensitivity analysis and how the pit would change with different metal prices, but also as a good technique for designing early pit phases and pushbacks within the ultimate pit. In 1995, Opti-cut was released to optimize cutoff grades over the life of the mine. This program was inspired by Ken Lane's theories on cutoff grade optimization (see ► Chap. 4). Furthermore, Opti-cut produced life of mine cutoff, stockpile strategy, tonnages, and cash flows. Hereafter, Whittle Four-X was released in 1997 with the purpose of handling multiple elements within the same orebody.

By 2000, the Whittle mine optimization products where used by over 400 companies in 48 countries around the world. This amount does not include the many mining consultants who used the Whittle product on their client's projects. Thus, Whittle had become the industry's most trusted strategic mine planning software solution, and some banks commonly asked if a mining project had been «Whittled.» The Whittle software package is currently considered as the premier open pit optimizer and is used as a benchmark for mining studies throughout the world. Companies depend on Whittle to help them determine their investment strategy and to deliver robust mine plans that maximize profitability by taking into account real mining constraints. Whittle Programming was sold in January 2002 to Gemcom Software International, which is now known as GEOVIA following its acquisition by Dassault Systèmes.

Some common characteristics of this type of software are: (a) They offer integrated tools for modeling, estimation, mine design, optimization, and scheduling; (b) they are compatible with many third-party software applications including GIS, Google Earth, ALS CoreViewer, and various mining applications; (c) they can manage and visualize very large and complex data sets, process the information, and rapidly generate models; (d) they usually offer multilingual support: English, Spanish, French, etc.; (e) they are commonly organized in modules or separate programs, each one specifically devoted to the different parts of the mining process, that is, exploration database, geological and block modeling, resource estimation, mine design, and optimization and scheduling.

8.3 RockWorks

RockWorks is a comprehensive software with modeling of spatial and subsurface data as the main tool. It was developed by RockWare Company at the beginning of 1990s and is probably the most important software worldwide for geological utilities. The latest version is RockWorks 17. RockWorks applications include: (a) petroleum: well spotting, structural and isopach mapping, logs and cross sections, stratigraphic models and fences, production graphs; (b) environmental: borehole database for lithologic, stratigraphic, analytical data; point and contour maps, logs, cross sections, plume models; (c) mining: drillhole database for lithologic, assay, geophysical data; 2-D and 3-D log diagrams, block modeling, detailed volume tools; (d) geotechnical: borehole database for lithologic, geophysical, geotechnical data; logs, sections, surface/solid models, structural tools. Mining applications can be arranged in three levels: borehole database tools, mapping tools, and other tools such as block modeling and volume calculations.

RockWorks is organized in two main data windows: Borehole Manager and Utilities. The Borehole Manager includes a data window and a suite of menus to enter and work with borehole data. In this tool, most of the subsurface modeling and visualization in RockWorks (e.g., 2-D and 3-D logs, cross sections, solid and stratigraphic models, among others) is carried out. The Utilities data window is a simpler, row-and-column type of data window with its proper group of menus. It is possible to generate diverse types of maps, charts, and diagrams.

8.3.1 Borehole Manager

Borehole data include location, orientation, lithology, stratigraphy, colors, fractures, water levels, symbols, patterns, bitmaps, vectors, construction, production, and data and text information for each borehole. For instance, orientation information in each borehole includes depth, azimuth, and inclination in downhole survey points. Once the information for all the boreholes is located at the proper item, many options are present to visualize one or several boreholes, from single borehole to sections including several boreholes and from 2-D to 3-D visualization.

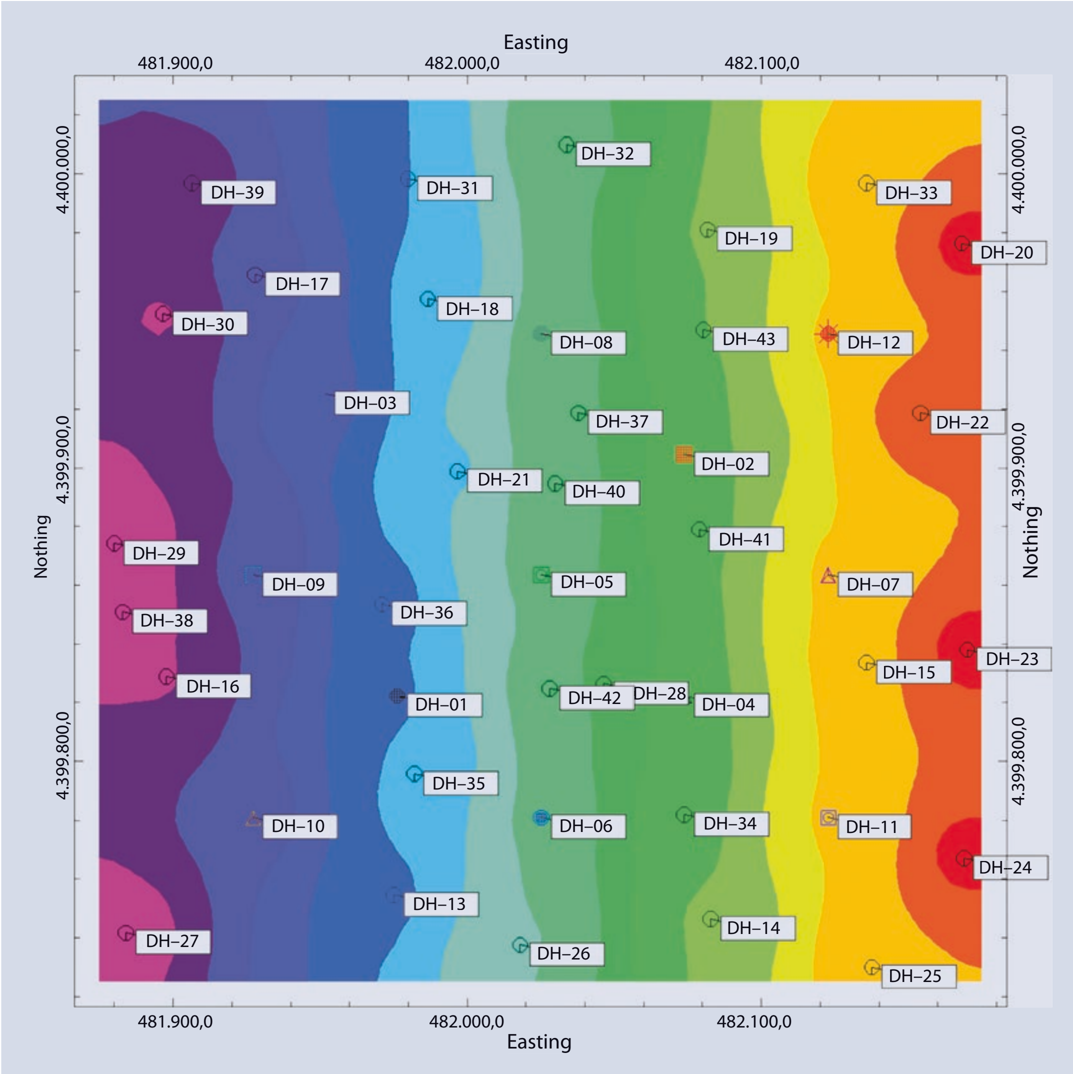

Fig. 8.2 Borehole locations with ground contours surface (RockWorks)

Figure 8.2 is an example of borehole locations with ground surface contours, and Fig. 8.3 is an example of 3-D visualization of the borehole lithology data.

Another interesting option in Borehole Manager is the Lithology/Model option. This program creates a 3-dimensional solid or block model representing interpolated lithology types and displays the model as a 3-D voxel diagram (Fig. 8.4). The lithologies will be represented in the model using the numeric «G-values» declared in the lithology types table. 3-D logs can be attached the image when needed. The completed voxel diagram will be showed in a RockPlot 3-D window where it is possible to handle the display, filtering specific values, and show volumes, among many others. Regarding the volume of each lithology, the Volumetrics option reads an existing lithologic solid model and creates a tabular report by computing the total volume and/or mass for each lithotype based on the relative depth/elevation. The estimations obtained are presented in a row-and-column datasheet.

For each type of data included in the borehole database, many options are available, for example, Plan Map and Histogram. Plan Map creates a 2-D map displaying the interpolated values where the

