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

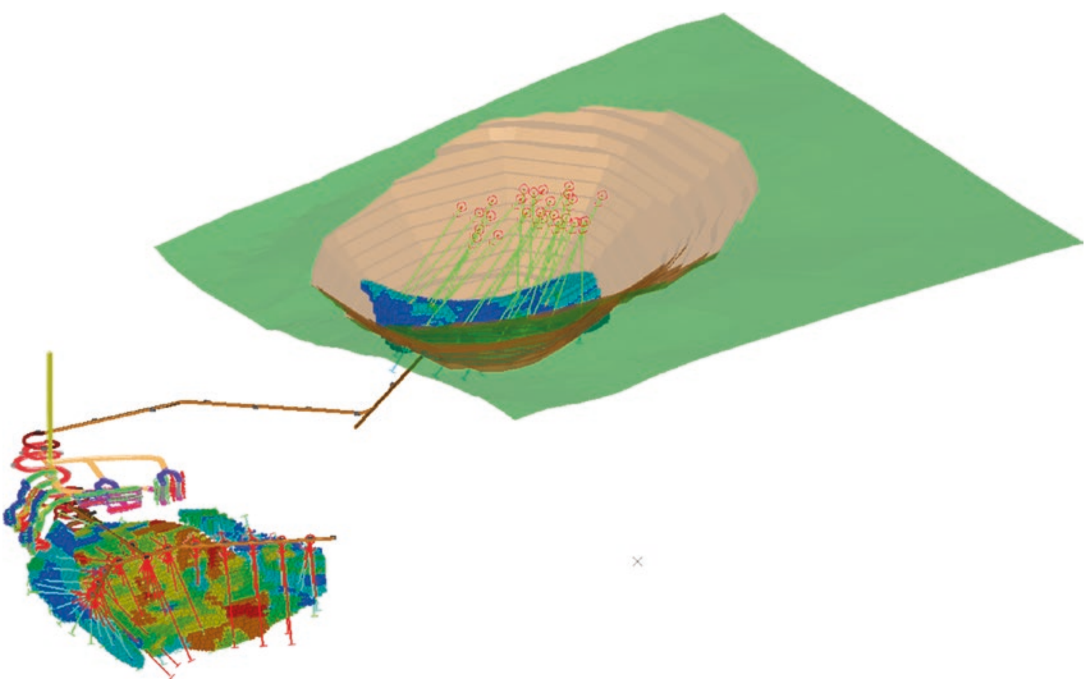


Fig. 5.3 Software mining design including an open-pit and underground planning with the respective bore holes and block models (Illustration courtesy of Datamine)

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

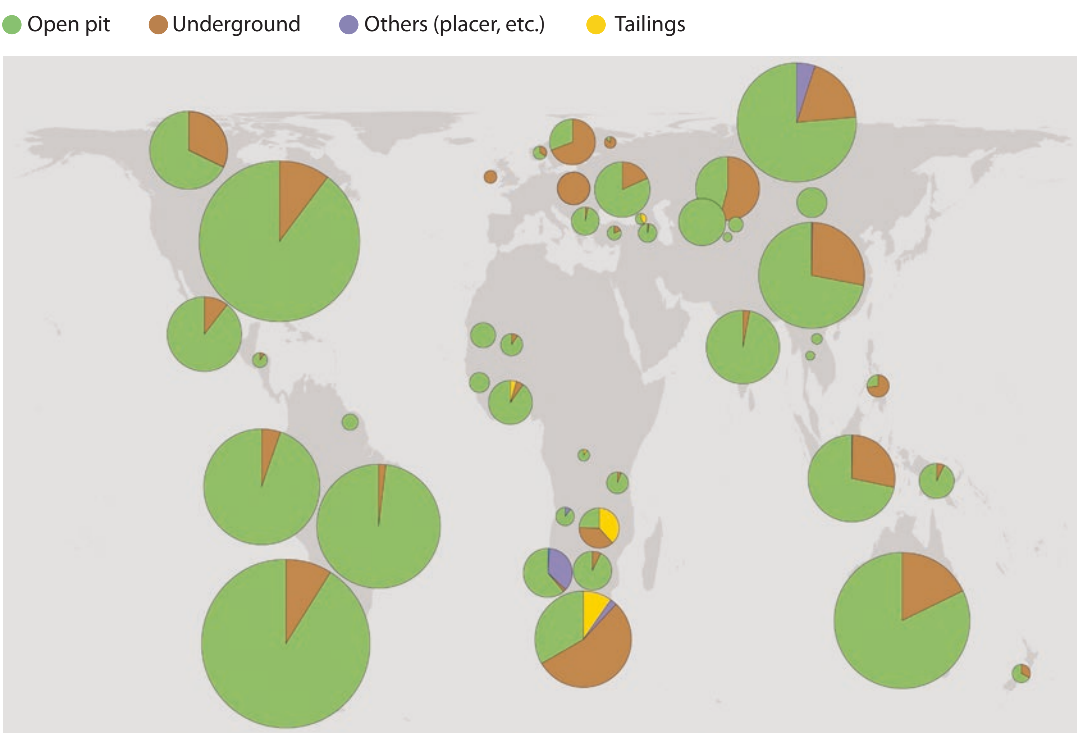


Fig. 5.4 Production by mining method 2011 (ICMM 2012)

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

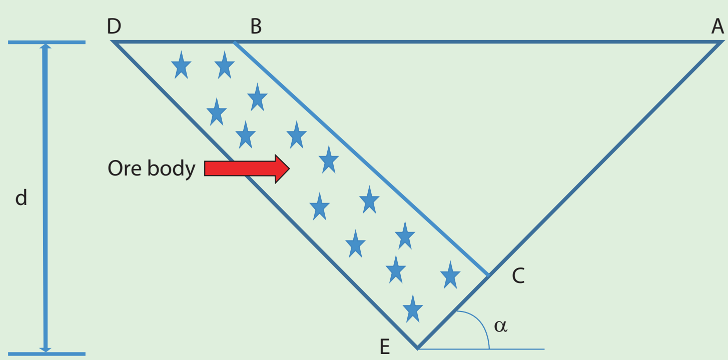
Stripping Ratio

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:

$$\text{Dilution} = \frac{\text{Waste (tonnes)}}{\left(\text{Ore (tonnes)} + \text{Waste (tonnes)}\right)} \times 100$$

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

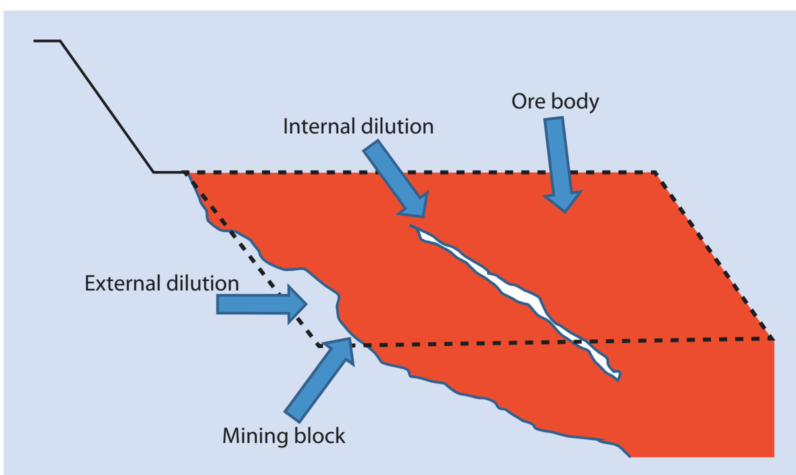


Fig. 5.6 Internal and external dilution (Modified after Ibrahimi 2013)

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

$$\text{Life(years)} \sim 0.2 \times \sqrt[4]{\text{Expected ore tonnage}}$$

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

$$\text{Life(years)} \sim (1 + 0.2) \times 6.5 \times \sqrt[4]{\text{Ore Tonnage in millions}}$$

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).

