Wrocław University of Technology

# Mining and Power Engineering

Anna Gogolewska

# SURFACE AND UNDERGROUND MINING TECHNOLOGY

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Contents

# 1 Introduction

According to Whyte (2007), there are more than 7,000 producing mines in the world. These mines are the source for almost all products consumed by people including electricity. Technologies of mining, mineral processing and metallurgical processes were developed primarily in Europe, although mining in Europe is not booming nowadays (Whyte and Cummings, 2007). The earliest miners may date back to 300,000 BC. The first excavations were either pits or placers. By about 40,000 BC, mine workings had reached underground in form of short adits or shafts, and by 8000 BC in form of interconnected elaborated openings 90 m deep. Copper was the first metal to have been liberated from ore about 7000 BC. Coal as a mineral fuel was discovered in the late 13th century AD (Hartman, 1992). Mining is not only the old industry but also strictly connected with civilization development to which it has contributed remarkably. Mining engineering is the interdisciplinary field as it involves geology, hydrogeology, physics, mathematics, economics, mechanics, machinery construction, and maintenance etc. Therefore, it takes a lot of time to study and learn even a part of this discipline.

This work is designed for undergraduate and graduate students who are interested in mining technologies as well as in geological constraints with which any mining method either underground or surface must deal. The work presents only briefly elements of underground mining technology. Surface mining is discussed in detail, especially coal and lignite mining. Elements of mining such as stages of mining and unit operations of mining are described in Chapter 2. In Chapter 3, production operations are presented. Surface and underground mining methods are briefly depicted in Chapter 4. Chapter 5 depicts surface mine development, Chapter 6 presents case studies to describe mechanical extraction methods in surface mining, and in Chapter 7 comparison of surface mining methods is discussed. In Chapter 8, a case study of Polish copper underground mining technology is presented. The information and data used in this work are primarily taken from *SME Mining Engineering Handbook* by Hartman (ed.) (1992). The Polish copper mines' description is based upon the book by Piestrzyński (ed.) (1996) titled *Monografia KGHM Polska Miedź S.A.* (In Polish).

## Part I Fundamental Concepts

Hartman (1992) distinguishes two different terms: mining and mining engineering. According to him, mining embraces the processes, the occupation, and the industry involved in the extraction of minerals from the earth. Whereas mining engineering is the art and science that the processes of mining and the operation of mines use. The mining engineers are the professionals who connect these two fields so that they can help to locate and prove mines, to design and develop mines, and to exploit and manage mines. The prime task for mining is to build (drive) an opening or an excavation that are used to enter the mineral deposit from the surface to extract minerals from the earth. These entries may lie on the surface or be made underground, which defines the locale of the mine. Each mining method is specified by the layout, procedure, equipment, and system involved. Moreover, its application is strongly related to the geologic, physical, economic, environmental, and legal conditions (Hartman, 1992).

The field of mining uses its own unique vocabulary. Some common words such as function, operation, and location used in technical fields are often confusing for laymen. To minimise such the confusion there are dictionaries of mining terms available from for example SEM. One example is the American Geological Institute's Dictionary of Mining, Mineral, and Related Terms. Some of definitions the terms are cited below (Bise, 2003).

Advance mining: Exploitation in the same direction, or order of sequence, as development.

Air course: A passage through which air is circulated.

Air split: The division of the current of air into two or more parts.

Airway: Any passage through which air is carried.

Barrier pillars: Solid blocks of coal or rock left between two mines or sections of a mine to prevent accidents caused by inrushes of water or gas or to protect pillars in headings against crushing.

Beam building: The creation of a strong, inflexible beam by bolting or otherwise fastening together several weaker layers. In coal and copper under ground mining, this is the intent of roof bolting

Belt idler: A roller usually of cylindrical shape, that is supported on a frame and that, in turn, supports or guides a conveyor belt.

Belt take-up: A belt pulley, under a conveyor belt located near the drive pulley, kept under strong tension parallel to the belt line. Its purpose is to automatically compensate for any slack in the belting created by start-up, etc.

Bleeder entries: Special air courses developed and maintained as part of the mine ventilation system and designed to continuously move air-methane mixtures emitted by the gob away from active workings and into mine-return air courses.

Breakthrough: A passage for ventilation that is cut through the pillars between rooms.

Burden (of holes in blasting): The distance between the rows of holes running parallel to the free vertical surface of the rock.

Cage: A device in a mine's shaft, similar to an elevator car, that is used to hoist people and materials.

Car: A railway wagon, especially any of the wagons adapted to carrying coal, ore, and waste underground.

Collar: The timbering or concrete around the mouth or top of a shaft.

Cover: The overburden above any deposit.

Crosscut: A passageway driven between the entry and its parallel air course or air courses for ventilation purposes.

Development: The work done on a mine after exploration to provide access to the ore and to provide haulage ways for the exploitation period.

Dump: The point at which a load of ore or coal is discharged.
Entry: An underground passage used for haulage or ventilation.
Face: The principal operating place in a mine.
Gob: That part of the mine from which the coal has been removed and the space has been filled up with waste rock.
Headframe: The structure surmounting the shaft that supports the hoist rope pulley and the hoist itself.
Inby: In the direction of the working face.
Intake: The passage through which fresh air is drawn or forced into a mine or into a section of a mine.
Main entry: A main haulage road.
Outby: In the direction of the mine entrance.
Panel: A coal or ore mining block that generally comprises one operating unit.
Retreat mining: Exploitation in the direction opposite from development.
Return: The air or ventilation that has passed through all the working faces of a split.
Rubbing surface: The total area around an airway.
Skip: A car being hoisted from a slope or shaft.
Spacing (of holes in blasting): The distance between two holes in the same row.
Split: Any division or branch of the ventilation circuit.
Subsidence: The deformation of the ground mass surrounding a mine due to the mining activity.
Trip: A train of mine cars.

## 2 Mining Stages and Operations

According to Hartman (1992), stages in the life of a mine refer to the overall sequence of activities taking place in mining. Hartman divides the life of a mine into four stages: prospecting, exploration, development, and exploitation. The mining engineer is responsible for their accomplishment. Prospecting and exploration are conducted prior to actual mining and are closely connected with each other, making a combined activity. Development and Exploitation are similarly interrelated (Hartman, 1992).

2.1 Stages of Mining

Mineral Prospecting (Stage 1). The first stage, prospecting is the search for minerals (e.g., coal, metallic or nonmetallic ores) on and near the surface or in boreholes. All prospecting is based on geology and utilizes two techniques: direct method and indirect one. The direct method can be used to discover surface deposits by visual examination or geological studies. The indirect method can search for hidden deposits, using mainly, geophysical methods, which are suitable for airborne, surface, and subsurface (in logging boreholes) application.

Exploration (Stage 2). The second stage, exploration assesses the size and value of mineral deposits with techniques similar to those of prospecting. The exploration is conducted on surface and within subsurface with geology and geophysics. Samples for analyses are obtained by trenching outcrops and by drilling and excavating below the surface. Then reserve estimation as well as examination and valuation of the deposit are made. The two stages should end with a feasibility study to determine the deposit potential to become a profitable and active mine, thereby to decide whether to abandon the project or not (Hartman, 1992).