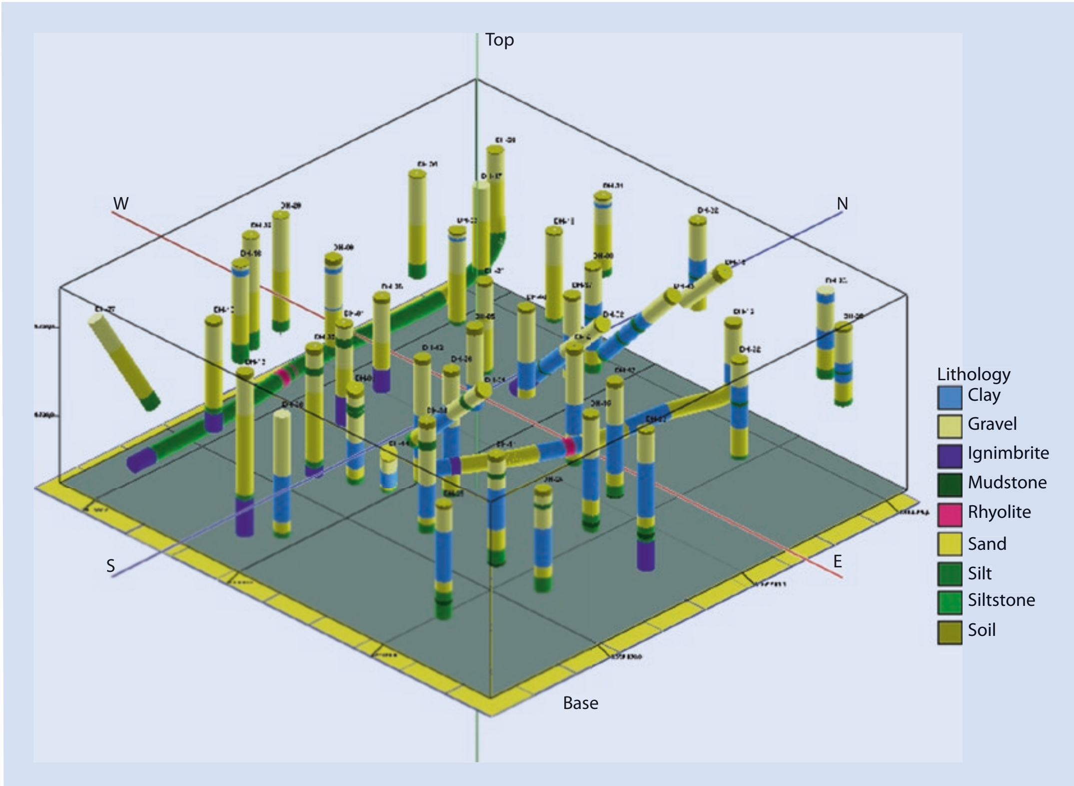

Fig. 8.3 3-D visualization of the borehole lithology data (RockWorks)

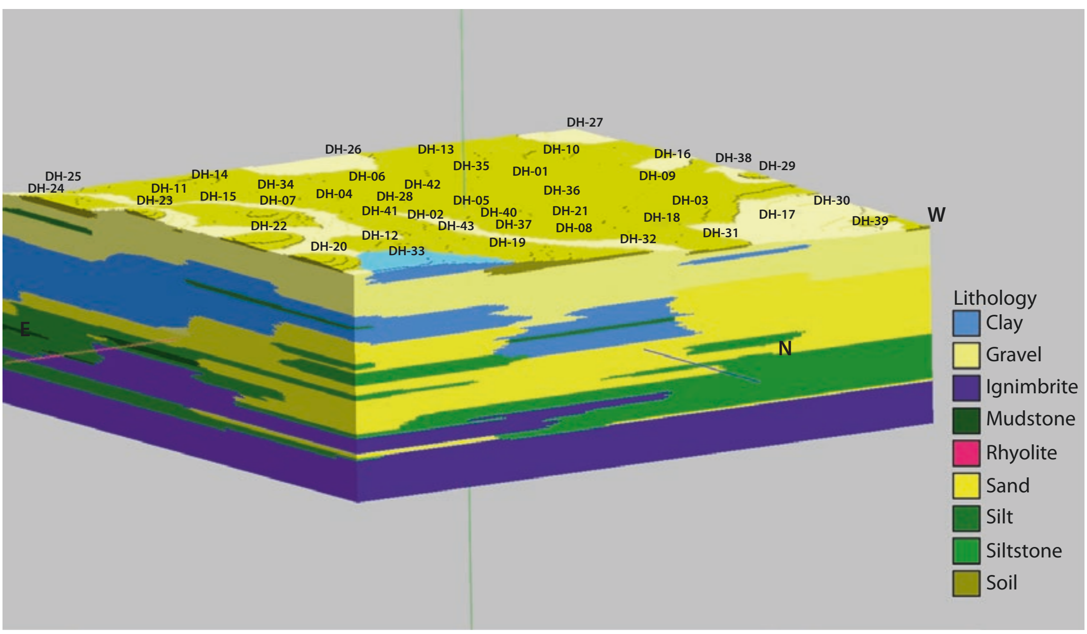

Fig. 8.4 3-D lithology block model (RockWorks)

Fig. 8.5 Grid model report (RockWorks)

```
Area/Volume:

   Center of Mass (x,y) .................. 482.025,149555 Meters , 4.399.875,091602 Meters
   Cell Area ............................. 25,0 Square Meters
   Map Area (X*Y) ........................ 90.000,0 Square Meters
   Grid Area (Sum(Cell Area))............. 93.025,0 Square Meters
   Model Volume (Sum(Cell Area*Z)) ........ 163.524.075,16018 Cubic Meters*
   Non-Zero node area .................... 93.025,0 Square Meters

Projection Information:

   XY (Horizontal) Units ................. UTM Meters
   Z (Vertical) Units .................... Meters

   UTM (Universal Transverse Mercator):
      Datum (Spheriod) ................... Datum = Airy 1830
      Zone ............................... Select UTM Zone
```

model intersects a horizontal plane; the completed map is showed in RockPlot 2-D. Histogram tool is used to read a single column of data from all boreholes and compute the frequency or percentage of the measures for that variable that is included in each grouping or cell previously defined by the user. The values then are displayed using a bar histogram plot.

8.3.2 Utilities

This module or group of programs includes many interesting applications in mining. In summary, four main menus are present: Map, Grid, Solid, and Volumetrics. The Map menu reads spatial data from the Datasheet Editor and generates a variety of different maps: points, contours, polygons, 3-D points, etc. The Grid menu manipulates grid models: statistics, filters, editing, imports, directional analysis, etc. The Solid menu creates, manipulates, and analyzes solid models. Finally, the Volumetrics menu computes volumes, creates extraction surfaces, creates GT reports, etc. In the Map menu, for example, the Grid-Based Map program reads XYZ data from the datasheet, interpolates a grid model, and creates a 2-D map with symbols, labels, grid-based line/color contours, triangle network, background image, and/or border annotation. 3-D surfaces are also created.

In the Grid menu, the polygon filter is utilized to set the grid nodes inside or outside a defined polygon to either a level specified by the user or to the values established in another grid model. A classical application of this tool is to set zero to all the values out of the polygon. For instance, if the input grid model depicts mineralization reserves and the polygon means a lease boundary, the grid statistics will compute mineralization reserves only within the lease boundary. In addition, Grid/Math carries out arithmetical operations with the grid node Z-values in two existing grid files. This is very useful to estimate the amount of raw material mined in a period (e.g., 1 week, 1 month, or 1 year) considering a grid model for the first day of the month and a grid model for the last day of the month. The subtracted grid will offer the amount cited. In each case, Grid/Statistics/Report option shows all the values of the grid model, including model volume, which represents the volume of the raw material included in the model (Fig. 8.5).

In the Solid menu, the Model program generates a solid model from X, Y, Z, and G data in the Utilities datasheet and creates a 3-D isosurface diagram or all-voxel diagram representing the solid model. RockWorks offers several different methods to interpolate the solid model and many different display settings. The values to be modeled (G values) can represent geochemical concentrations (e.g., grades in a mineral deposit), geophysical measurements, geotechnical parameters, etc. Figure 8.6 is an example of this type of model, representing lead concentrations displayed as an isosurface diagram with boreholes.

In the Volumetrics menu, two programs are highlighted: 2-D (grid model) and Extract Solid. The grid model is utilized for estimating formation volume from a column of thickness values in the datasheet and including different filtering parameters. The computations are grid-based, being the gridding algorithm (e.g., kriging, closest point, or inverse distance weighting) selected by the user. Some of the advanced filtering options contain thickness, stripping ratio, up to five quantitative data column range constraints, polygon areas, and distance. In addition, it is possible to invoke a polygon clipping filter so that only those thickness nodes within a user-entered polygon area are included in the computations. The output report can even list «Proven,» «Probable,» and «Inferred» reserves based on user declaration of distance confidences.

The Grid-Based Volume calculator offers several types of output: (a) a grid file containing the numeric model for the final (filtered) grid of formation

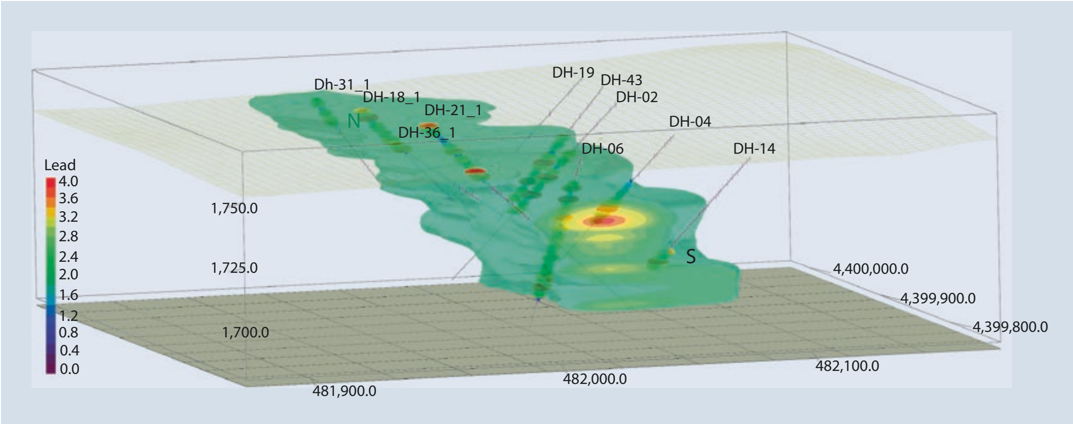

Fig. 8.6 Isosurface diagram including boreholes (RockWorks)