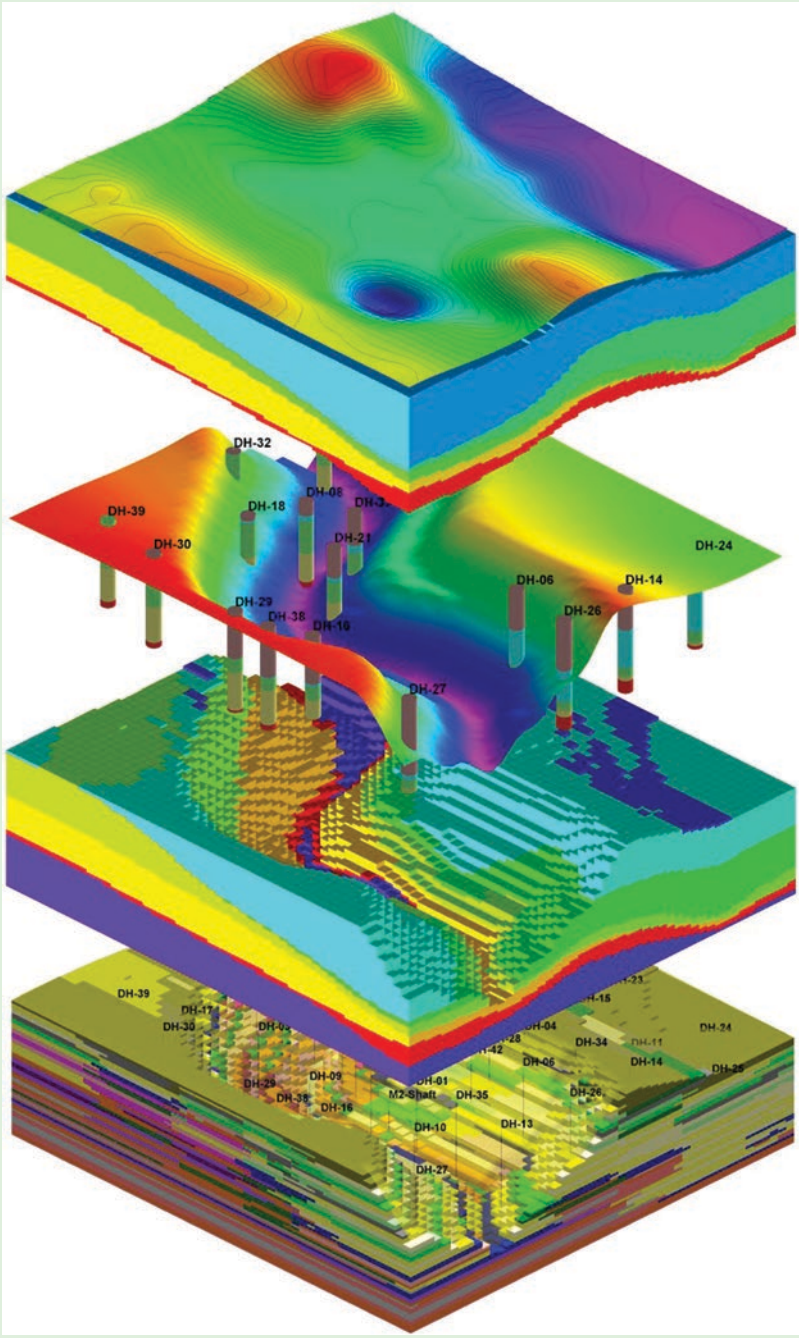


Fig. 5.9 Geotechnical composite from stratigraphic block model (*top*) to lithologic model (*bottom*) using RockWorks 17 (Illustration courtesy of RockWare)

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

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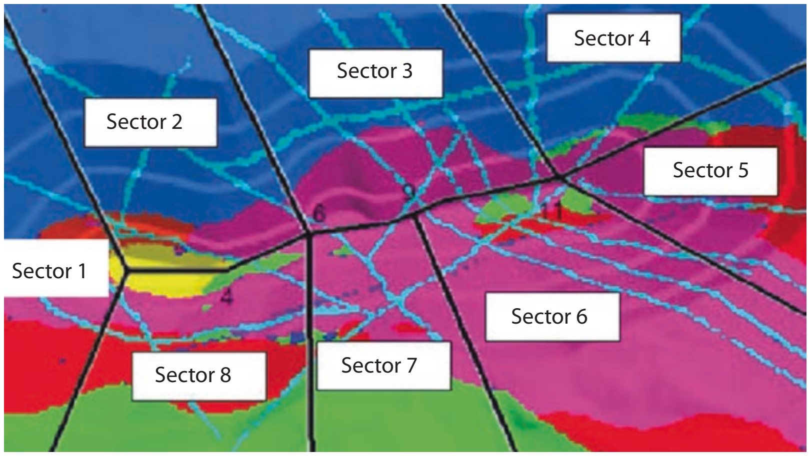


Fig. 5.10 Pit geotechnical domains based on geometric, geological, rock mass quality characteristics, and hydrogeological considerations (Illustration courtesy of KGHM)

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 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)

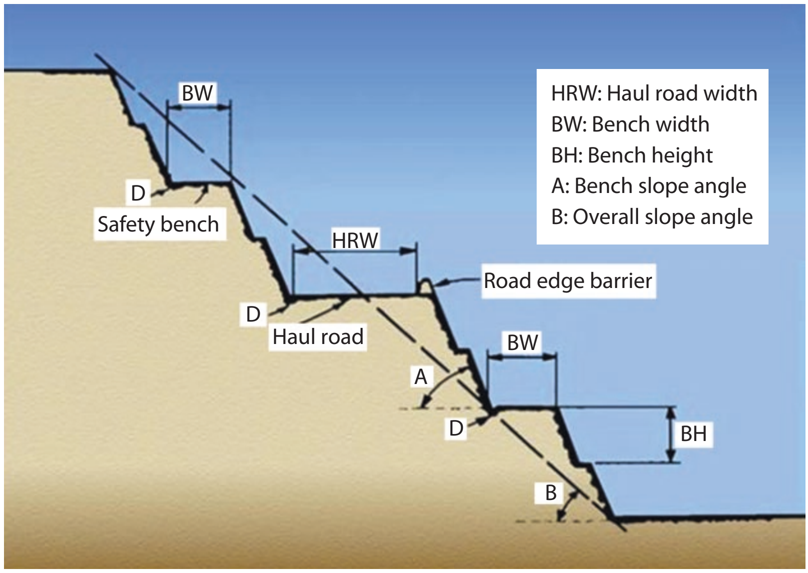


Fig. 5.26 Some concepts of open-pit design

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.

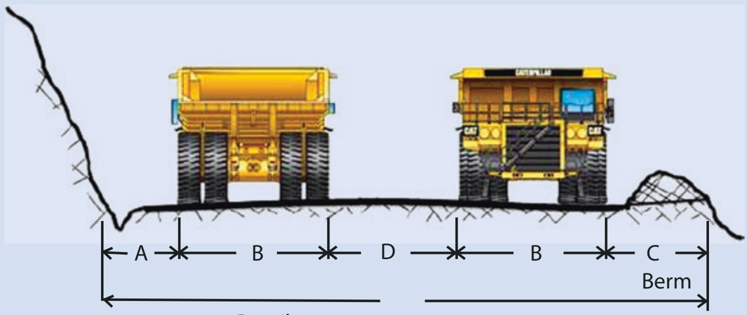
There are several types of benches: (a) working benches, which are in the process of being mined; (b) inactive benches, the remnant of working benches left in place to maintain pit slope stability; and (c) safety or catch benches, with a purpose of collecting the material which slides down from benches 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.