Mine Development (Stage 3). The purpose of work conducted in the third stage of mining, development is to open a mineral deposit for exploitation - This stage starts real mining. It is

necessary to obtain the access to the deposit. In the case of surface mining, near-surface mineral deposit is exposed by stripping overburden, the earth and/or rock overlying the mineral. In the case of underground mining, openings are excavated from the surface to more deeply deposited minerals to make them accessible. Before the actual mining begins, the following preliminary development work has to be done: elaborating an environmental impact statement, obtaining mineral rights and financing, and preparing access roads, transportation, power sources, dams, as well as waste disposal sites. The steps in mine development are sequential. When the mine is to be a surface one, stripping of the overburden takes place. The sequence of operations to break up and remove the overburden may be different from or the same as that used to exploit the mineral. This is primarily governed by the properties of the waste and the mineral. Fig.2.1 shows development for surface mining, employing the open cast method. In this case, mining is possible on multiple levels since one or more bench faces are prepared (Hartman, 1992).
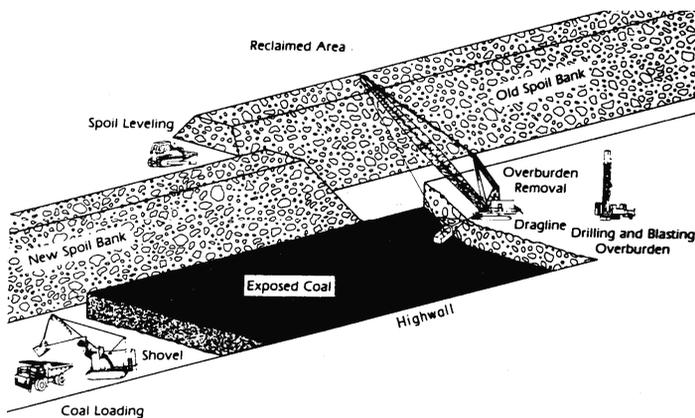


Fig.2.1. Surface mining, open cast method (after Anon, 1982)

Development for underground mining requires more complex work, hence is more expensive. Access openings have to be carefully planned and laid out for safety, suitability and usefulness as well as permanence. A shaft is the main opening to the surface. Shafts may be of rectangular or circular cross section, and vertical or inclined (termed a slope). Their size should be big enough to carry people and machines. In areas where the differences in height are significant, the deposit may be reached with horizontal openings termed adits or tunnels. Massive or steeply sloping underground deposits (usually of metallic minerals) are mined from levels (horizons) regularly spaced in a vertical plane. Each level includes the openings with drifts (major tunnels) and linking crosscuts (secondary). Access between the levels is achieved by means of vertical openings (raises or winzes) or inclined openings (ramps). All these development openings are linked with exploitation chambers termed stopes. This situation is presented in Fig.2.2. Deposits of coal and nonmetallic minerals are often flat-lying and bedded. In such the case, mining can be conducted using systems of linked horizontal openings (termed entries or crosscuts) and rooms or longwalls (Hartman, 1992).
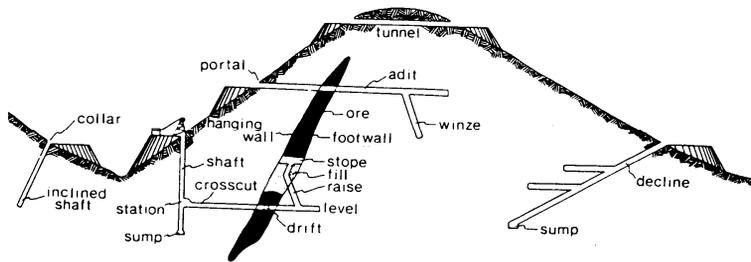
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Fig.2.2.Underground mining, stoping method (after Anon, 1983a)

Mine Exploitation (Stage 4). Exploitation is the final stage of mining. In this stage, mineral is actually recovered from the earth. Exploitation focuses on production only. However, it is accompanied by some exploration and development work, which should last until the end the life of a mine. The characteristics of the mineral deposit, safety requirements, technological and economic limits determine the mining method to be chosen for exploitation. The most important factors influencing the method selection are geological conditions such as the dip, form and shape of a deposit, and the strength of the wall rock and ore. Generally, there are surface and underground exploitation methods. In the case of surface mining, traditional exploitation methods embrace two groups of methods: (1) mechanical extraction methods including open pit mining, open cast (strip) mining, quarrying, and auger mining; (2) aqueous extraction methods including placer mining (hydraulicking and dredging) and solution mining (in situ techniques and surface techniques). In the case of underground mining, there are three groups of methods identified as follows: supported (cut and fill stoping, stull stoping, and square set stoping), unsupported (room and pillar mining, stop and pillar mining, shrinkage stoping, sublevel stoping, and vertical crater retreat mining), and caving (longwall mining, sublevel caving and block caving). Apart from traditional methods, there are novel methods such as underground gasification, marine mining, automation, or rapid excavation in hard rock (Hartman, 1992).

2.2 Unit Operations of Mining

Unit operations involved in both development and exploitation stages should be highly standardized since they extract natural materials (rock or soil, ore or waste) from the earth. The unit operations of mining are defined by Hartman (1992) as 'the basic steps employed to produce mineral from the deposit, together with the auxiliary steps involved'. The steps connected directly with mineral extraction are called production operations. They constitute the production cycle of operations. The production cycle is supported by steps termed auxiliary operations (Hartman, 1992).

The production cycle involves unit operations, which have to break rock and handle materials. Rock breakage is usually performed using drilling and blasting. In underground coal mining, coal may be cut prior to drilling and blasting. In quarrying, channelling may be used instead of blasting and drilling. Materials' handling embraces loading or excavation and haulage (horizontal transport), along with hoisting (vertical or inclined transport), when necessary. General production cycle consists of the following unit operations: cutting, drilling, blasting, loading, hauling, and hoisting. In non-coal or surface mines this general cycle may be shorter, (called then a conventional cycle), and embrace the following unit operations: drilling, blasting, loading, and hauling. Sometimes it is better to make production operation continuous. For example, a machine such as bucket wheel excavator that operates without drilling or blasting may excavate soil. If rocks need to be loosened, this can be done

by ripping, which eliminates explosives. Continuous mechanical excavation and breakage in coal or soft ores, also eliminates drilling and blasting. The same goal can be achieved in soft to medium-hard rocks by using boring machines. The production cycle in these cases is called continuous cycle and includes only: mining and hauling. The cycle of operations in surface and underground mining are quite similar. The differences result from the size of equipment. In surface mining the equipment scale is larger than that of underground mining equipment. In surface mining, when consolidated rock must be excavated, mobile rotary or percussion drills bore blastholes (tens of millimetres in diameter) for blasting agents or high explosives to be placed into. The charge is then inserted and detonated to have the ore or waste fragmented. Next, power excavators such as shovels, draglines, or bucket wheel excavators load the broken material into haulage units (belt conveyors, trucks, or railroad cars) or cast it on a spoil (waste) bank. Coal and spoil are excavated in the same way but without blasting, which is not required. Blasting is avoided in quarrying by using channelling machines or saws to free blocks of dimension stone. In underground mining, scaled-down equipment is used, however the cycle is only slightly different. Smaller drillholes are bored for blasting with small drills, and small loading machines and small-sized trucks, trains, or conveyors are employed to haul the ore or coal. The final operation may be hoisting by conveyor, skip, or cage (Hartman, 1992).

The productive stages of the actual mining need certain auxiliary unit operations to be carried out. In underground mining, these auxiliary operations include: roof support, ventilation and air conditioning, power supply, pumping, maintenance, lighting, noise abatement, communications, handling of supplies, and providing and maintaining adequate health and safety. In surface mining, most auxiliary units operations are the same, except for roof support and ventilation, in place of which slope stability, waste disposal, air pollutant control, and land reclamation must be performed (Hartman, 1992).

# 3 Production Operations

In mining, the unit operations of production are fragmentation, loading (mucking), and haulage (transportation). The auxiliary unit operations encompass site preparation, ground control, transportation of personnel and supplies, reclamation procedures, as well as provision for pumping, ventilation, and power. A unit operation defines a process and not the subject of that process. It is an elemental step that cannot be divided into substeps. In consequence, a unit operation is identified with a single operator or machine. Moreover, it can be performed continuously and is of repetitive nature. When overall mine design is appropriately prepared to minimize moves of equipment between adjacent cycles, the operation can be as high effective as a continuous one (Saperstein, 1992).