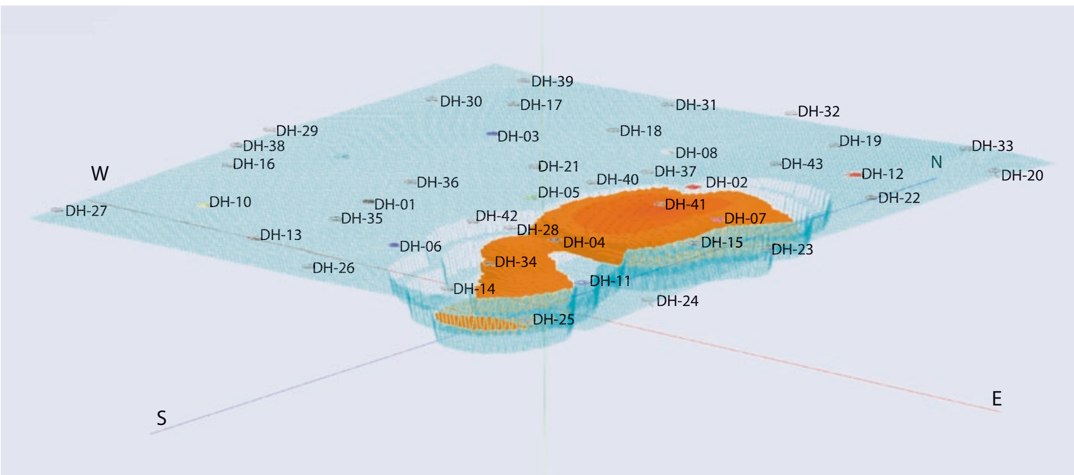

Fig. 8.7 3-D pit diagram (RockWorks)

thickness values (or mass values, if requested), (b) a 2- or 3-D map that illustrates the final thickness (or mass) grid, and/or (c) a report that lists in detail the intermediate and final volume computations (with weight computations, if requested).

Regarding the Extract Solid option, this program consider an existing solid model (e.g., lithology type) and establishes the volume of a pit that would be needed to extract the parts of previous solid model that fall within a defined range (e.g., a selected lithotype). The output is a report that includes the pit, the volumes, and the stripping ratios, a 2-D diagram showing the pit elevations, and/or a 3-D diagram illustrating the filtered solid and the pit elevations (Fig. 8.7). Obviously, the surface grid model must have the same dimensions as the X and Y dimensions of the input solid model. Maximum overall slope and bench heights can also be defined.

8.4 Datamine

Datamine provides a range of integrated mining solutions for the all the processes involved in mining development. It incorporates software tools for exploration fieldwork, resource modeling as well as all levels of mine planning, strategic optimization, detailed design, and short-term decision-making. Thus, the different programs can be grouped into four main categories: geological data management software, resource/reserve modeling software, open pit planning software, and underground planning software. Four of these programs,

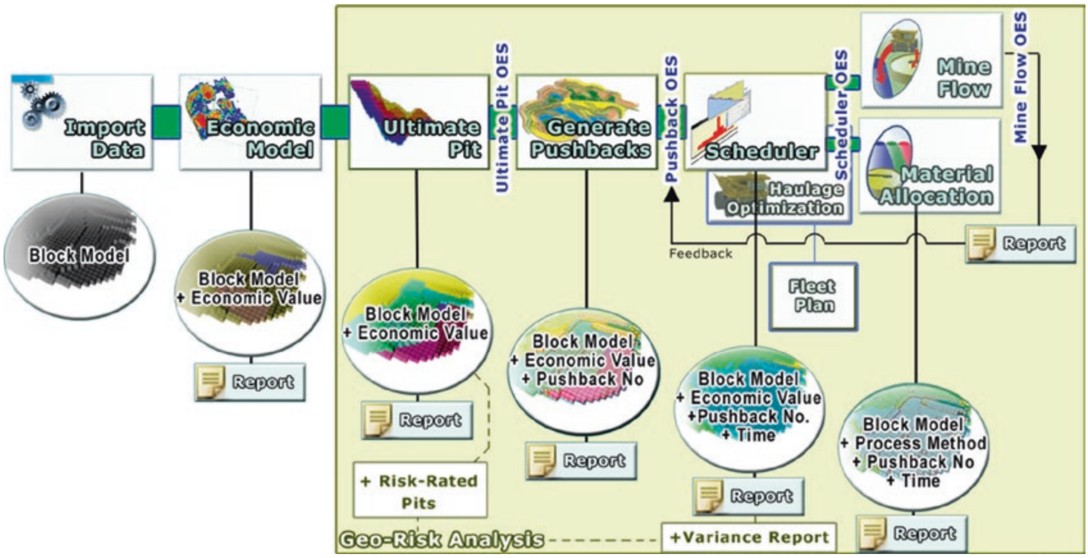

Fig. 8.8 Different applications of Datamine software

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DHLogger, Datamine Studio, Datamine Studio OP, and NPV Scheduler are briefly commented below.

Exploration data forms the fundamental underlying basis for resource models and mining resource/reserve evaluations, being critical for mining companies to carry out a protection process of the investment with modern systems that control all the processes and safely store the data. In this sense, Datamine provides a range of integrated tools for data capture, analysis, storage, and reporting of geological, geotechnical, and geochemical information. This software category includes programs such as DHLogger, Sample Station, Mine Mapper, Fusion, Report Manager, Laboratory Information Management System, and Core Shed Management.

Datamine's resource modeling system delivers geological models for all types of mines across the full range of deposit types. It includes programs such as Studio EM, Studio RM, Strat3D, and Ore Controller. Regarding the third group, Datamine provides a full range of open pit planning applications from strategic long-term optimization, pit design, reserve generation, and operational equipment scheduling incorporating programs such as NPV Scheduler, Summit Strategic Open Pit Planning, and Studio OP (Fig. 8.8). Finally, Datamine underground mining software helps to design, plan, and schedule all aspects of the underground mining. This category includes programs such as Mineable Shape Optimiser, Summit Underground Strategic Optimisation, Studio 5D Planner, Enhanced Production Scheduler, Aegis, and Ventsim.

8.4.1 Fusion/Geological Database Management System

The main purpose of Fusion software is to enable clients to collect, validate, manage, and deliver geological data for the project, improve the flow of geological data throughout an organization, and minimize the time required to work with the data at all levels. Their capabilities include (a) data collection (e.g., data import from many external data sources like .txt, .csv, MS Access; data validation ensuring that the data is correct; standardized pick lists; or automatic validation of laboratory results), (b) data management (e.g., captured data transferred to a central location), and (c) data delivery (e.g., share data with other users utilizing the Fusion applications or share data with external applications using custom export routines). Regarding data collection and validation applications, DHLogger software captures and manages drillhole data, the sample station module captures and manages point sample data, the MineMapper3D option captures and manages 2-D/3-D mapping data, and Century LIMS captures and manages mine lab data.

DHLogger

The main objectives of DHLogger are the following: (a) to complete drillhole data capture including collar, geological and technical details, samples, and related QA/QC materials; (b) to import analytical results using the Lab Import utility; (c) to import external geology data using the Drill Hole Import utility; (d) to convert grid coordinates from one system to another; (e) to calculate sample composite intervals; and (f) to manage dispatch of samples and drill costs.

In DHLogger, the data is entered into DHLogger through the use of three types of data entry screens: collar, details, and samples. The details view of the drillhole window is used to log major and minor interval data for the hole. Minor intervals must be associated to major intervals and typically describe small or less significant core properties such as alteration or minor lithology. All texture, structure, alteration, and mineralization records are generally linked to specific major and minor intervals. All RQD data, magnetic susceptibility, direction, coordinates, and wedge records are generally linked to the hole. If there are gaps in core, a «no core available» rock type or something similar must be created and used to represent such intervals in the details view.

The Samples interface contains all of the sample and assay data associated with a particular hole. It is possible to import drillhole data from a comma or tab delimited format into DHLogger in a two-step process: defining which tables data will be imported into and defining which columns data will be imported into. On the other hand, the worksheet of the Samples interface is used to calculate and save averages from a series of hole samples. Regarding data management, the Fusion Administrator option administers the Fusion data model, manages users, and defines validation constraints; the Fusion Client option transfers data between databases and synchronizes administration changes; and the Fusion Scheduler option transfers data from a remote site to a central location on a scheduled basis. For data delivery, QueryBuilder queries the database, exports results, manages QA/QC charts, and creates custom reports; and Crystal Report Viewer displays, prints, and saves custom reports. Finally, Report Manager creates automate QA/QC reporting.

8.4.2 Studio RM

Creating a geological and resource model is an iterative process with greater understanding of the geology and grade distribution, being achieved as the study proceeds and more data becomes available. The process of creating a 3-D geological ore body model typically makes use of the topography contours, drillholes, structural data (e.g., fault surface), ore body string model (section strings, top or bottom contact contours), ore body wireframe model, and waste and ore block models. The addition of user-defined alpha or numeric attributes (e.g., zone (mineralization zone number) or density (rock density)) and the use of data filters, views and formatting, symbols, and linestyles facilitate the geological modeling process and enable the generation of professional outputs such as summary reports, plots, and 3-D views. In general, the geological modeling process using a mining software includes the following steps: (a) importing drillhole data, (b) importing topography, (c) geological string modeling, (d) geological wireframe modeling, and (e) geological block modeling.