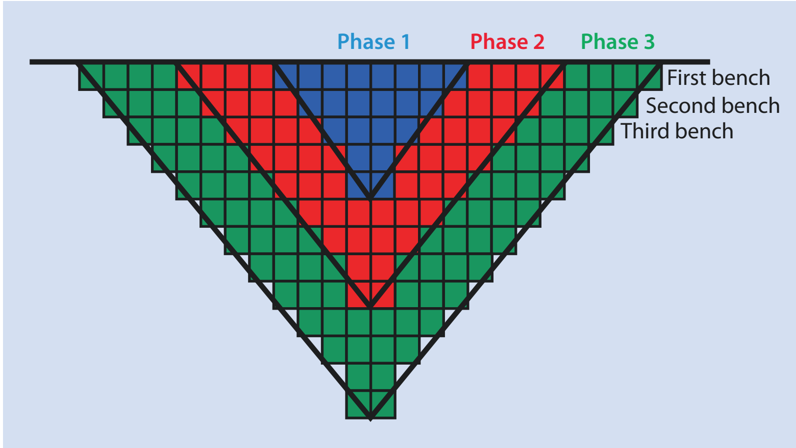


Fig. 5.28 Nested pits

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

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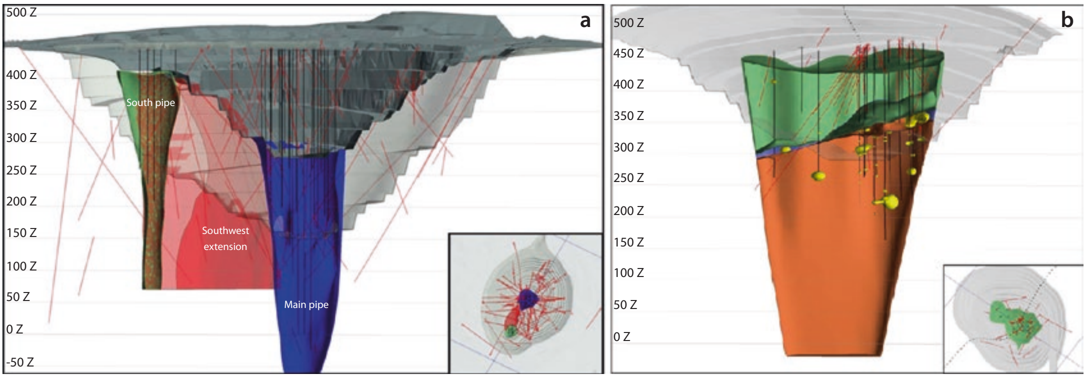


Fig. 5.30 Isometric views of Misery Main **a** and Pigeon pipes **b** (Illustration courtesy of Dominion Diamond Corporation)

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

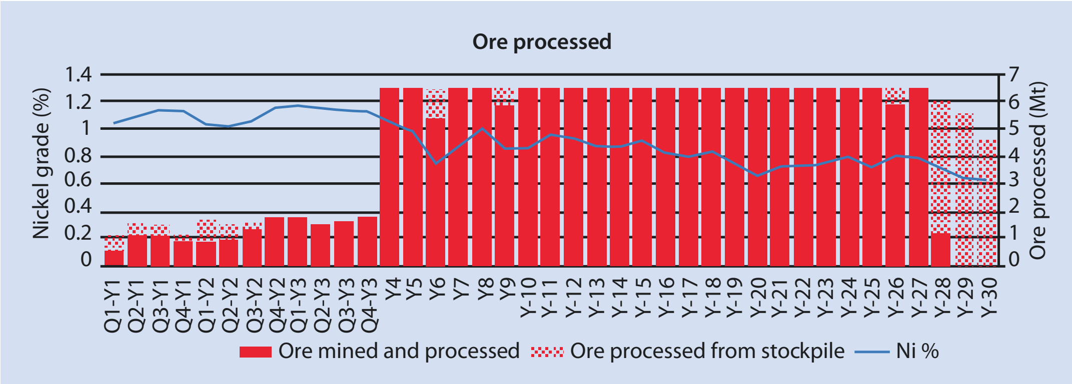


Fig. 5.31 Mill feed production schedule – mined and stockpiled feed and nickel grade by period (Illustration courtesy of Sherritt International Corporation)

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

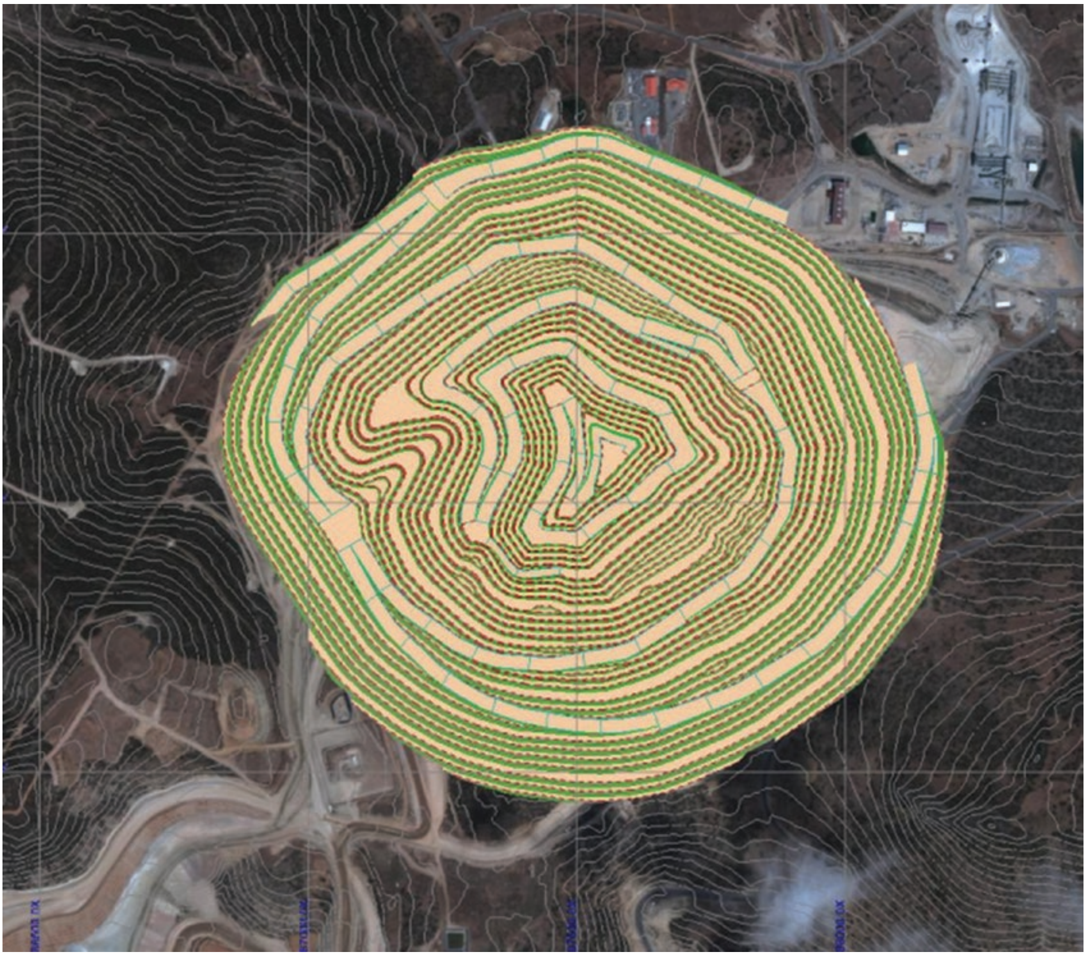


Fig. 5.32 Kişladağ final open-pit design at 2030 (Illustration courtesy of Eldorado Gold Corporation)