Saperstein (1992) identifies the following operations embraced by surface mining cycle: (1) Remove topsoil (Only when mining new ground), (2) Prepare drill bench (If the material is soft enough to not require blasting, go to step 5), (3) Drill blastholes, (4) Blast (Load explosives into boreholes, connect detonators in order to have desired firing order and times, fire when safe), (5) Load fragmented material (Soft material is fragmented by the cutting action of the loading machine; in large-scale area strip mining, the loading machines also haul materials, which is called casting), (6) haul material (Mineral goes to processing, waste is taken to permanent storage; Haulage can be cyclical: trucks or load-and-carry vehicles, or, continuous: conveyor belts or pipelines), (7) Dispose of overburden or waste (Overburden storage: waste piles, rebuild mine lands; Preparation waste: waste disposal ponds, integrate into overburden storage), (8) Restore topsoil, (9) Reclamation, and (10) Come back to step 1. For underground mining he identifies the following operations involved in mining cycle: (1) Enter work place after previous blast (check that ventilation has eliminated blasting fumes, check back and ribs for loose material), (2) Install initial ground support, (3) Load fragmented material (Onto gathering haulage, directly onto main haulage), (4) Haul material (In gathering system: Load-Haul-Dump (LHD), truck, shuttle car or gravity transfer; Main haulage: truck, train, conveyor belt, pipeline), (5) Install permanent ground support (If needed), (6) Extend utilities (ventilation, power: electricity or compressed air, transportation), (7) Drill blastholes, (8) Blast (load explosives into boreholes, connect detonators to provide expected order of firing and times of firing, blast when safe), and (9) Go back to step 1.

## 3.1 Rock Breakage: Mechanical

The methods that are used to excavate rocks employ mainly either mechanical rock cutting tools or explosive charges. Almost all rock excavation machines break rocks by the penetration of a wedge. Indenters and drag bits are two prime types of mechanical rock cutting tools. Drag bits are also called picks. Many indenters are wedge-shaped. An indenter breaks the rock using a force, which acts in the direction perpendicular to the rock surface. Whereas the main force used by a sharp drag bit to break the rock is in the direction parallel to the rock surface. The action of sharp drag bits is to cleave rock chips from the face. The directions of loadings for both kinds of tools are presented in Fig.3.1 (Hood and Roxborough, 1992).
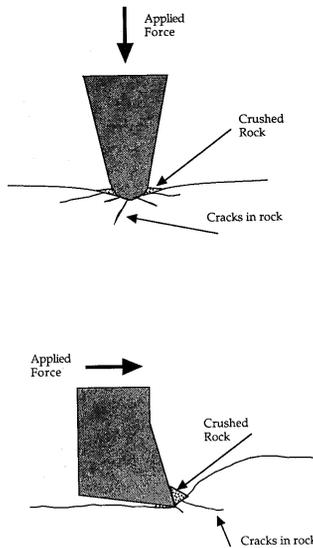
Fig.3.1. Methods of rock loading by an indenter (top) and a drag bit (bottom)
(after Hood and Roxborough, 1992)


Indenters are more commonly used than drag bits. All types of roller cutters such as disk cutters and rolling cone bits, etc., break rocks by indentation. Likewise, all types of percussive tools such as percussion drillbits, down-hole drillbits, and high-energy impact bits, fracture rocks by indentation. Rotary drillbits and picks that are used on coal excavation machines do not break rocks by indentation (force normal to the rock surface) but exclusively by applying the main force in the direction parallel to the rock surface. Fig.3.2 shows these breaking tools. Since they need more energy than do drag bits, the indenters are less efficient than drag bits, although they are more commonly used (Hood and Roxborough, 1992).



Fig.3.2. Mechanical rock breaking tools (after Hood and Roxborough, 1992)

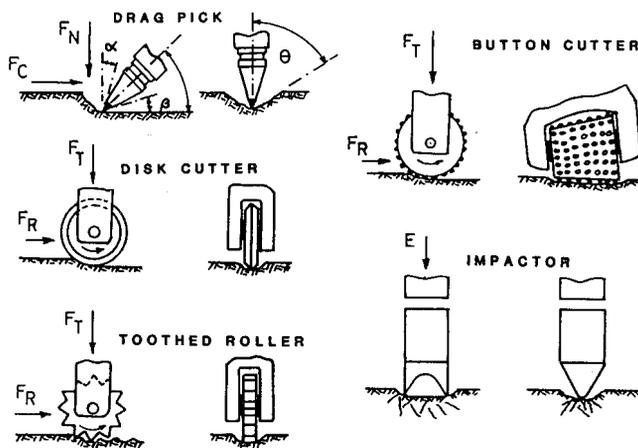Drills and mining machines utilizes an array of cutters on powered cutting head so that a rock face can be broken with numerous cutters at a time. Most cutting heads rotate about an axis oriented either parallel to or normal to the rock face. Such the machine format is characteristic of rotary drills, continuous miners, roadheaders, shearers, bucket wheel excavators, and raise and tunnel borers. There are other machine types such as ploughs and tractor rippers that employ linear motion and external traction. Rock breakage is a continuous process with rotary machines; the cutter head moves forward gradually as the rock is cut off.

Drilling. Rock excavation in most non-coal underground mining operations is conducted employing drill-and-blast techniques. Almost all holes drilled in underground mines are made with conventional percussive drills. A small number of the holes are drilled with down-hole drills. The rest of these holes are produced with small-diameter, rotary drills. Most of these small-diameter holes are drilled for the purposes of roof bolting (emplacement of roof bolts). In surface mines, drill-and-blast methods are used to excavate most of the rock. In these mines, blastholes are produced with large-diameter rotary drills (Hood and Roxborough, 1992).

3.2 Rock Breakage: Explosives

Explosives can be defined as follows: 'An explosive, or blasting agent, is a compound or mixture of compounds, which when initiated by heat, impact, friction, or shock, is capable of undergoing a rapid decomposition (decomposition is a self-propagating, exothermic reaction called explosion), releasing tremendous amounts of heat and gas.' Industrial explosives are classified as one of the following: nitroglycerin-based (dynamite), dry blasting agents (ANFO), wet blasting agents (water gels or slurries, emulsions, and heavy ANFO), permissibles (they do not ignite methane or fine coal dust), primers (a primer charge is an explosive ignited by initiatior, which in turn initiates a non-cap-sensitive explosive or blasting agent), and boosters (highly sensitised explosives or blasting agents of weights bigger than those used for primers; they are placed within the explosive column to provide additional breaking energy). Explosive components are: oxidizers, fuels, absorbents, thickeners, and stabilizers (Dowding and Aimone, 1992).

3.2.1 Initiators and Initiation Systems

Initiators are devices with high explosives that create detonation or burning when given an adequately big electrical or mechanical impulse. A system of explosives and other devices contain initiators that are used to begin detonation of all other components of a system. Initiation systems are electric or non-electric, and consist of blasting caps, safety fuse, detonating cord, or non-electric shock tubes. Certain primes with wells for cap or cord insertion are shape-cast or in plastic shells. Electric blasting caps are the most popular method of initiation. An internal-connecting bridgewire is heated by electrical energy sent through copper or iron legwires. This heat begins a chain reaction of explosives that burn in metal cap shell through a powder delay train. This process ignites a cap-sensitive explosive by detonating a high-explosive base charge. The electric blasting caps can be made with an instantaneous (no delay train) time of initiation, or with time delays in milliseconds used in delayed blasting. There are long- or short-period delays. Short-period delays are for surface blasting operations whereas longer ones are used in underground mines where blasting conditions are more limited. The application of time delays improves rock fragmentation and the monitoring of ground vibrations. Caps can be energized with AC power lines and capacitor-discharge DC power sources.

Nonelectric initiation systems contain a cap similar to that of electric cap, however in order to initiate the cap, they are linked to plastic tubing or a transmission line that carries shock or heat (an initiation). The energy source in the tubing is either a gas or mixture of an internal coating of special explosive. Nonelectric tubing is not applied in underground coal or gassy mines since the initiation produces an open flame, and hence is hazardous. The plastic tube itself does not detonate. Unique blast pattern arrays can be configured by means of caps and tubes of variable lengths linked with special connectors between holes. Surface delay elements are used with in-hole delays in order to have almost infinite numbers of delays in blasting patterns. There are delays of short and long periods, and of surface and in-hole type, available. Nonelectric systems enable one to design blasts with a greater number of holes than in the case of traditional electric blasting. If explosives are to be used to break rock, they and blasting devices have to be properly chosen, the borehole patterns, loading characteristics, and delay blasting sequence must be carefully designed, and ground vibration, airblast, and fly rock have to be controlled properly (Dowding and Aimone, 1992).