Importing Drillhole Data

Drillhole data is used as a basis for creating geological models. The drillhole tables typically consist of collars, downhole surveys, and downhole samples tables. The minimum field requirements for the tables are as follows: (a) collars: BHID, drillhole identifier; XCOLLAR, collar X coordinate; YCOLLAR, collar Y coordinate; and ZCOLLAR, collar elevation; (b) surveys: BHID, drillhole identifier; AT, downhole depth; BRG, bearing in degrees; and DIP, dip in degrees; (c) assays: BHID, drillhole identifier; FROM, downhole interval start depth; TO, downhole interval end depth; ASSAY1, first assay field, numeric values, units as defined by the user, for example, in g/t or percentage; and ASSAY2, second assay field; many assay fields are permitted, depending of the mining software; (d) depth data tables: BHID, drillhole identifier; AT, downhole depth; and ATTRIBUTE1, alpha or numeric attribute such as geological structural measurement or downhole geophysical survey parameter; (e) interval data tables: BHID, drillhole identifier; FROM, collar X coordinate; TO, collar Y coordinate; and ATTRIBUTE1, alpha or numeric attribute such as interpreted mineralization zone flag code or rock density. ◘ Figure 8.9 is an example of this type of drillhole data.

The drillhole data can be imported from other formats such as ASCII space-delimited format or Microsoft Excel worksheet format to generate the mining software (Datamine) format. Once the drillhole data is imported, a standard procedure for checking and correcting possible errors in the data is carried out immediately. It would be compared the listed errors against the relevant records in the source files (e.g., database, text files, or spreadsheet) and correct data entries where required. With the data imported in the Datamine table editor, different processes can be performed such as to locate the positions of the drillhole collars, to show the drillholes in the 3-D window, to define sections (◘ Fig. 8.10), and much more. Another possibility is to obtain the summary statistics of different data, including minimum, mean and maximum mineral grade values, for instance to investigate the parametric summary statistics and to check for outliers.

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Compositing down drillholes is a process that will be essential in successive steps. The composited drillholes will be (a) composited by rock type or domain by setting a very large interval to generate individual rock type composites to be used for rock type or domain boundary modeling and/or (b) composited by a fixed interval such as minimum mining width or block size for geostatistical analysis.

Importing Topography

Although the topography can be created using a number of points, the most common way to obtain the topography of the selected area is to import the topography contours data from an AutoCAD file and generate the Datamine format strings file. The CAD drawing file commonly has the data of polylines, which represent topography contours and a bounding perimeter (e.g., contour interval 10 m, elevation range 60–250 m, X coordinate range 5610–6780 m, and Y coordinate range 4600–5779 m). At this point, an integrated topography-drillholes view can be displayed in the 3-D window (◘ Fig. 8.11).

Geological String Modeling

The next step in the process is to create the basic framework for a geological ore body string model. It consists of sets of vertical section strings that are guided by the mineralized zones displayed in the drillhole data; the strings will then be saved to a Datamine file. The section perimeters (closed strings) are used to model the ore body where (a) drillhole data is organized in sections, (b) ore bodies have complex geometries (e.g., irregular shapes), and/or (c) needing to generate closed wireframe volumes. The interpretation of mineralization zones and the creation of geological string models for ore bodies can be done using a variety of string modeling methods such as vertical, horizontal, or inclined section perimeters (closed strings) and contour strings (e.g., separate top and bottom ore zone contact contours, surface topography).

As a rule, the perimeter is digitized in a clockwise direction. The start point is the extrapolated top of the upper zone position. Points are digitized on the top (top contact) or bottom (bottom contact) of the relevant drillhole segments, by pointing or snapping to segment ends. The string points are labeled with the digitizing sequence, and the string will be closed to create a closed string (perimeter). The same process is carried out in the different drillhole sections to create a string model. Therefore, the string model consists of sets of section strings that have been digitized, for example, in vertical N-S planes and spaced 25 m apart. The translation distances of the sections along a specific coordinate axis can be either positive or negative, that is, translation in the direction of increasing coordinate values (e.g., +25) or translation in the direction of decreasing coordinate values (e.g., −25). ◘ Figure 8.12 shows an image of the extended string model relative to the drillhole and topography surface contour data.

Then, tag strings are added to the existing geological ore body strings model. They are added in order to control the exact placement of wireframe edges and overcome the problem of twisted wireframes associated with complex geometries. The strings will link the northern and southern ends of the upper and lower mineralized zone strings between adjacent N-S sections that are spaced 25 m apart. Finally, the different string models (topography and ore body) are combined into a single object. It is useful for simplifying object management where a large number of string model objects are used and as a means of simplifying a data set for presentation purposes.

RECORD	BHID (A8)	FROM (N)	TO (N)	AU (N)	CU (N)	DENSITY (N)
1	VB2675	215.636902	217.886902	-	-	2.62
2	VB2675	217.886902	219.886902	-	-	2.65
3	VB2675	219.886902	221.886902	-	-	2.65
4	VB2675	221.886902	223.886902	-	-	2.66
5	VB2675	223.886902	225.886902	-	-	2.66
6	VB2675	225.886902	226.186996	-	-	2.66
7	VB2675	227.886902	229.886902	0.63	0.7	-
8	VB2675	229.886902	231.886902	0.56	0.6	-
9	VB2675	231.886902	233.886902	0.7	0.7	-
10	VB2675	233.886902	235.886902	0.42	0.5	-
11	VB2675	235.886902	237.886902	0.98	0.9	-
12	VB2675	237.886902	239.886902	2.45	1.9	2.8
13	VB2675	239.886902	241.886902	2.03	1.6	-
14	VB2675	241.886902	243.886902	2.38	1.9	2.8
15	VB2675	243.886902	245.886902	15.19	20.299999	4.09
16	VB2675	245.886902	247.886902	0.7	0.7	3.21
17	VB2675	247.886902	249.886902	1.06	1	3.85
18	VB2675	249.886902	251.886902	1.25	1.2	4.18
19	VB2675	251.886902	253.886902	1.41	1.3	4.25
20	VB2675	253.886902	255.886902	1.42	1.3	4.25
21	VB2675	255.886902	257.886993	0.63	0.7	3.22
22	VB2675	267.886993	269.886993	0.02	-	-
23	VB2675	321.886993	323.886993	0.13	-	2.76
24	VB2675	323.886993	324.937012	0.23	-	2.91
25	VB2675	324.937012	330.886993	-	-	2.93
26	VB2737	80.037277	138.837296	-	-	2.66
27	VB2737	138.837296	140.537292	0.31	0.5	-
28	VB2737	142.537292	144.537292	20.65	9.8	3.879
29	VB2737	144.537292	146.537292	4.41	2.4	3.01
30	VB2737	146.537292	148.537292	2.24	1.4	-
31	VB2737	148.537292	150.537292	4.34	2.5	3.04
32	VB2737	150.537292	152.537292	0.28	0.5	-
33	VB2737	152.537292	154.537292	1.89	1.3	-
34	VB2737	154.537292	156.537292	1.26	1	-
35	VB2737	156.537292	158.537292	0.42	0.5	-
36	VB2737	158.537292	160.537292	0.28	0.5	-
37	VB2737	160.537292	162.537292	2.66	1.8	-
38	VB2737	162.537292	164.537292	5.18	3.4	3.35
39	VB2737	164.537292	166.537292	0.42	0.5	-

Fig. 8.9 Drillhole data table (Datamine)

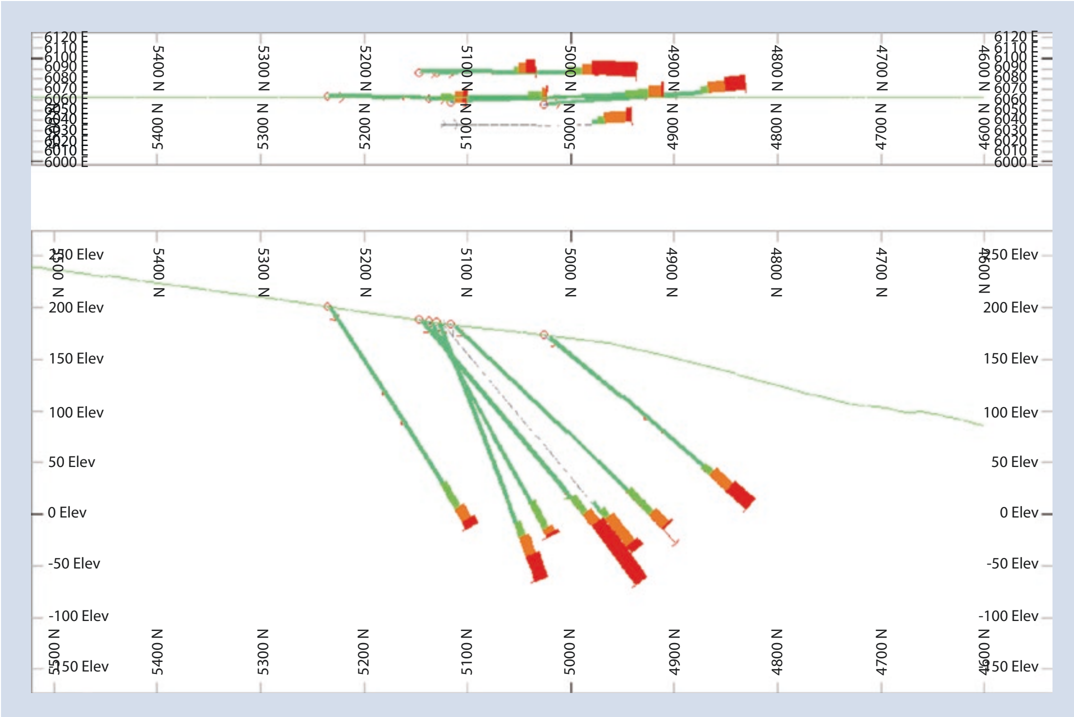

Fig. 8.10 Section including drillholes (Datamine)