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

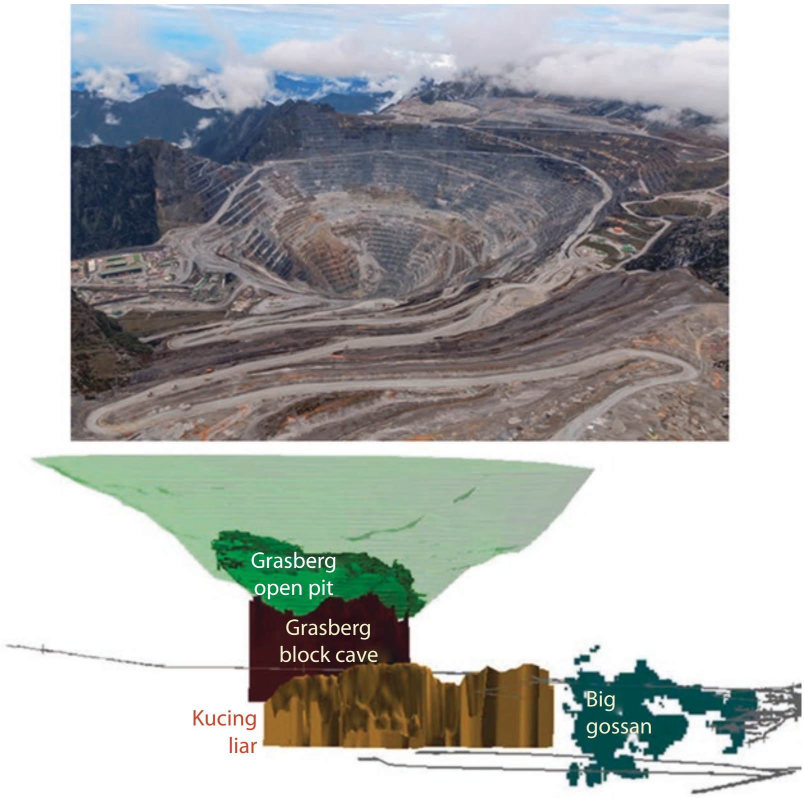


Fig. 5.42 Grasberg (Indonesia) copper and gold open-pit mine at 2014 and proposed transition to underground mining (Image and illustration courtesy of Freeport-McMoRan)

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

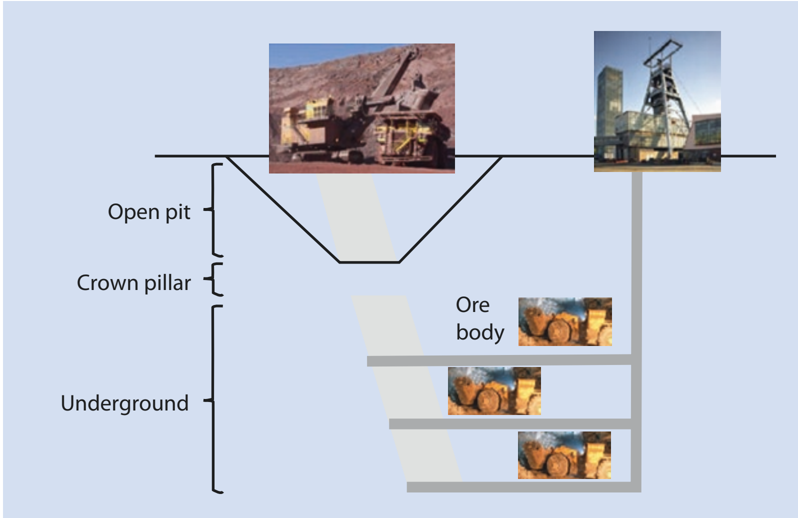


Fig. 5.43 Illustration of a crown pillar between an open-pit and underground mine

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.

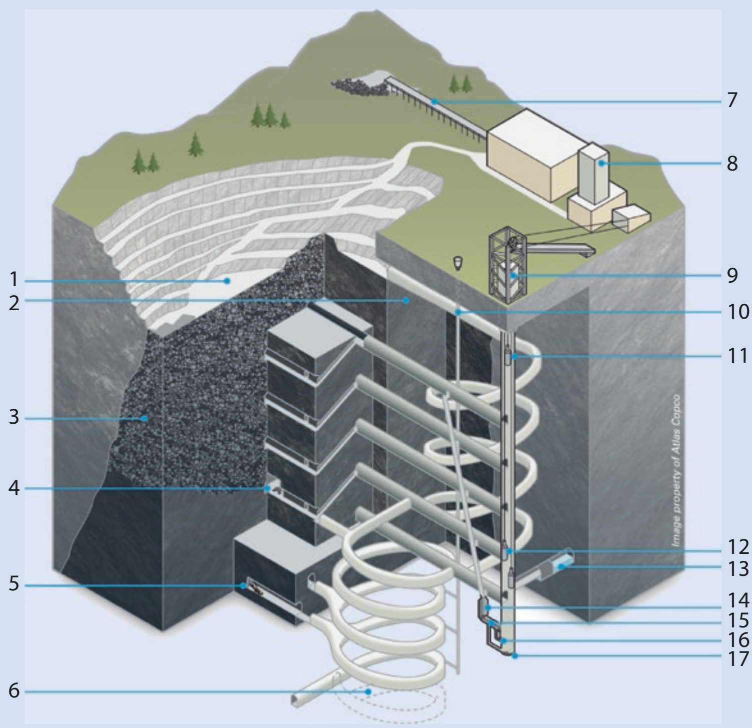
Underground mining methods are always selected below 1000 m depth, because it becomes difficult in a surficial exploitation to maintain the stability of a 1000 m high rock slope. Large tabular mineral deposits with long vertical or horizontal dimensions or mineralization lying more than 300 m below the 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)



1 - Open pit (mine out)
2 - Decline
3 - Mined out and backfilled or caving masses (SLC, BC)
4 - Exploration drilling
5 - Producing stopes and development of stopes
6 - Future reserves
7 - Tailings
8 - Production plant
9 - Headframe
10 - Ventilation shaft
11 - Skip
12 - Cage
13 - Water basin - pump station
14 - Ore bin
15 - Convenyor belt
16 - Measuring pocket
17 - Sump

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.

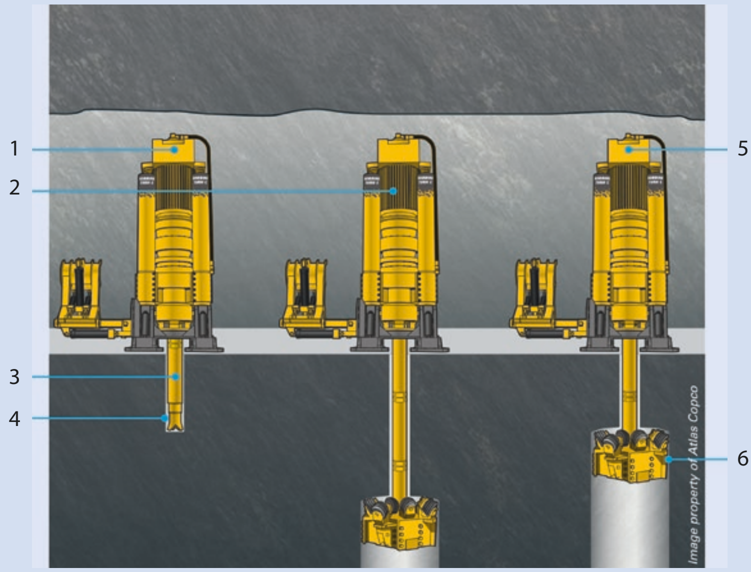


Fig. 5.51 Raise boring process (Illustration courtesy of Atlas Copco)

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), chain-link 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.

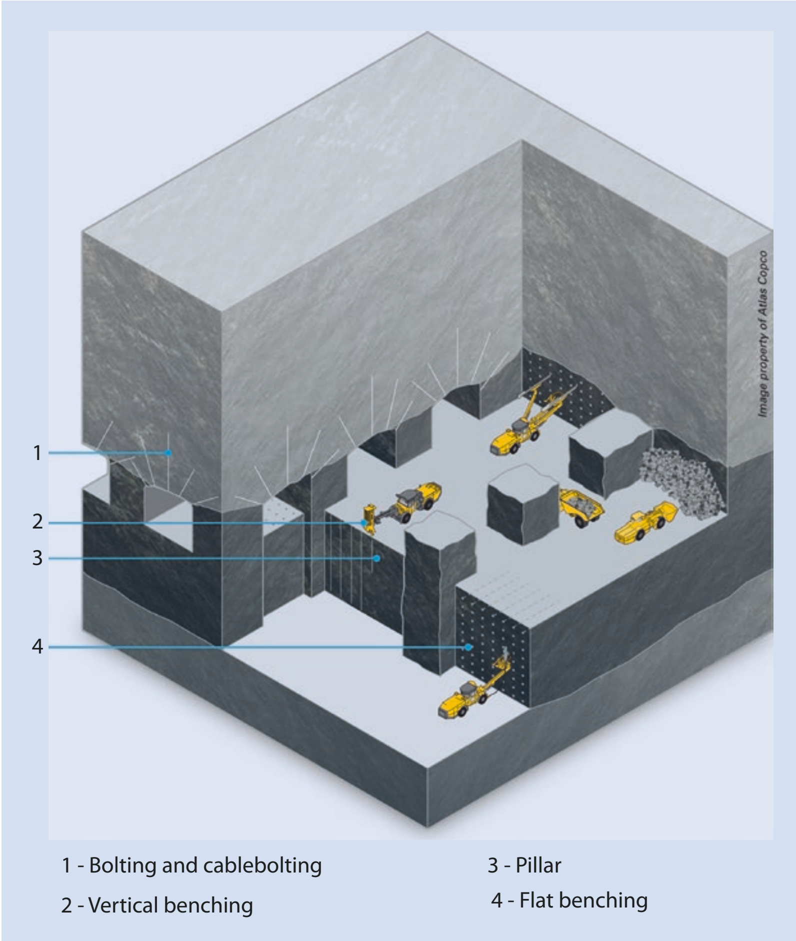


Fig. 5.60 Room-and-pillar method (Illustration courtesy of Atlas Copco)