3.2.2 Surface Blast Design

Efficient and satisfactory blast designs obtain the desired particle size distribution and placement of muckpiles so that rock removal and handling become easy. Surface blast design requires the choice of hole spacing S, burden B, charge eight W or powder factor PF, top-hole stemming length T, and subgrade drilling depth J. Design parameters are presented in Fig.3.3. Borehole patterns are drilled square (S/B=1) or rectangular (S/B≥1) on centre or offset (staggered). The shape of the broken rock pile and the degree of rock fragmentation, are determined by the sequence of hole-initiation timing, S/B ratio, actual timing between charge detonations, and by the number of blasthole rows (Dowding and Aimone, 1992).

Surface Delay Blasting. Delay blasting techniques are used to improve fragmentation, control of rock movement and overbreak, and decrease ground vibrations. Delays are implemented into the blast design employing electric or nonelectric caps or delay connectors armed with detonating cord. The sequence of hole initiations is defined by the delay patterns used in design. Therefore, this pattern determines the overall direction in which blasted rock moves and the resulting fragmentation. The actual timing (in milliseconds) between detonating charges determines muck pile displacement height and distance from the bench. The variations of timing patterns utilized by surface mining are shown in Fig.3.4. Different initiation sequences give a different effective burden $B_e$ and different effective spacing $S_e$, as can be seen from Fig.3.4. The effective spacing is the distance between holes in a row determined by adjacent time delays (e.g., delays by rows). Effective burden is the distance in the direction of resultant (broken with blasting) rock mass movement. The V and echelon (diagonal) patterns have to be used when rock displacement is confined. When designs use two free faces, they improve fragmentation (Dowding and Aimone, 1992).

3.2.3 Underground Blast Design

Development of tunnels, shafts, raises, stopes, caving, and other underground openings is performed by means of blasting rounds. Powder factors may vary from 0.9 to 6 kg/m$^3$. The lower values are suitable for large open rooms in soft weak rock; whereas confined raises and shafts made in hard competent rock requires the higher ones. Underground explosives should be easily loaded and handled. In gassy mines permissibles are required. The underground application of explosives is restricted by their fume class. Dynamites and wet and dry blasting agents are widely used underground. A constant watch for unshot explosives has to be made before subsequent round is to be drilled and when blasted rock is being removed. The charge

can be loaded into a hole by tamping, pushing or using mechanized pneumatic and pumping machines. Underground holes mainly employ priming methods. The holes with large diameters are primed at the base. Blasting caps are put into with the exploding tip that faces the line of charge. The horizontal holes smaller in diameter have to be primed at the bottom of the hole so as to obtain maximum confinement. The primer containing the initiator can be thrown away by detonation if it is placed at the hole collar and the stem is not long enough. The design of underground blasting rounds can utilize two types of the rounds: those with one free face and those with more than one free face. Single-face rounds are used in development openings (tunnels, shafts, raises) and the methods of mining such as room and pillar, longwall, and shrinkage stoping methods. The development work in all mining methods needs single-face blast designs. Multiple-face rounds are necessary for open stopes, sublevel caving, and tunnels large in diameter that utilize benching methods. Sometimes, the design of multi-face rounds is similar to that of surface blasting (Dowding and Aimone, 1992).
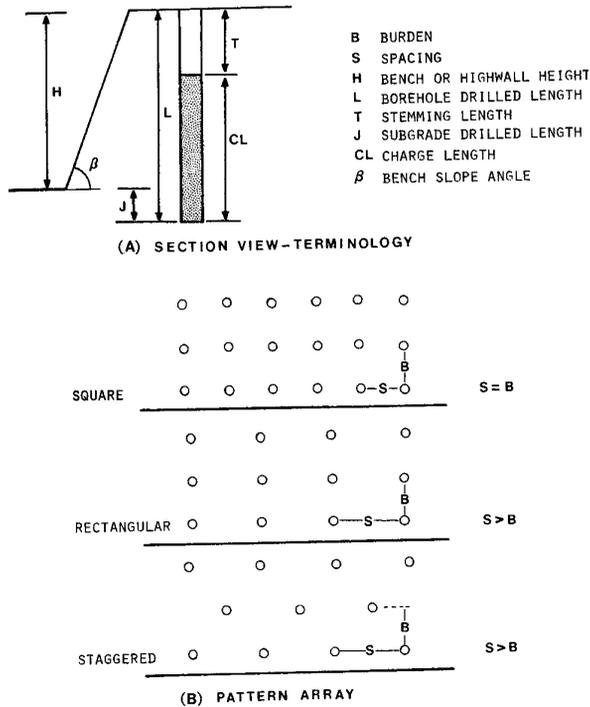


Fig.3.3. Blasthole section view (A) showing terminology used in design and (B) pattern array for layout of holes (after Dowding and Aimone, 1992)
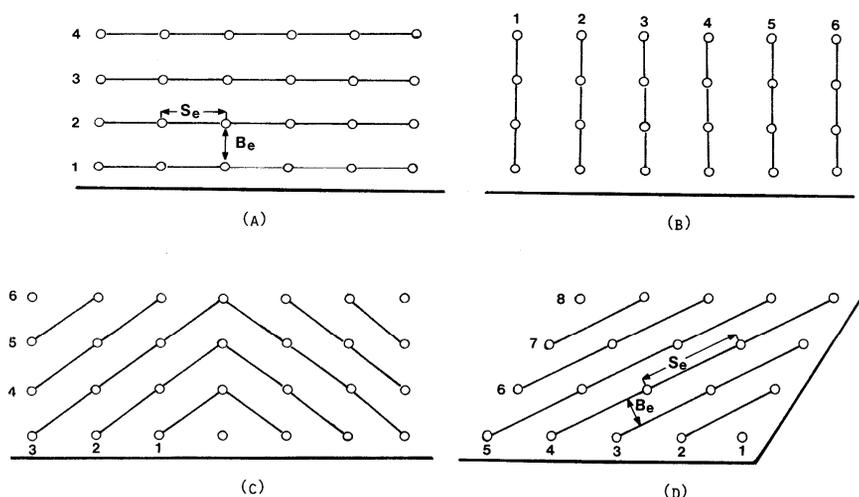
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Fig.3.4. Typical initiation patterns for surface blasting showing initiation by rows (A) parallel and (B) perpendicular to a single free face, (C) in a 'V' configuration and, (D) echelon pattern using two free faces; $B_e$ is effective burden and $S_e$ is effective spacing (after Dowding and Aimone, 1992)

Single-Face Rounds. Fig.3.5 presents names of the blastholes drilled in a round. Cuts define a group of holes, centrally placed at the face and detonated on the first several delay intervals. The cut should create initial relief to which the rest of the holes break. Cuts consist of parallel holes called the burn cut, or angled holes, defined as V cuts. The exemplary types of burn and V cuts are presented in Fig.3.6. When single faces are considered, the burden is the distance, on the face, between individual holes and adjacent relief. It is possible to produce the relief by means of empty holes in the cut, or the cut itself while being blasted and ejected, or the holes around the cut as the delayed sequences of holes are initiated. Borehole patterns are chosen according to the size of the face and rock type. Blasting results are strongly dependent on drillhole alignment. Holes should be correctly spaced to provide adequate fragmentation and avoid damage to wall. Hole alignment is influenced by hole size, hole length, direction drilled and geology. Powder factors and spacing of holes are closely interrelated and should be carefully considered. Short-period (ms) and long-period (s) delays are used. They can be electric or nonelectric. Delay patterns and hole spacings used for single-face blasting are presented in Fig.3.7. Sinking shafts and tunnels use similar blast designs, and full-face rounds use a burn cut or V cut. A bench round or sump cut, shown in Fig.3.7 can be used if dewatering is necessary during development. Down-hole drilling and loading is safer and more efficient than drilling and loading upholes from below (Dowding and Aimone, 1992).