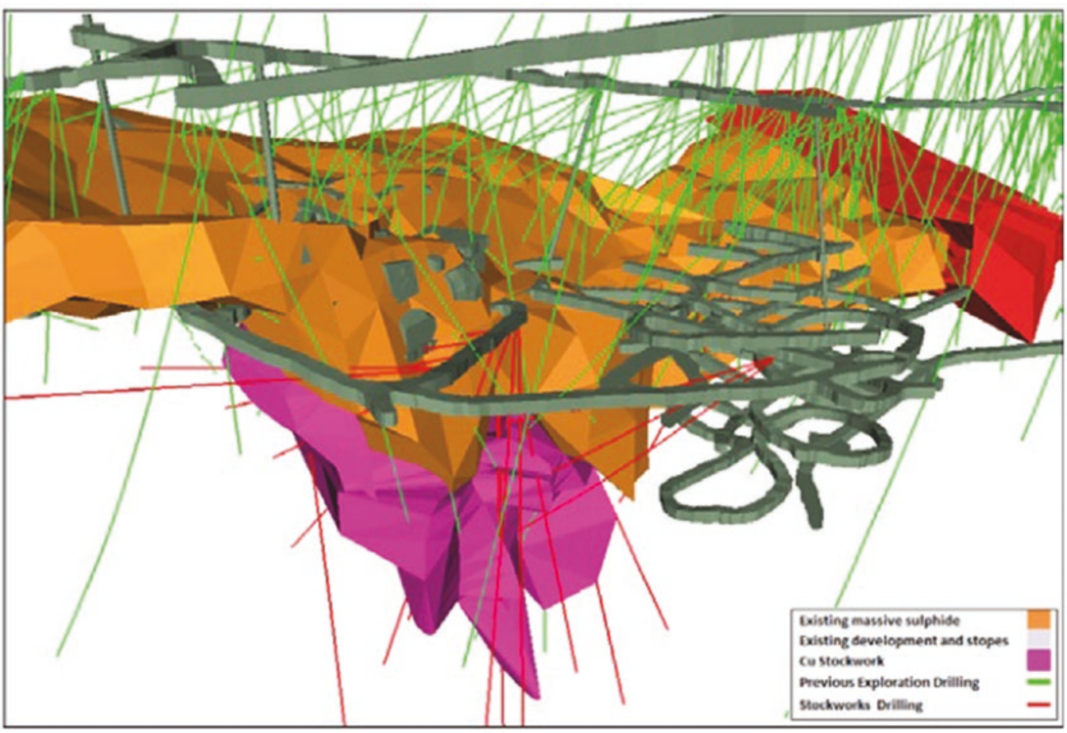

Fig. 8.11 3D View of Cu Stockwork Wireframe Model (illustration courtesy of Matsa, a Mubadala & Trafigura Company)

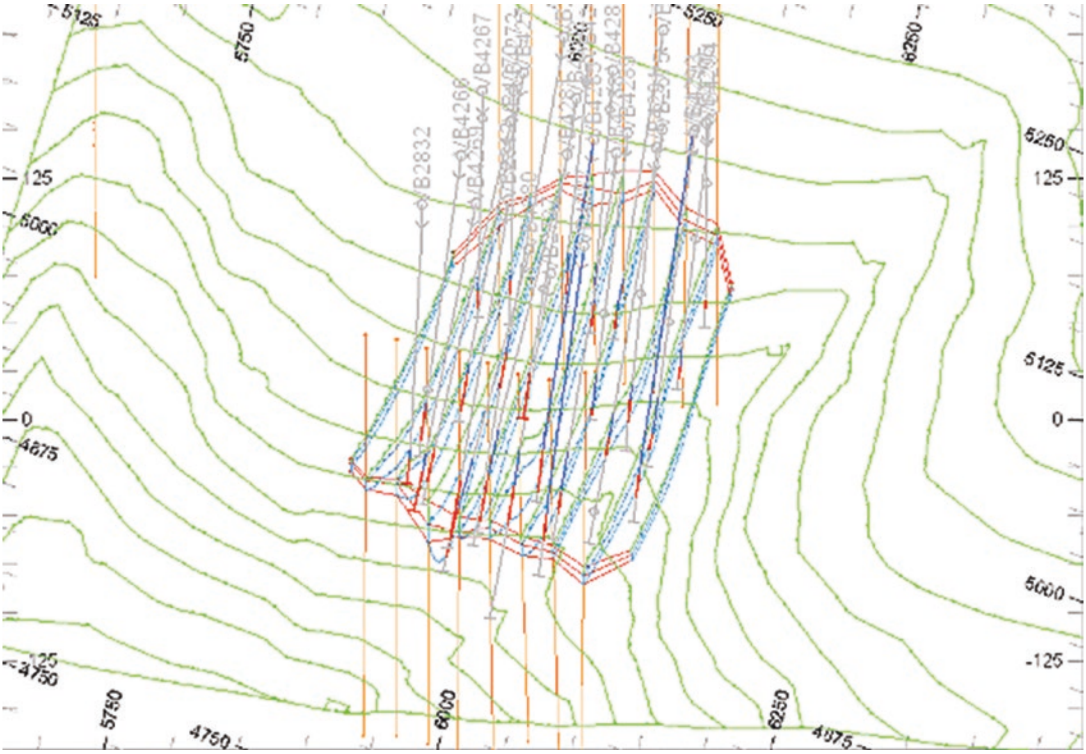

Fig. 8.12 String model (Datamine)

Geological Wireframe Modeling

It includes preparing several wireframe models such as topography surface model and ore body model. The topography surface wireframe model can be designed using digital terrain model (DTM) tools. In general, these tools are used to create wireframe models of open undulating surfaces such as topography, geological features (fault surfaces, lithology, or mineralization contact surfaces), open pit designs, and open pit survey measurements. The topography surface wireframe model is done using the topography contour strings object as a basis for the wireframe. The next step is to create a closed volume wireframe model of the ore body. Using a wireframe linking toolbar, a wireframe of the ore body is designed by clicking corresponding points. The wireframe obtained is a closed volume containing both wireframe surfaces at each end and between each section string (Fig. 8.13). There should not be any gaps nor holes in the wireframe volume. It is essential to verify wireframes before calculating wireframe volumes, evaluating wireframes (for tons and average grades) against drillholes or block models, and/or using wireframes for block modeling purposes. Otherwise, unverified wireframes can potentially cause problems with wireframe volume calculations, block modeling using wireframes, or other processes that use wireframes as input. After that, volumes for closed volume such as an ore body can be calculated.

Geological Block Modeling

Geological block modeling involves three steps: (a) creating a waste block model below the topography surface wireframe, (b) creating and ore body block model from a closed volume wireframe, and (c) creating a combined ore body and waste block model. For the first model, it is necessary to establish the cell size parameters and other settings. Once these data defined, the model is shown in Fig. 8.14. Similarly, an ore model within the ore body's closed volume wireframe object is created. Finally, the extent of the ore and waste block model against the surface DTM and the ore body wireframe is combined (Fig. 8.15).

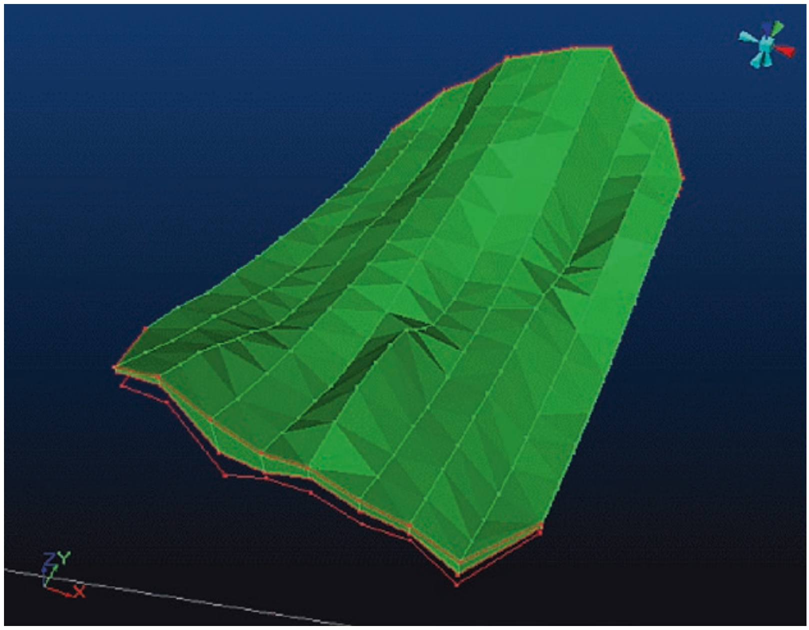

Fig. 8.13 Wireframe model of the ore body (Datamine)

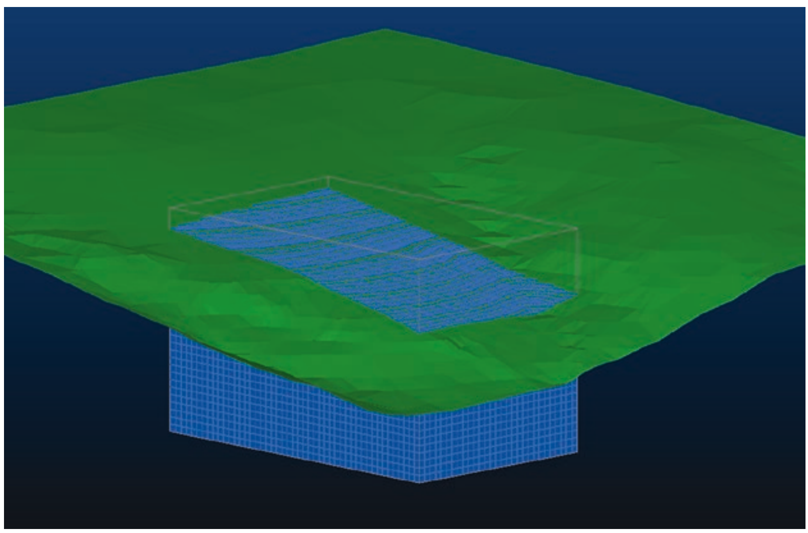

Fig. 8.14 Block model (Datamine)

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Then, it is needed to check the ore body block model. The process is carried out by visual methods or using summary statistics. The aim of visually checking a block model against its wireframe is to search errors such as cells extending beyond the limits of the wireframe (e.g., the cell center lies beyond the wireframe – the surface may be damaged or contain holes). Regarding checking of the ore body block model using summary statistics, the goal is to review the block model's numeric fields looking for errors such as absent data (e.g., cells not flagged with the mineralization zone field), records with missing values, and unexpected minimum or maximum values (e.g., coordinates or mineral grade values).