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

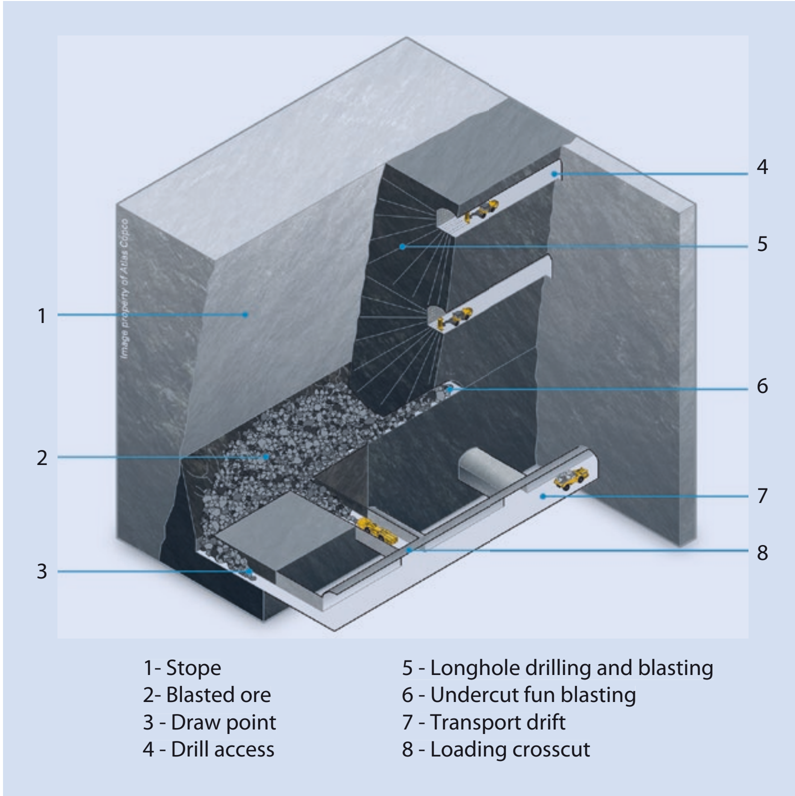


Fig. 5.62 Sublevel stoping method (Illustration courtesy of Atlas Copco)

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.

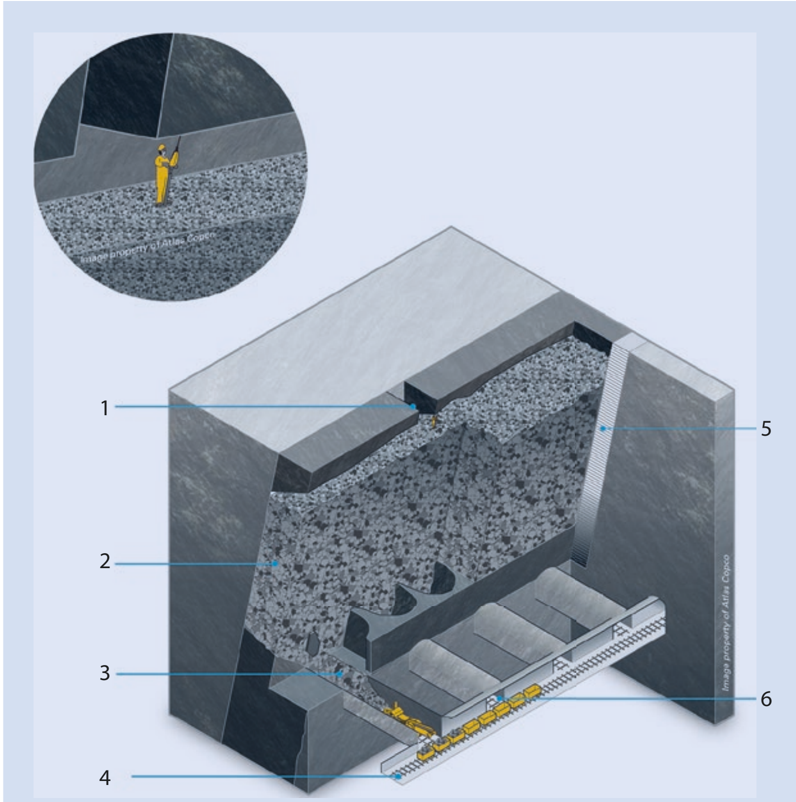


Fig. 5.64 Shrinkage stoping method (Illustration courtesy of Atlas Copco)

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,

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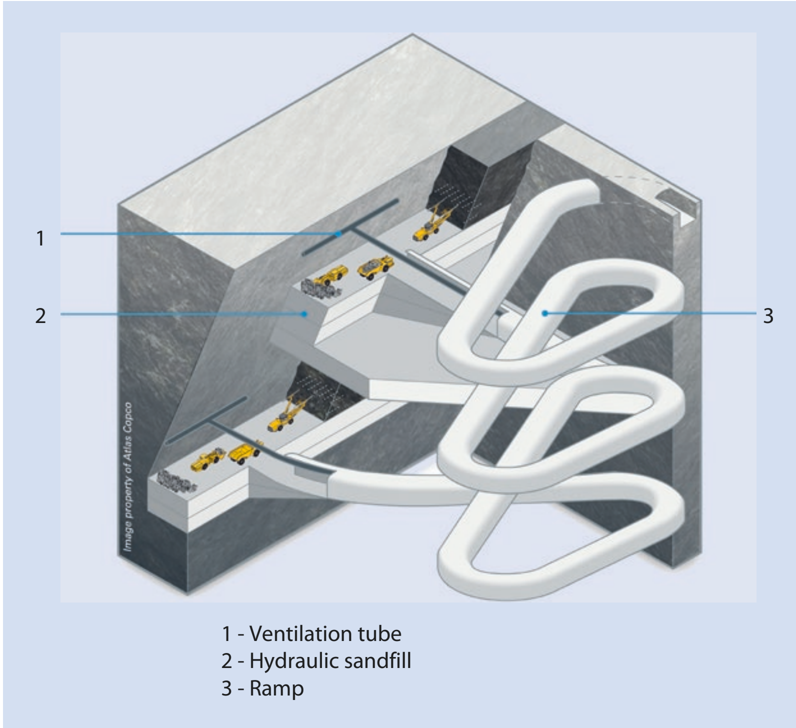

Fig. 5.65 Cut-and-fill method (Illustration courtesy of Atlas Copco)

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.