Multiple-Face Rounds. Underground blasting techniques employing multiple faces are presented in Fig.3.8. For room and pillar and open stoping methods, benching, similar to surface methods, is used. Medium- to large-diameter holes are drilled horizontally or drilled vertically by means of subdrilling. Shrinkage stopes are blasted with the blasthole shrinkage or overhand method. When using the overhand or breasting methods, small-diameter upholes, horizontal holes, and holes in inclined back are drilled with hand-held jacklegs or stoppers, while the previously blasted muck pile provides working support. Cut and fill mining methods work from the stope bottom upward to excavate horizontal slices of ore. The ore is blasted and removed after each slice, and then waste material replaces it. Hydraulic filling produces

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a competent and even floor, which allows horizontal and vertical holes to be drilled along the back by means of mechanized drilling machines. Sublevel or blasthole stoping methods involve small-diameter hole ring and fan drilling techniques or large-diameter parallel holes drilled through the whole stope length. The sublevel caving method uses fan drilling of long, small-diameters holes between sublevels or drill drifts to undercut and blast ore. To begin caving, block caving techniques require an initial development blast above the undercut level (Dowding and Aimone, 1992).
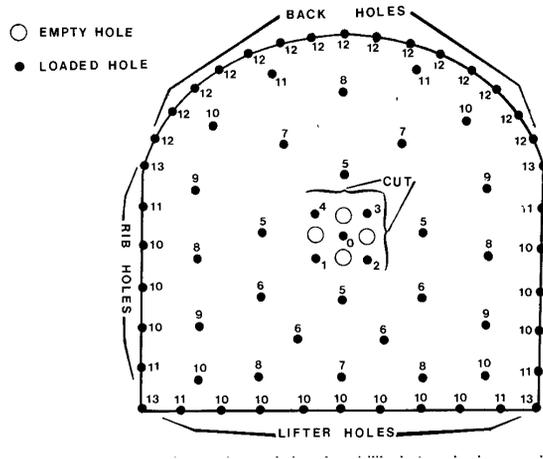


Fig.3.5. Typical tunnel round showing drillhole terminology and an initiation sequence for delay timing; all holes between the cut and perimeter holes are referred to as relief holes
(after Dowding and Aimone, 1992)
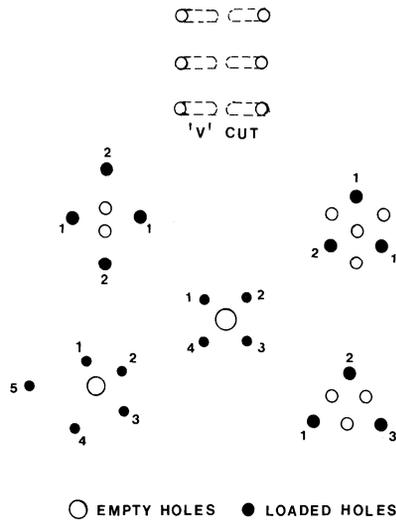


Fig.3.6. Example of burn and V cuts used in underground blast designs
(after Dowding and Aimone, 1992)

13
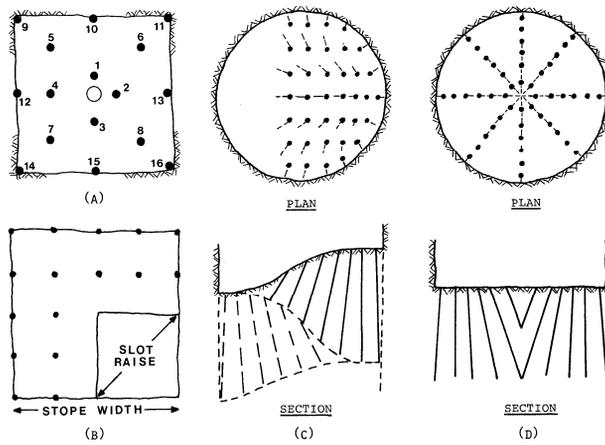
Fig.3.7. Single face drilling patterns  for (A) a long-hole raise, (B) drop raise for blasthole stopes, (C) shaft bench round, and (D) full face V shaft round (after Dowding and Aimone, 1992)



Fig.3.8. Multiple face drilling patterns for (A) room and pillar, cut and fill, and shrinkage stoping methods; (B) sublevel caving; (C) sublevel stoping; and (D) long-hole stoping or vertical crater retreat methods (after Dowding and Aimone, 1992)

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3.3 Materials Handling: Loading and Haulage

In the mining industry, loading and haulage perform combined tasks. A loading unit is employed to load fragmented ore into a unit conveyance (hauler), which carries the ore to a facilit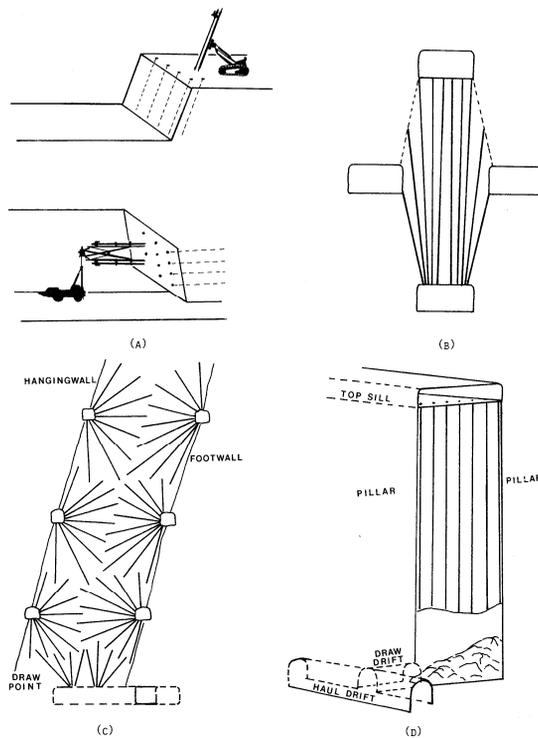y where the full advantage of it will be taken. The loader and hauler must be chosen to meet the requirements and needs of the operation. In the mining industry, it is common to load and haul large volumes of material, i.e. mineral and waste. The largest earthmoving machines are used in surface coal mines. However, selective operations deal with the smaller quantities of materials, and hence can employ much smaller loading machines. In mines, the main material to be handled, is rock that has been fragmented by explosive or mechanical methods. This rock can be coal, phosphate, industrial minerals, and an ore, or it can be waste rock that has to be excavated to uncover (expose) the mineral deposit of adequate economic value. The handling of materials is affected by physical properties such as adhesion, abrasion, cohesion, strength, angle of repose, bulk density, particle density, moisture content, compressibility, friability, fragment size, fragment shape, hygroscopicity, and void ratio. The method of breakage determines the fragment size. All these properties must be taken into account in designing and choosing materials handling system for individual property. To successfully choose the loading and haulage equipment, it is vital to assess the production requirements of the system prior to the selection procedure. These requirements are usually reported in units of weight or volume per unit of time (e.g., $m^3$/year). When the production requirements have been determined, various combinations of sequences of operations and equipment can be elaborated that are designed to produce the required volume of material. It can be done if the production capacity for each component of the proposed loading and haulage system is known. Moreover, the time required to complete one cycle of operation by any loading or haulage unit should be known (production rate). A discrete unit of material produced by a single machine with a properly determined cycle time makes the simplest materials handling systems. When within a system, multiple machines are required, continuous movement of material should be provided, or storage is necessary and the systems become more complicated (Sweigard, 1992).