The last item in the geological modeling process is to combine the waste and ore block models because it is essential to have a single model for presentation, economic optimization, or evaluation purposes. However, it is important to note that combining block models can potentially result in very large files. Obviously, block models that do not have the same block model definition (e.g., block model prototype) cannot be combined. Figure 8.16 shows the result of combining both models. Hereafter, it is possible to optimize the resulting block model

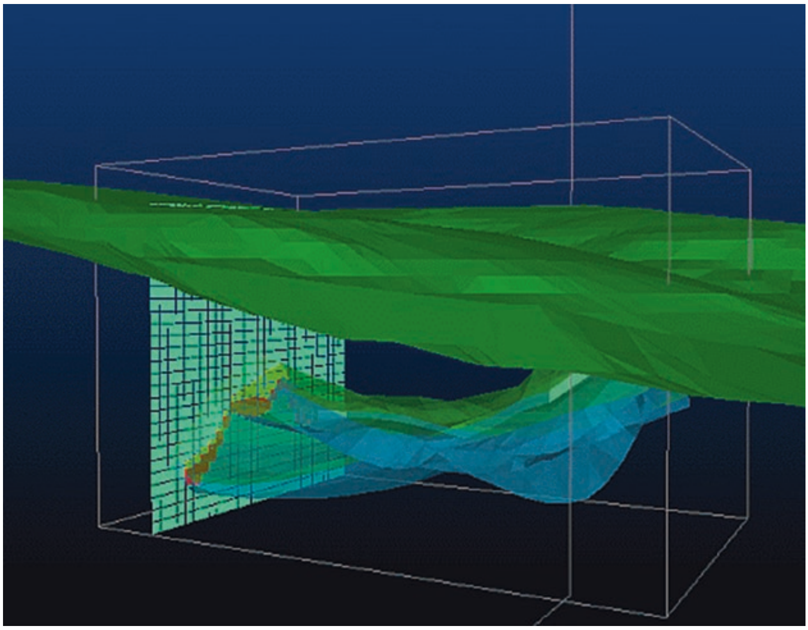


Fig. 8.15 Block model including ore body and topography (Datamine)

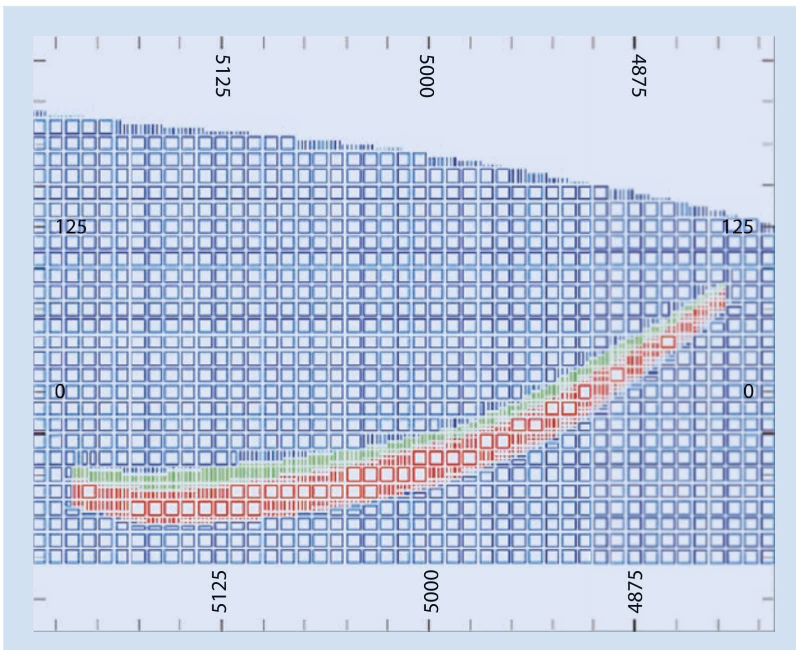


Fig. 8.16 Section of a waste and ore block model (Datamine)

since the optimization (combination of subcells within the limits of parent cells by one or more key fields) typically results in a smaller file size.

8.4.3 Datamine Studio OP

Since there are many approaches to task of surface mine design, there is not a predetermined way for completing the design work. Moreover, there are a great number of tools to design surface mines. Designing an open pit is an iterative process involving consideration of many design criteria, constraints, and objectives. Before to start a detailed pit design, it is necessary to have some idea of the existing surface topography, the extent and nature of the orebody, and the economic or ultimate pit limit of the mine. There are several ways to do this depending on available information. As aforementioned, two ways to represent the three topics commonly exist: block model or surface wireframe. For economic or ultimate pit limit, it is also possible to use strings. Typically these limits will have been determined from a financial analysis using techniques such as the Lerchs-Grossmann nested pits or floating cone pit optimization (see section devoted to NPV Scheduler).

There are four basic methods of creating a detailed pit design in Datamine OP: (a) Toe + Ramp + Crest Design: this method involves designing the pit on a bench by bench basis often starting from the lowest bench and working upwards; the toe string is created, the ramp is inserted, and finally the crest strings are added; the method builds ramps that include access to the berms and it can be applied working either from the bottom up or the top down; (b) Toe + Ramp Design: This method involves creating all the toe and ramp strings first and then adding in the crest strings later; it is quicker than previous approach and it yields a continuous ramp with no offsets; the pit can only be designed from the bottom upwards; (c) contour design: the third technique involves inserting a ramp and crest strings into an existing set of toe strings; these toe strings are typically created by generating contours around an optimum pit shell block model; and (d) automated pit design: this process is designed to speed up open pit design; it enables an iterative process whereby the design of a pit can be altered (ramp positions, gradients, berms, etc.) to find the best results. The first method is considered as an example.

Toe + Ramp + Crest Design

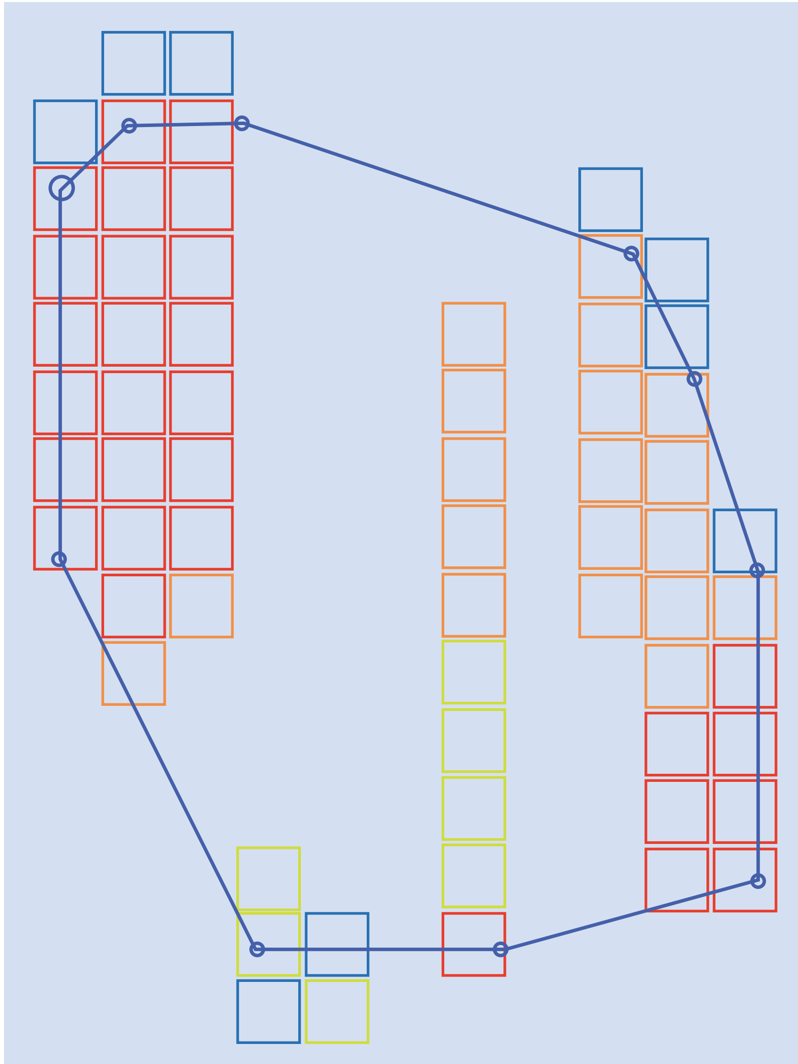
The following procedure outlines the stages that are commonly worked through in order to create a Toe + Ramp + Crest design. The process can be summarized in three steps: (a) to create the base string and the first bench, (b) to define the second bench, and (c) to complete the design.