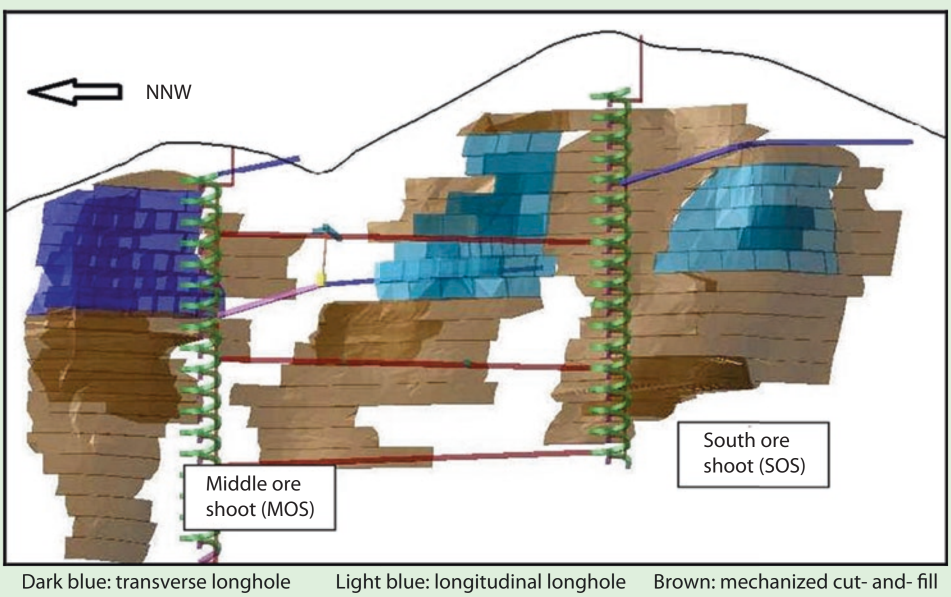

Fig. 5.66 Mining method (Illustration courtesy of Eldorado Gold Corporation)

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.

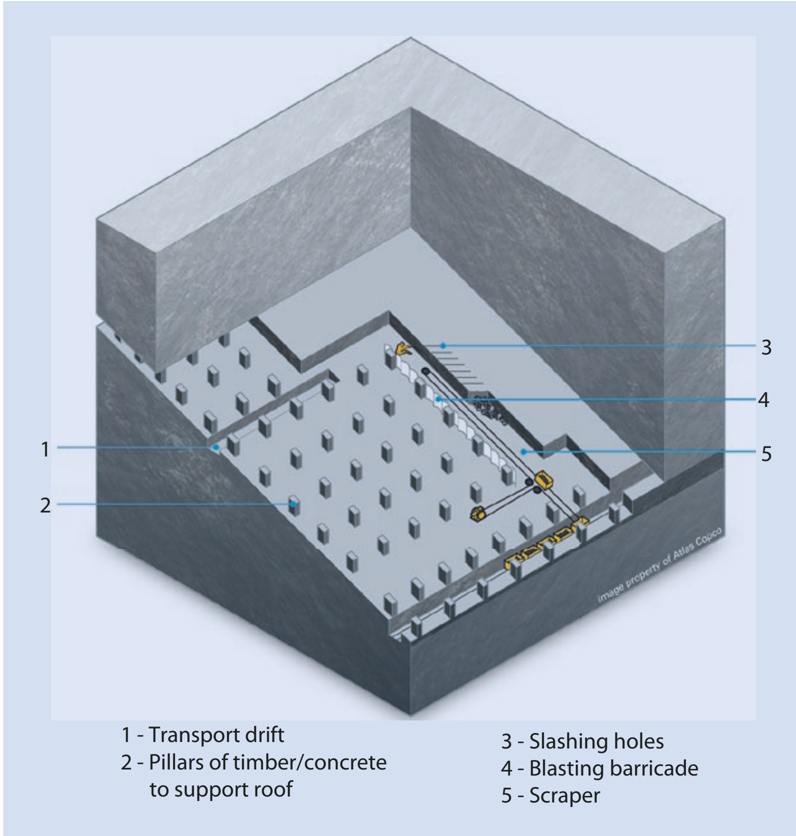

Fig. 5.67 Longwall mining method (Illustration courtesy of Atlas Copco)

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.

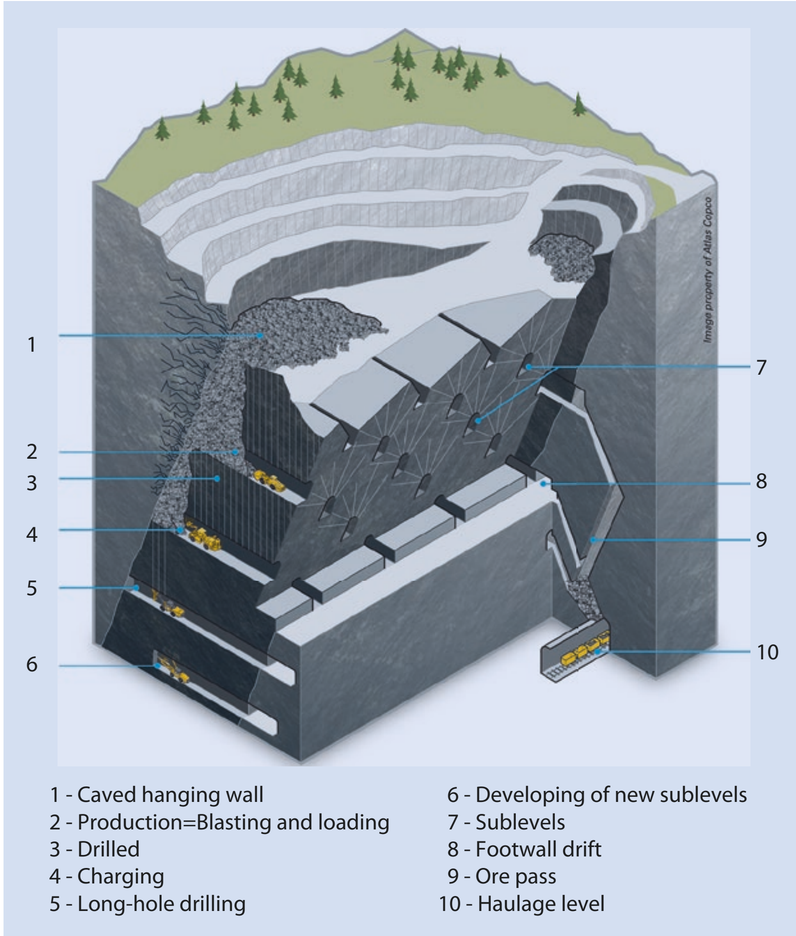

Fig. 5.70 Sublevel caving method (Illustration courtesy of Atlas Copco)

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).

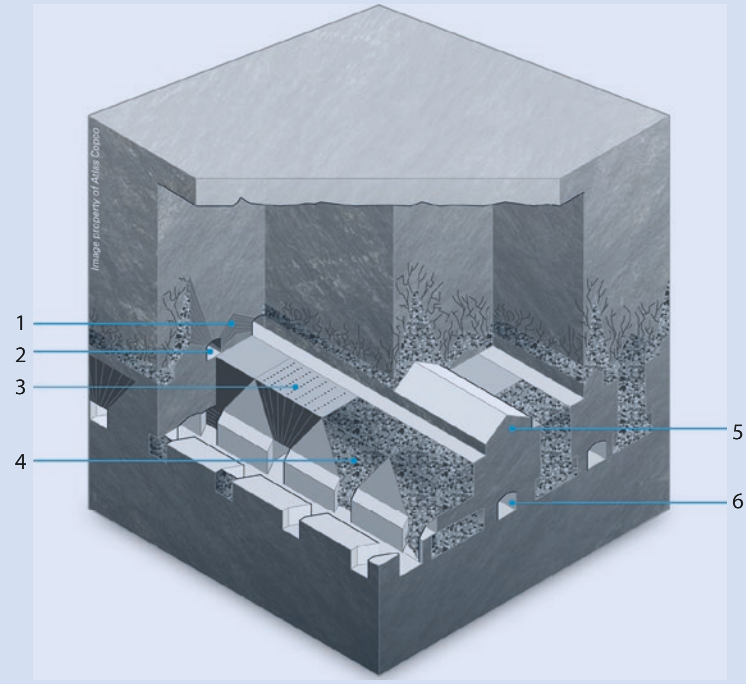

Fig. 5.71 Block caving method (Illustration courtesy of Atlas Copco)

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

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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.