3.3.1 Classification of Loading and Haulage Equipment

Sweigard (1992) proposes the classification system that is based on the two primary distinctions (1) the type of function of the machine (loading, haulage, or a combination of loading and haulage), (2) the form of material in the process (discrete units of material and continuous flow of material). The two main distinctions give six separate categories of equipment. The continuous flow of material is provided by multiple machines rather than by a single machine. Thus, separate continuous loaders and haulers are set in series to produce a continuous flow of material, for example, BWE and belt conveyor, auger head and screw conveyor, longwall shearer and belt conveyor. The other categories describe distinctions in the way of the performance of the primary function. For instance, all loading and haulage equipment that moves discrete units of material can be depicted on the base of a cycle time. In turn, these categories can be subdivided into one group of machines that does not need to travel during their cycles and another group that needs translocation. The result of the classification by Sweigard (1992) is presented below.

Loading Equipment: (1) Discrete Unit loaders that require no tramming: backhoes, hydraulic excavators, mining (or loading) shovels, and small draglines that are used to load-haul vehicles, (2) Discrete Unit loaders that require minimal tramming: front-end loaders (FELs) and load-haul-dumps (LHDs) (3) Continuous Flow loaders (they require no

tramming): bucket wheel excavators (BWEs), bucket chain excavators (BCEs), bucket dredges, continuous miners.

Haulage Equipment: (1) Discrete Unit haulers with fixed paths to travel: rail, skip hoists, slushers, and aerial tramways, (2) Discrete Unit haulers with non-fixed paths to travel: trucks, and shuttle cars (electrically powered shuttle cars do not travel on fixed paths but they are restricted by the trailing cables and cable reel location, battery-powered cars are not constrained), (3) Continuous Flow Fluid Transport (haulers with fixed paths to travel): slurry pipeline and pneumatic transport, (4) Continuous Flow Bulk Solids Conveyors (haulers with non-fixed paths to travel): belt conveyors, screw conveyors, chain conveyors.

Combined Loading and Haulage Equipment: (1) Mobile Discrete Unit Combined Loader-haulers: tractor scrapers, dozers, load-haul-dumps (LHDs), (2) Fixed-Based Discrete Unit Combined Loader-haulers: draglines, stripping shovels, (3) Continuous Flow Combined Loader-haulers: separate continuous flow loading and haulage units are operated in series, such as BWE and belt conveyor or longwall shearer and chain conveyor.

3.3.2 Hoisting Systems

Hoisting system is necessary for slopes or shafts if the mine is not accessible by a drift or adit. The prime components of a hoisting system are: a wire rope and hoist. The wire rope transmits forces along its axis. It has a wide range of mining applications, especially for shafts and slopes, and decides, to large extent, the length of a mine life. Therefore, particularly in the case of shafts and slopes, it should be made of the highest-quality improved plow steel to work in the difficult conditions of loading, winding, vibration, abrasion, and corrosion. Moreover, ropes have to be properly chosen for slushers, car spotter, etc. The maximum rope pull and hoist-motor parameters (e.g., horsepower) should be determined. To do this, the following information is necessary: (1) hoisting layout, (2) shaft/slope inclination, (3) weight of the loads, (4) weight of skips, cages, and cars, (5) rope size and weight, (6) hoisting distances, (7) drum dimensions, (8) rope speed, (9) production expected, (10) load and dump/caging times, (11) the effective weight of all rotating parts of the hoist such as the drums, gears, and sheaves, at a given radius from the drum, and (11) acceleration and deceleration rates (Bise, 2003). The most popular conveyance for carrying the ore to surface is the skip. Skips are self-dumping buckets and work in counterbalance in two separate compartments of the shaft. The weight of the skip being hoisted is compensated partly by the weight of the empty skip running down. The skips are filled from the loading points at the base of the ore pass. Some mines use belt conveyors to hoist the ore. In mines that are not developed by a shaft but by a decline, the ore is trammed by a truck or rail car directly to the surface. In the case of steeply dipping deposits, an internal shaft (called winze) is needed because the ore can be far away from the main shaft at depth. The winze is provided with its own underground hoist with skips and cages. For the purposes of drainage, there is a sump at the bottom of any shaft or winze to take groundwater that seeps into the mine. This groundwater is then pumped to surface water circuit. It is also possible to discharge this water into mine's water circulation system. Shafts consist of a separate compartment for the skips; a separate compartment for the cage; one for the man-way (ladders to provide an emergency access); and one compartment for service equipment (water discharge, compressed air, electric power cables, communication cables). One or more raises to the surface should be built for ventilation. These raises should be located at a distance from the main shaft (Whyte and Cumming, 2007).

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### 3.3.3 Rail and Belt Haulage Systems

In an underground or surface mine, personnel, supplies, as well as mined mineral, are transported by rail systems, belt conveyor systems, or the combined systems of the two. Large underground mines may employ rail haulage from the mining parts to the hoist to carry supplies and people, and conveyor belts for hauling mined materials. In small underground mines, mainly the belt conveyor systems are utilized. The application of these haulage systems in surface mines is also influenced by the size and layout of an individual mine. In underground mines, belt conveyors are used effectively for intermediate haulage and for main-line haulage of mined material. Overland belt conveyor systems are employed in surface mining. The elements of a belt-conveyor system are: the belt, terminal pulleys, idlers (the cylindrical metal rollers used to support the belt), support structure, tensioning device, motor and drive assemblies, and protection tools (Bise, 2003).

Rubber-Tired Haulage Systems. Rubber-tired haulage vehicles mainly provide face haulage in underground mines and pit haulage in surface mines. Cable-reeled, battery-powered, or diesel-powered haulage machines are employed in underground mines. Underground face-haulage vehicles transfer the mined material from the face (from loaders) to the intermediate haulage system (belt conveyor system). In surface mines, trucks predominate (Bise, 2003).

# 4 Surface and Underground Mining

The method of mining is unique for each different size and each shape of orebody. Mineral deposits differ in the shape and orientation of an orebody, the strength of the ore and surrounding rock, and the type of mineral distribution. These geological features influence the selection of a mining method and the plan for the ore development. Operating mines vary in size from small underground mines (with production under 100 tonnes of mineral a day) to large open pits excavating tens of thousands of tonnes of ore a day (Whyte and Cumming, 2007).

## 4.1 Surface Mining Methods

If a mineral deposit lies close to the surface, is of sufficiently big size, and its overburden is not too thick, surface mining (an open-pit mine) can be the most suitable method to extract the ore (Whyte and Cumming, 2007). Surface mining is the exploitation method in which minerals are mined from the surface. It seems to be the most commonly used mining method in the world. Two types of methods may be used in surface mining: mechanical extraction and aqueous extraction. The former, predominates, the latter cannot be employed unless there is sufficient water quantity available (Hartman, 1992).

### 4.1.1 Mechanical Extraction Methods

The mechanical extraction involves mechanical processes to obtain minerals from the earth. There are four mechanical extraction methods: (1) open pit mining, (2) quarrying (of dimension stone), (3) open cast mining, and (4) auger mining. In open pit mining, a thick deposit is mined in benches or steps. However, a comparatively thin deposit may be excavated from a single face as in the case of quarrying, augering and open cast mining. In the case of these methods, a large capital investment is necessary, however, they can provide high productivity, low operating costs, and satisfactory safety conditions (Hartman, 1992).

Prior to or during mining, it is necessary to remove any overburden by a stripping operation. In open cast (or strip) mining, overburden is removed by casting into mined-out areas, and mineral is excavated in consecutive operations. Open pit or open cast mining is used to mine a mineral deposit close to the earth's surface that is of low stripping ratio, shows large extension, and is fairly uniform. It should be noted here that a stripping ratio of 2 to1 means that there is twice as much waste rock mined as mineral (ore, coal, etc.) (Hartman, 1992).

Quarrying is a highly specialized small-scale method, slow and the most costly of all mining methods. Only square set stoping method is as much expensive as quarrying. However, quarrying is the only method able to provide both a sized and shaped product (Hartman, 1992).

Augering is employed to recover coal from the highwall at the pit limit. This method is also specialized but involves low costs. Open pit and open cast methods can be widely and variably applied. They use a conventional mining cycle of operations to extract mineral: rock breakage is achieved by drilling and blasting, which precedes the material handling operations of excavation and haulage (Hartman, 1992).