To create the base string and the first bench:

1. Set the face angle and berm width for the new design, for example, 60 degrees and 4.5 meters, respectively. It is possible to change this setting at any time during pit design without affecting any previously created pit strings.
2. Load all supporting files (e.g., any relevant block model, orebody wireframe, and/or topography data).
3. Navigate to an appropriate plan view and apply clipping limits to the data so that only the data that is relevant for the initial string can be seen; assuming a bottom-up design, this would be the pit base string.
4. Digitize the initial string using a view of the relevant data objects (e.g., block model section); this string must be closed (◘ Fig. 8.17).
5. Create the first road segment and edit the new road string as required; it is essential to take care not to erase the point at which the road string meets the initial (toe) string. It must be defined a road gradient percentage, a road width, and a target elevation; this is commonly the toe elevation of the next bench in which a berm will be inserted.
6. Create the crest and toe strings; the bench elevation (the height of the first step running around and above the pit base) must be established.
7. Once a contour string has been defined, it is possible to add the road berm; the string created becomes the toe string for the next bench. At this stage, the basic components for the pit floor rising to the first bench position have been created.

Once the initial pit strings have been created, it is necessary to design the next bench; to create subsequent benches, it is important to click in the correct location where creating new road segments, switchbacks, and contours. This is to ensure that the integrity of the haul route design is maintained. The following procedure outlines the general method for the bench design above the pit floor. The first step is, with the initial pit design work in a plan view, to create a road segment specifying a road gradient (the steepness of the road to the crest of the next bench), a road width, and a target elevation (if designing from the base upwards, this will be a higher elevation than that specified for the previous bench); the new road segment will be projected at the specified gradient until it reaches the target elevation. Then, it is necessary to specify the direction around the pit the road will be constructed. Thus, a pit base string has been designed and a ramp projected. A contour string and road berm have also been applied at this level.

Then, the next step is to define the position of the next road contour by specifying an elevation for the new string and specifying a new road berm; this will create the toe string for the third



■ **Fig. 8.17** Digitized initial string of the first bench (Datamine)

bench. Consequently, the present design includes a pit base, with a road rising to the first bench, and geometry for the second bench including a continued haul route. To complete the design, it can continue using the procedure above to introduce all subsequent benches. The process finishes where the projecting benches reach the existing topography.

Creating a Digital Terrain Model

Once a full pit design is completed, it is common to build a digital terrain model (DTM) of the pit design strings. At the first stage, it is often useful to see your pit design in context, showing the level of the surrounding topography and trimming data where it sits above the surface. There are several ways of achieving this, but the method described below is a good option to create a merged pit/topography.

The first step is to load the topography wireframe that is relevant to the pit design. The next step is to create a string where the topography file intersects with your new pit design (■ Fig. 8.18). At this stage, it is recommended to «condition» the intersection string. Conditioning will result in the string being comprised of chord lengths not exceeding a maximum length and/or angle. This method will help to create a more accurate (i.e., containing more data) intersection string. It will produce a more realistic result where used to trim the pit wireframe. Next, a DTM of the clipped pit strings is created (■ Fig. 8.19).

The final stage is to carry out a hole in the topography and draw it with the trimmed pit

DTM. There are several methods for doing this, but the simplest is to project an existing string onto a wireframe surface and remove the data that is contained within. This option also offers the possibility of verifying your wireframe data beforehand. The final display shows both digital terrain models aligned at the pit boundary (Fig. 8.20). The view of these data can be enhanced as required, with each object (pit and topography) being independently controllable.

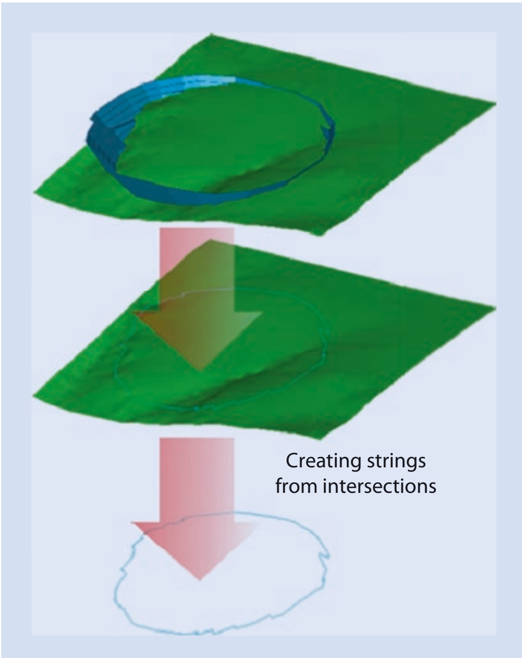
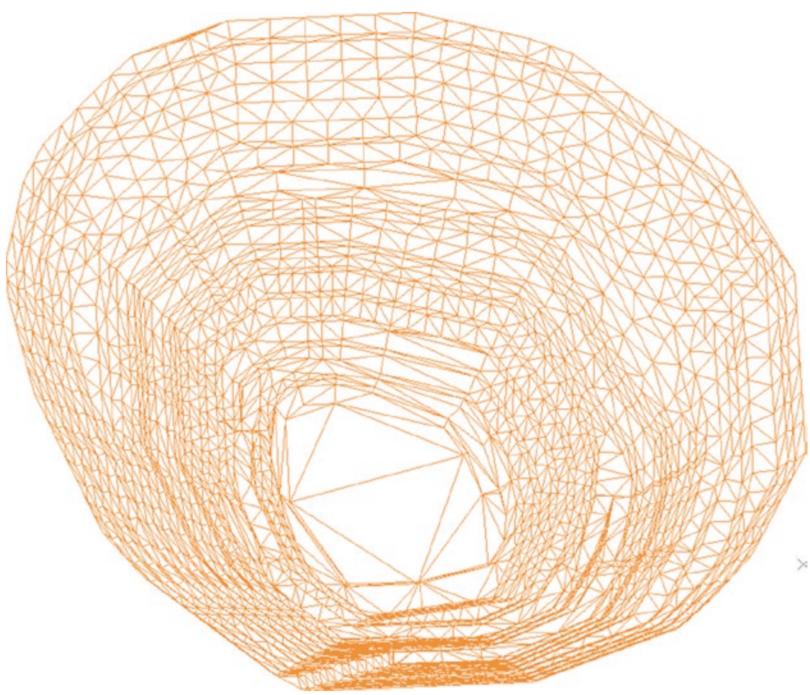


Fig. 8.18 Creating strings from intersections (Datamine)

8.4.4 NPV Scheduler

The goal of any mine scheduling operation is to define a sequence of events that will generate the highest possible net present value (NPV) over the life of the mine where applied to a real-world mining scenario. Therefore, determining the optimal strategic plan for an open pit operation requires the solution of complex mathematical problems that are bound by various constraints, including but not limited to (a) the geological realities of the ore body (e.g., grade distribution), (b) the economic conditions for mining (e.g., costs, commodity value, and discount rate), and (c) the engineering requirements for pit slope, dilution, minimum mining width, mill recovery, etc. In a brief summary, the scheduling process involves the following steps: (1) importing data (e.g., geological model and region perimeters), (2) generat-

Fig. 8.19 DTM of the clipped pit strings (Datamine)



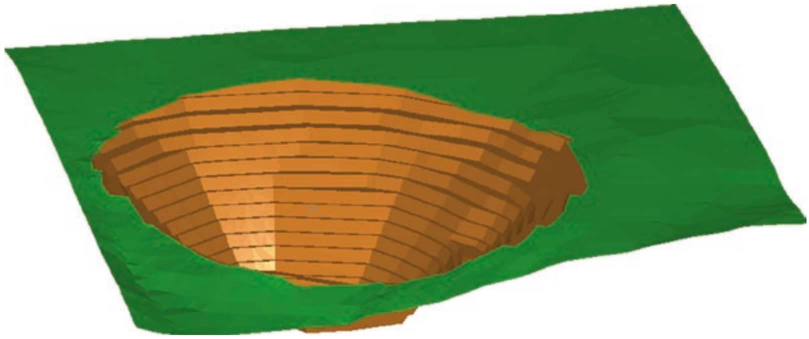


Fig. 8.20 Pit and topography DTM (Datamine)

ing economic model, (3) optimizing the pit, and (4) generating pushbacks.

Importing the Data

The first step is to import the geological block model created previously. The geological block model includes the ore (raw material) and waste cells containing product grades and other information. Air cells (voids) do not need to be present. The database includes many items such as product (metal or other mineral element that can be recovered from ore and sold at a price), density, rock type, grade, tonnage factor, among others. Hereafter, it is necessary to import the region perimeters, an AutoCAD file including topics such as the final pit limit created in previous section, slope regions, etc.

Generating Economic Model

This step applies economic and technological parameters to the geological model with the objective of obtaining the economic model. An economic model is a geological block model supporting additional information relating to the value of a particular block of ore. It is defined by setting cost and price parameters for the life of the mine and then calculating an intrinsic value per processing method of each block in a geological block model as a function of its geo-metallurgical attributes. The value is commonly calculated by NPV Scheduler as part of the definition of the economic model, but values can also be imported as an attribute of a block model if it has been calculated externally in another system. The parameters used to calculate the block value are (a) the selling price of any commodity recovered from processing where the recovery is defined as a mathematical expression of values in the block, (b) a unit cost of mining (ore and waste) and a unit cost of processing (ore) and any adjustment factors that apply, (c) dilution and recovery factors for the ore, (d) a unit cost of rehabilitation for waste, and (e) an additional cost for processing each unit of a commodity.