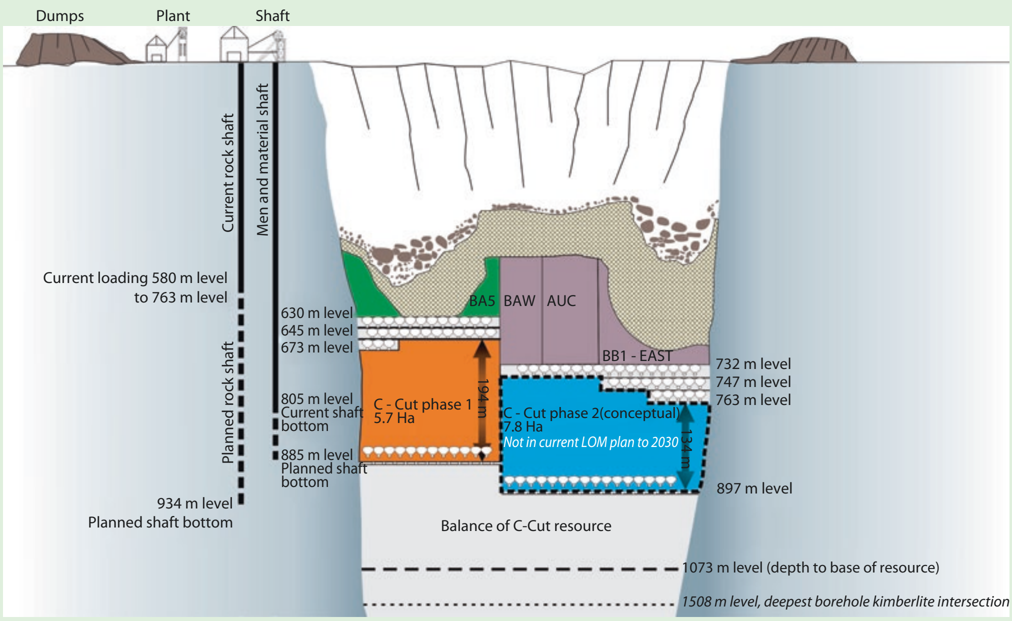

Fig. 5.72 Block cave method in Cullinan Diamond Mine (Illustration courtesy of Petra Diamonds)

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)

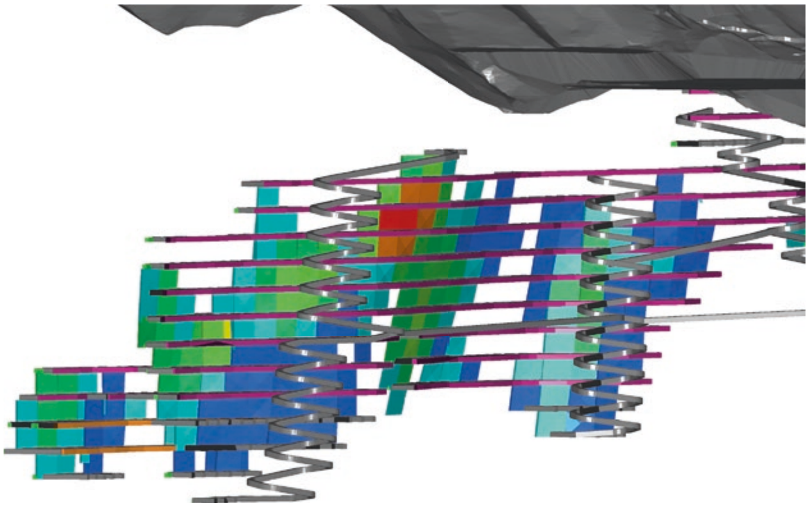

Fig. 5.74 Optimized stopes using Mineable Shape Optimizer (Illustration courtesy of Datamine)

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

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

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

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/cm^3. 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

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

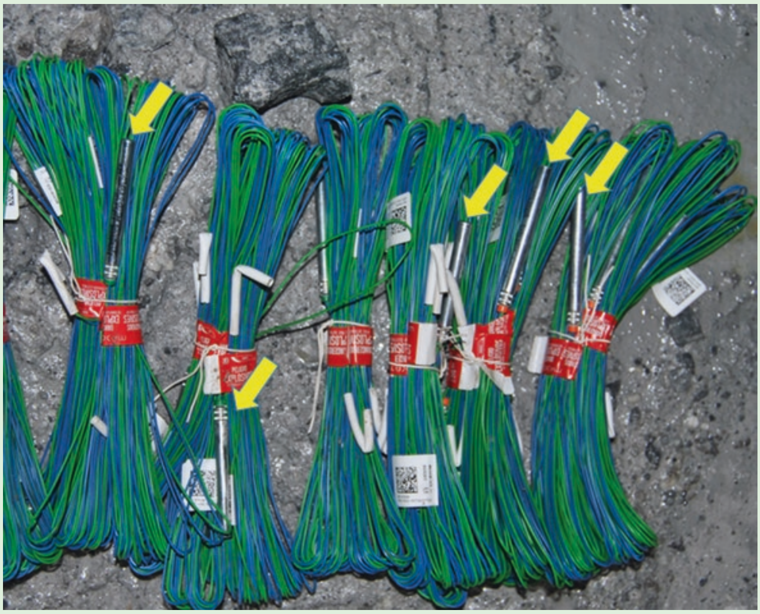

Fig. 5.84 Detonators (Image courtesy of Magnesitas de Rubián, S.A.)

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.

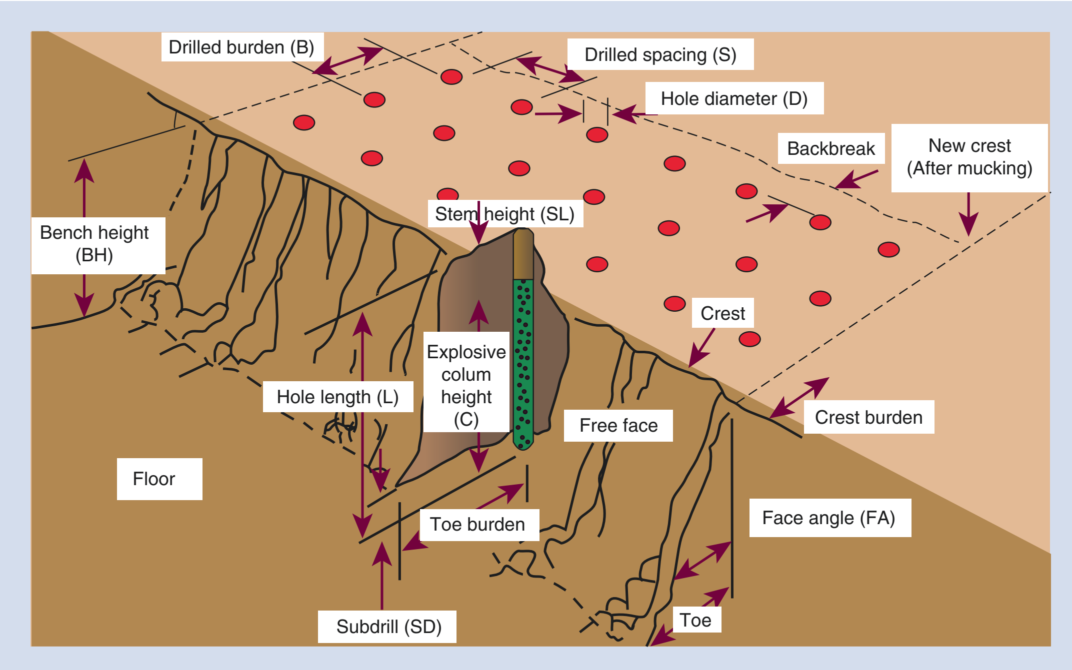

Fig. 5.86 Main blast pattern parameters in surface bench blasting

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

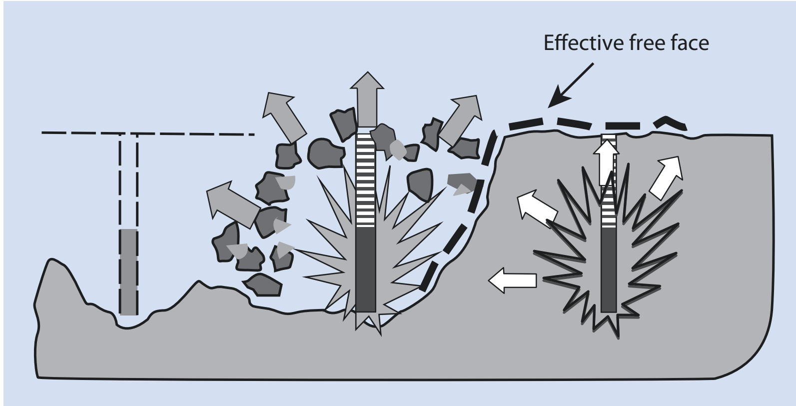

Fig. 5.87 Effective free face (Illustration courtesy of Atlas Copco)

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).