### 4.1.2 Aqueous Extraction Methods

The aqueous extraction methods must be provided with the access to water or an aqueous mixture during mining and processing. They recover the valuable mineral by jetting,

slurrying, melting or dissolving. There are of two types of these methods: (1) placer mining methods and (2) solution mining methods (Hartman, 1992).

Placer mining is used to mine mineral deposits that are not consolidated, such as sand, gravel or alluvium in which a valuable heavy mineral exists freely. Valuable heavy minerals such as diamonds, native gold, native platinum, and titanium can be found in placer form. There are possible two methods for placer mining: hydraulicking and dredging.

Hydraulicking (hydraulic mining) uses a high-pressure stream of water to undercut and force an exposed bank to fall down. Dredging utilizes floating vessels from which the ore minerals are extracted mechanically or hydraulically. Since the waste material is usually lighter than the valuable heavy mineral to be extracted, this mineral can be removed from water-base slurry by concentration in both methods Solution mining employs in situ techniques and surface techniques. Salt wells, uranium dissolution, and the Frasch process to melt sulphur are examples of the in situ techniques. Surface techniques utilize mainly solvent leaching of mineral values from heaps or dumps (Hartman, 1992).

Hydraulicking, dredging, and the solution mining are the most economical exploitation methods. They use unique and different cycles of operations entirely different from mechanical extraction methods (Hartman, 1992).

## 4.2 Underground Mining Methods

Underground mining is defined as mineral exploitation in which extraction operations are performed under the earth's surface. To select a proper mining method, one should know if ground support is necessary or not, and determine its appropriate type, extent, and form. Moreover, they should design an appropriate opening arrangement and extraction sequence to follow the size, volume, depth, shape and orientation of the mineral deposit. Ground support plays crucial role in underground excavation. Therefore underground extraction methods are classified with regard to the extent of support used. Hartman (1992) categorizes these methods in three classes. They are (1) unsupported (self-supported), (2) supported, and (3) caving, with individual methods distinguished according to the type of wall and roof supports utilized, the configuration of production openings, and the direction in which mining operations advance (Hartman, 1992).

### 4.2.1 Definitions

The definitions of certain terms used in underground mining should be given so that the content of the chapter could be clear and understandable. According to Brackebusch (1992) the key words can be defined as follows.

'Back: The roof or overhead rock surface of an underground opening.
Breast: The vertical end or face of a horizontal cut. The breast is a mining face that is as wide as the ore body and as high as the cut height.
Crosscut: A nominally horizontal tunnel, driven at right angles to the strike.
Chute: The loading arrangement that utilizes gravity flow in moving broken rock from a higher level to a lower level. A gate is used to control flow.
Cut (and Fill): That part of the cut and fill mining cycle that includes drilling, blasting, slushing, mucking, and ground support.
Cut: The volume of the ore body that is mined and filled in one cut and fill mining cycle.
Drift: A nominally horizontal tunnel, driven parallel to or coincident with a vein.
Heading: The working face of a drift, crosscut, or ramp. In timber sets, a heading is a bundle of wooden boards placed between a cap and the wall rock.

Load-Haul-Dump (LHD) units: Rubber-tired, front-end loader-transports designed for efficient tramming of broken rock.

Manway: The compartment of a raise or shaft that is equipped with ladders and landings.

Mucking: The process of loading and transporting broken ore from the point where it is excavated to an orepass or haulage vehicle.

Posts: Timbers that are placed vertically. When used individually for roof support in a flatly dipping ore body, they serve the same functions as stulls. When part of a timber set, posts support caps vertically.

Raise: An opening driven upward on the vein. A raise is an access way to a stope. A raise consists of one or two chutes, a manway, and a timberslide.

Ramp: An inclined underground tunnel driven downward with LHD equipment.

Slushing: The process of moving broken ore from the point where it is excavated to an orepass, using a winch-driven, rope-drawn scraper bucket.

Stope: An opening in which excavation of ore takes place. A stope may be any shape.

Stulls: Round timbers less than 203 mm in diameter, placed perpendicular to the vein for wall support.

Timberslide: a compartment in a raise used for hoisting materials.

Tramming: The haulage of broken ore with a train, truck, or LHD from a working place to an orepass.'

## 4.2.2 Unsupported (Self-Supported) Methods

The unsupported class embraces self-supporting methods, which do not need any major artificial system of support to carry the load comprised of both the weight of the overburden and any stresses in the rock caused by tectonic forces (this load is called superincumbent load). In this case, the walls of the openings and natural pillars are of sufficient strength to carry this load instead. However, miners may help them along with bolts and screens. They may use rock or roof bolts or light structural sets of timber or steel unless such artificial support impacts on the load-carrying ability of the natural original structure. Unsupported (self-supported) methods are used to extract mineral deposits that are tabular, flat or steeply dipping, and are surrounded by competent wall rock. There are five methods in this class: (1) room and pillar mining, (2) stope and pillar mining, (3) shrinkage stoping, (4) sublevel stoping, and (5) vertical crater retreat (VCR) mining (Hartman, 1992)

### 4.2.2.1 Room and Pillar Mining

Room and pillar mining is used for regular flat-lying deposits in reasonably competent rock, with the advance horizontal; the roof is supported by natural pillars of coal or ore that are left standing (between rooms) or are systematically recovered. Rooms make working faces and are cut from access entries. When necessary, additional support is provided using roof bolts or timbers. Coal or ore is extracted from rectangular shaped rooms or entries in the ore body or coal seam, leaving parts of the coal or ore between entries in form of pillars to support the roof or hanging wall. The pillars are arranged in a regular pattern, or grid, to facilitate operating and planning. They are usually rectangular or square but other shapes are also possible. The stability of the hanging wall, the strength of the mineral in the pillars, the depth of mining, and the thickness of the deposit determine the dimensions of the rooms and pillars. The mineral left in pillars may be treated as irrecoverable or recoverable. Room and pillar mining is performed in sections or panels. The panel or section determines the area to be worked and ventilated (Farmer, 1992).

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4.2.2.2 Stope and Pillar Mining

Stope and pillar mining is a similar method to room and pillar mining but employed in thicker, more irregular non-coal deposits. Stopes are the production centres of the mine where the ore is first broken. Pillars are spaced randomly and contain waste or low-grade ore, because the ore of higher grade is extracted in stopes. Stope and pillar is also called breast stoping, breast and bench stoping, board and pillar, stall and pillar, and panel and pillar (Hartman, 1987). This method has some characteristics such as (1) pillars, left for support, are of irregular sizes and shapes; (2) the method is used for mineral deposits that show large horizontal extent and dip less than the angle of repose; (3) rock masses are required to be competent; (4) the load capacity of the pillars limits the depth of mining; (5) there is no surface subsidence (Haycocks, 1992). Stope and pillar method is cyclic. It involves the following sequence of unit operations (Hartman, 1987): Drill-Blast-Support-Load-Haul-Dump.

4.2.2.3 Shrinkage Stoping Mining

In shrinkage stoping mining operations are conducted upwards. It is vertical, overhand mining method. Stopes are mined upward in horizontal slices. Slabs of ore are broken along the length of the stope. The broken ore is left in the stope to serve as a working platform for the miners. Then it is removed by chutes and placed on the level below into haulage drifts. The broken ore is left in the stope not only to form the floor for miners but also to provide additional wall support until the stope is finished and can be drawn down. This method can be used for narrow (1.2m) and wider (up to 30m) ore zones (Haptonstall, 1992).

4.2.2.4 Sublevel Stoping Mining

Sublevel stoping is also called blasthole or longhole stoping. It is an open stoping, bulk mining method used to excavate large, regular, steeply dipping ore bodies. The ore and rock should be competent so that no or little support is needed. This method uses several working benches, oriented vertically or staggered. On each bench, horizontal (breast) mining is carried out. Rock is broken by means of long blastholes drilled into the ore in a fanlike or parallel pattern (Haptonstall and Aelick, 1992).