This model is the starting point of NPV Scheduler processes. From the information it contains, a set of mining phases can be defined based on the basic principle of ensuring areas of high grade are mined, wherever possible, before areas of lower grade. Considering that realizing the best value first is a basic principle of maximizing NPV, the order of the phases represents the first high-level categorization of the value and the first stage of determining the optimal extraction sequence.

Optimizing the Pit

The pit optimization process is carried out using Lerchs-Grossmann (LG) algorithm (▶ see Chap. 5), which is a mathematical algorithm belonging to the family of network flow methods that finds open pit shell yielding maximum profit. The optimization of the pit includes successively defining slope regions and setting up pit optimization parameters. The former sets the slope angles by region or for one global region encompassing the entire block model volume. The latter has no effect on the computation of the Lerchs-Grossmann ultimate pit and is only used for the calculation of the NPV estimates (e.g., annual discounting to 15%, average ore output rate to 6,000,000 tons per 365 days, etc.). Pit optimizer generates an ultimate pit, LG phases, and optimal extraction sequence. Figure 8.21 shows the chart for the LG phases report.

Generating Pushbacks

In simple terms, a pushback (also called stage, phase, and cutback) refers to a designated zone representing an area to be mined as a single

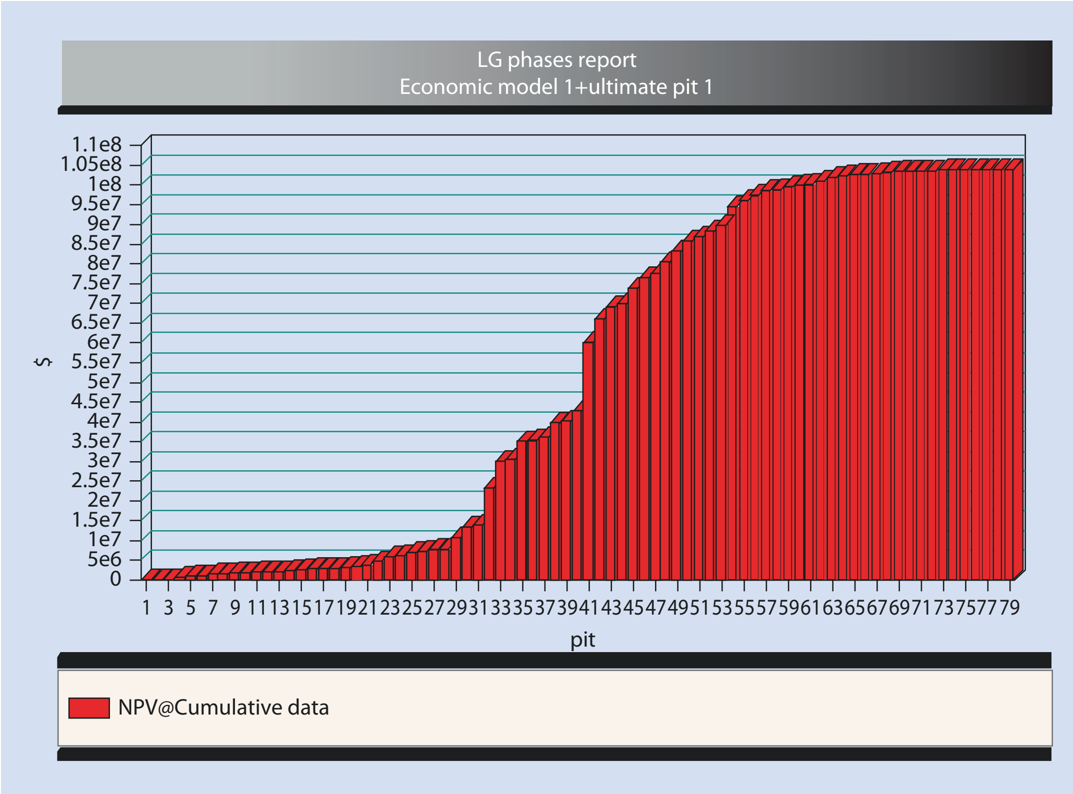


Fig. 8.21 Report of the LG phases (Datamine)

operation in terms of accounting. A pushback, in the context of NPV Scheduler, is a collection of spatially connected blocks with more or less regular geometry that, after some minor modifications at the design stage, could become a practically minable pit. In most cases, no single pushback will ever represent an actual mine topography because two or more pushbacks will be mined at the same time. The actual mine topographies are generated by the Scheduler and can be saved as annual surfaces. Once the ultimate pit optimal extraction sequence has been established, the pushback generation can commence. Pushback generator reevaluates pit optimizer optimal extraction sequence combining the blocks into spatially connected sets and adjusts these sets to meet the requirements of pushback definition. Thus, the basic objective is to create a pushback shape that meets some primary targets, namely, the ore tonnage to be won from each pushback as well as its minimum mining width.

The other parameters that can be set include mining constraints on the maximum number of pushbacks generated as well as some more general parameters for making the pushbacks practical. These include (a) defining the feasible size of the «remnants» between a pushback and the ultimate pit that must be included so that later mining need not return to capture this ore, (b) defining the «smoothness» of the pit walls to make the shape practical to blast and mine, and (c) in addition, the pushback can be directed to include the area of its predecessor or consist of contiguous blocks. NPV Scheduler then creates the pushbacks by re-sorting the OES to get contiguous blocks of ore that create a practical mining shape. These are accumulated in different combinations, and the different combinations are compared until an optimum is found.

Figure 8.22 shows the steps by which the pushback generator selects blocks for the pushback, joins them together to form contiguous shapes, takes the remnant blocks into the pushback shape, and then joins the two pushbacks so that they form practical mining strategies. For example, PB2i represents a grouped sequence of blocks established by the optimal extraction

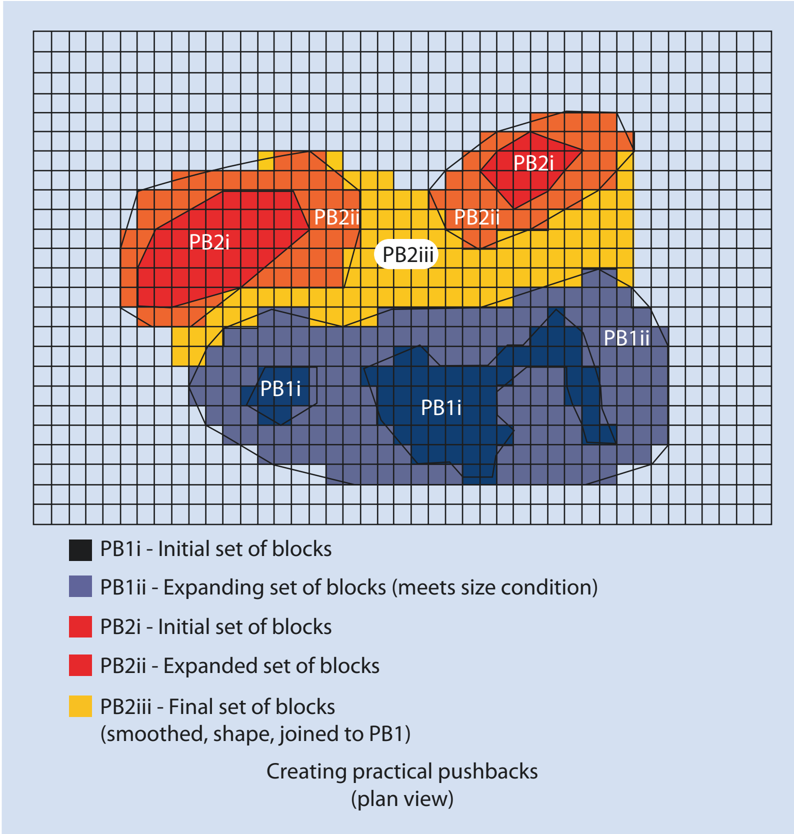


Fig. 8.22 Steps of the pushback generator (Datamine)

sequence. These groupings are expanded to create PB2ii by including late-sequenced blocks so that a specific ore tonnage condition for the second pushback is achieved. The further addition of blocks to create PB2iii successfully joins second pushback to the previous pushback and creates smooth edges. The importance of the pushback shapes therefore is that rather than being the physical stages in a mine design, they form economic boundaries about which management decisions must be made regarding the mine life

Other Options

Once an extraction sequence has been calculated, it is possible to schedule the pushbacks. It is possible to schedule either NPV Scheduler-generated pushbacks (default) or another pit sequence, for example, imported pushbacks. With any pushback file, the default topography specified by pit optimizer settings can be used or to designate any other surface as topography. Mine flow optimization is also included in the NPV Scheduler. It explores the ways of increasing project NPV by increasing mining rates without costly expansion of processing capacities. Higher mining rates result in higher ore production, and the excess ore that cannot be processed by the optimal method is processed by another (suboptimal) method, stockpiled for later processing, or dumped as waste. In some cases, where mine flow optimization succeeds, it generates higher NPV, lower life of the mine profits (some wasting of resources is inevitable), and shorter mine life. For a more complex scenario, other topics such as assigning capital costs or assessing geological risk can be used in the NPV Scheduler.

8.5 Questions

Short Questions

- What are the main types of mining software?
- List some examples of inexpensive mining software.
- What is the most famous public domain geostatistical software in the history?

- List several packages of specific mining software.
- Explain the importance of Whittle software.

Long Questions

- Discuss the differences between a software such as RockWorks and a software such as Datamine.

Reference

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Supplementary Information

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