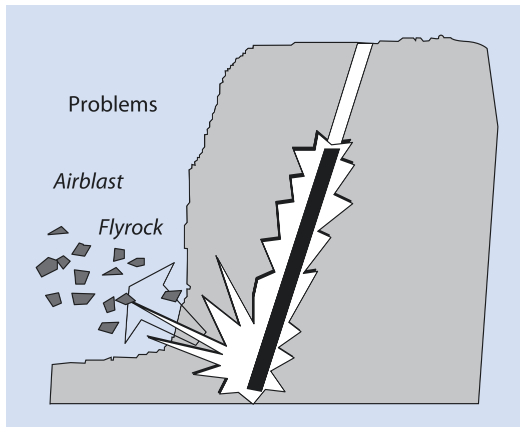

Fig. 5.88 Excessive blasthole angles cause problems (Illustration courtesy of Atlas Copco)

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/m^3	Medium 2700 kg/m^3	High 3200 kg/m^3
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.

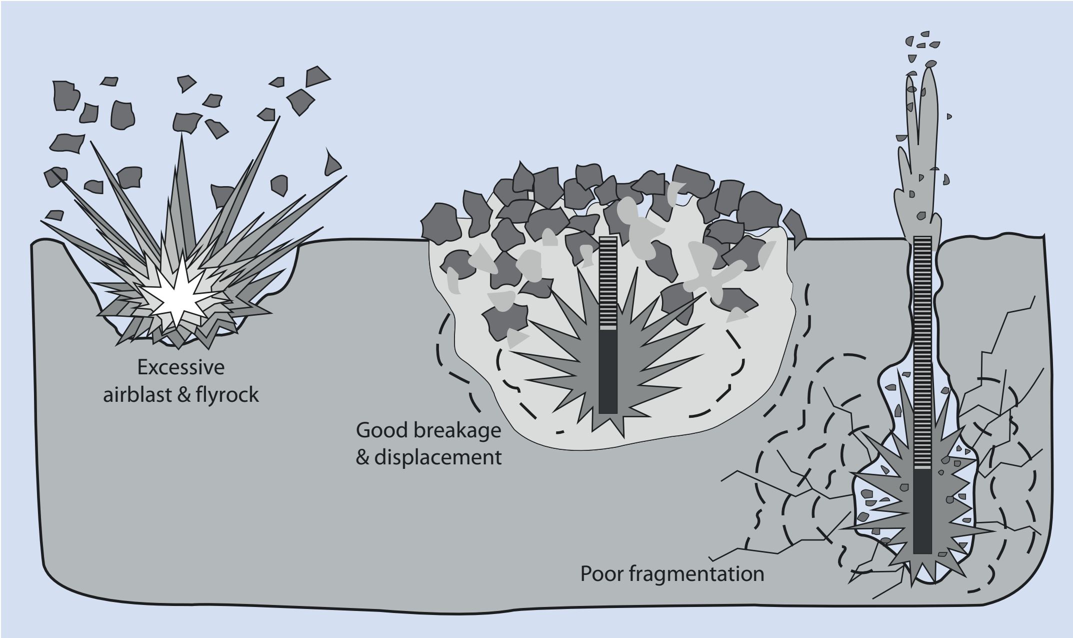

Fig. 5.90 Effect of correct and incorrect stemming (Illustration courtesy of Atlas Copco)

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

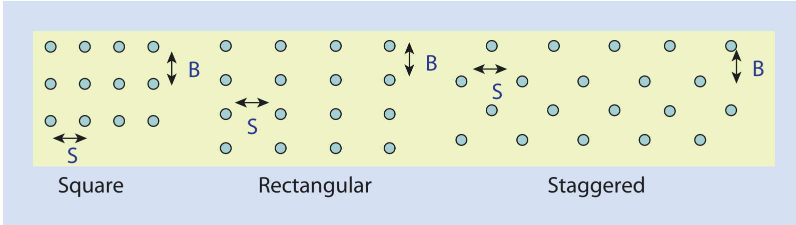

Fig. 5.91 Square, rectangular, and staggered blasthole patterns

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.)

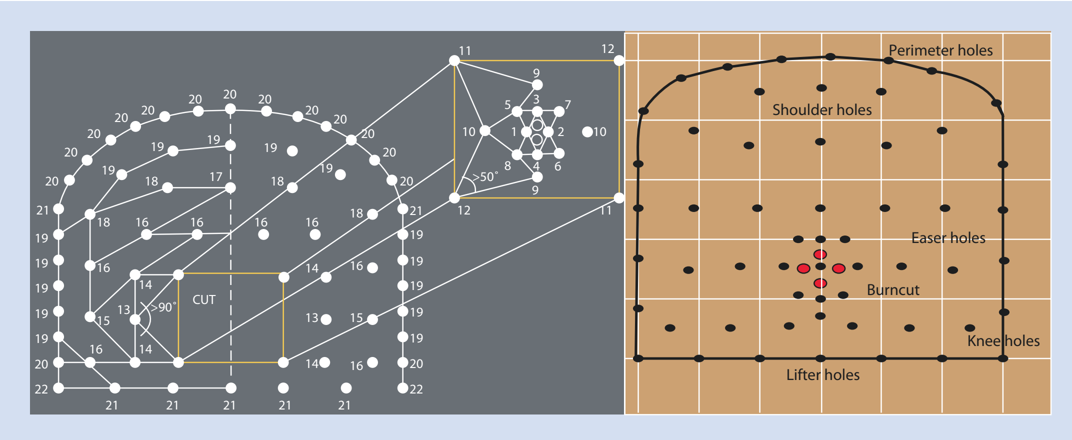

Fig. 5.93 Some examples of patterns of holes

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

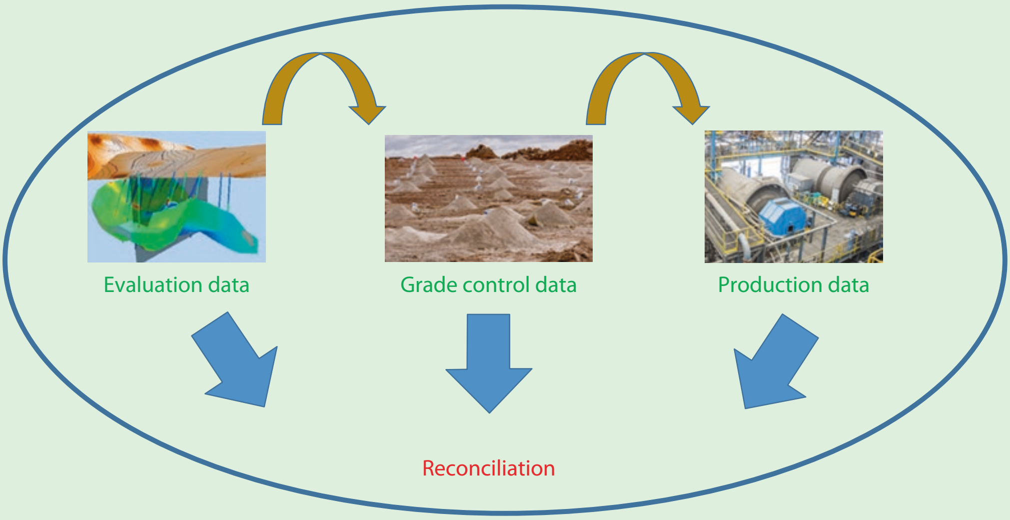

Fig. 5.97 Reconciliation process

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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