4.2.2.5 Vertical Crater Retreat (VCR) Mining

Vertical crater retreat (VCR) mining comes from sublevel stoping. It is a horizontal, flat-back type of sublevel stoping applying spherical crater charges to break the ore. This method can be used for ore that has moderate strength. Blasting is performed at the base of vertical holes, creating horizontal cuts and advancing upward. Large, parallel, vertical drillholes are made to place nearly spherical explosive charges. Such the shape of charges is the most efficient one for blasting. This type of blasting technique allows horizontal slices of ore to be broken into an undercut (Hartman, 1992). First, a block (stope, panel, section, pillar) of ore is defined. The design of the block should enable the ore to flow down to drawpoints under the influence of gravity. The ore block must be of shape of two sills spaced a big vertical distance apart. Then the blasting characteristics of the rock are determined as well as hole size and drilling system. Next, the top sill and bottom sill are cut. Then the stope is drilled. The holes are drilled from the top sill down to the bottom sill. The stope can be cut when the drilling is finished. Blasting takes off vertical slices in the direction from the bottom sill to the top one. During blasting the open stope is full of broken rock to support the walls. When blasting is

complete, the broken rock is extracted. Once the ore is extracted, the bottom sill accesses can be closed off, and the stope can be backfilled from the top sill to provide rock stability (Osborne and Baker, 1992).

These various underground methods differ a little in the cycle of operations. The differences may occur in the direction of mining (vertical or horizontal), the ratio of opening-to-pillar dimensions, and the character of artificial support utilized. In room and pillar mining and stope and pillar mining openings are horizontal, opening-to-pillar ratios are low, and support in all openings is light to moderate. Shrinkage and sublevel stoping employs vertical or steeply inclined openings (they allow gravitational flow of bulk material), opening-to-pillar ratios of high values, and light support primarily in the development openings (Hartman, 1992).

## 4.2.3 Supported Methods

According to Hartman (1992) the supported methods of underground mining are those that need significant amounts of artificial support to maintain stability in exploitation openings and systematic ground control over the entire mine. These methods when production openings are not left standing during their active life and when the surface cannot be affected by major caving or subsidence. Supported methods are applied when unsupported and caving ones cannot be. Support systems for production workings are able to control wall closure and ground movements. Pillars and backfill are used for supporting. However, backfill is the most satisfactory form of support because it is able to support the superincumbent load almost entirely without yielding. Some yielding is yet preferable. Heavy support systems may involve timber stulls and cribs, timber or steel sets and trusses, as well as steel jacks, arches, props, chocks, shields, and canopies. Steel is stronger and yields less than timber, which is not sometimes desirable. Moreover, timber is more flexible, workable and easy to install, and more economical. The supported mining methods are designed for extraction of rock varying in competency from moderate to incompetent. (A competent rock is able to sustain openings with no heavy supports). There are three methods of this type: (1) cut and filling stoping that is the major one, and (2) stull stoping and (3) square stoping that are the minor ones. They are used in metal and non-metal mining (Hartman, 1992).

## 4.2.3.1 Cut and Fill Stoping

Cut and fill stoping is mainly used for steeply dipping vein deposits and large, irregularly-shaped deposits. It can be used for weak tabular deposits. This type of mining refers to a method in which a single excavation pass (cut) is completed and backfilled before another cut is carried out. The cycle starts with the first round after backfilling on the previous cycle. The drill-blast-mucking-ground support cycle is a sub-cycle of the cut and fill cycle. Major work elements of the cut and fill mining cycle include: (1) Drilling, (2) Blasting, (3) Mucking/Slushing, (4) Ground support, (5) Cleanout, (6) Raising up, (7) Preparation for backfilling, and (8) Backfilling. Mining is carried out upward. As the exploitation progresses, walls are supported by placing sand, tailings, or waste backfill into the stope. The ore is recovered in horizontal slices then moved to chutes or ore-passes mechanically. Waste is distributed hydraulically (Brackebusch, 1992).

## 4.2.3.2 Square Set Stoping

Square set stoping is a timbered-support method. This method employs backfilling. It also uses timber sets to support the walls as mining progresses. These timber sets are mounted in a

continuous support construction in form of skeletal prisms that are gradually filled with waste material to make the support long. This method involves the highest overall cost (Hartman, 1992).

4.2.3.3 Stull Stoping

Stull stoping is also a timbered method used in narrow, tabular, pitching ore deposits. It is a small-scale, supported method employing single timbers of rock bolts (Hartman, 1992).

Cut and fill and stull stoping are designed for moderately competent rock. Square set stoping method can be applied to the least competent rock. Cut and fill method can be to some extent mechanized (Hartman, 1992).

4.2.4 Caving Methods

The previously described methods maintain exploitation workings open, that means they are intact, during the mine's life. If the ore and rock are competent enough, unsupported methods are suitable; if ore and rock are incompetent to moderately competent, supported methods have to be used. In caving methods the exploitation workings are designed to collapse. Caving of the ore or rock or both is done deliberately (Hartman, 1992).

Caving methods involve induced, controlled, huge caving of the ore body, the overlying rock, or both, simultaneous with and crucial to mining performance. This type of mining precedes unavoidable eventual subsidence of surface. There are three major caving methods: (1) longwall mining, (2) sublevel caving, and (2) block caving (Hartman, 1992).

Longwall mining is used in tabular, horizontal deposits, mainly coal; the others are used for inclined, vertical, or massive deposits, metallic or nonmetalic. They are the cheapest mining methods (Hartman, 1992).

4.2.4.1 Longwall Mining

Longwall mining is a caving method especially suitable for mining thin seams, usually coal or nonmetalics at some depth. In this method, a face of considerable length (a 'long' wall) is maintained. As the mining progresses, the overlying strata are caved, thus facilitating the breakage of coal itself. The panel layout is simple and can be easily ventilated, and miners work under protective supported roof. Longwall panels are blocked out by the panel entries that are excavated in-seam on both sides of the main entries. In the case of retreating longwall method, the immediate entries on both sides of the panel are called the head entry and tail entry. The former is used to the passage of intake air and the transportation of coal, personnel, and supplies. The latter is used for the passage of the return air. Coal at the face is cut by the shearer or plough and then is loaded onto a conveyor and transported to the head entry T-junction. Then coal is transported onto a stage loader, which, next, empties onto the entry belt conveyor. The conveyor and powered supports are moved after each cutting cycle of the machine (shearer), and the roof strata behind the supports are permitted to cave immediately. The extracted area between the rear edge of the supports and faceline is called the face area or working face. The area behind the roof supports is called gob. In this method roof is controlled by means of a roof caving method. The panel entries can be supported using roof bolting (Peng and Chiang, 1992).

4.2.4.2 Sublevel Caving

Sublevel caving is a different method from that of longwall one. It is a mass mining method using gravity flow of the blasted ore and the caved waste rock. It is applied to dipping tabular or massive deposits. In sublevel caving, all ore must be drilled and blasted to use the gravity flow of broken ore. The fragmented ore after blasting makes so-called 'coarse material'. The main purpose of mining is to have a coarse material of a fragmentation suitable for its gravity flow, by means of minimum drilling and blasting. Moreover, undisturbed ore extraction should be achieved. As mining progresses downward, alternate slices of ore are mined out and the intervening layers of ore are recovered by caving. The overlying rock is also subsequently caved (Hartman, 1992).

4.2.4.3 Block Caving

Block caving is a remarkable, large-scale, mass-production method. The extraction of the ore in this method depends on the action of gravity. The general concept of this method is based on the following sequence of actions. A thin horizontal layer at the mining level of the ore column is removed. As a result, the vertical support of the ore column above is removed and the ore then caves by gravity. As broken ore is removed from the mining level of the ore column, the ore above breaks and caves by gravity. It is highly productive, low-cost, and suitable for massive deposits. A large block of ore is undercut to make it cave. As the block fragments and collapses, the ore is withdrawn through chutes or loading points into haulage drifts. Block caving can be used for fairly weak or fairly strong rock and for large-sized massive or dipping tabular deposits as well as for deposits with fairly large lateral and vertical extent (Hartman, 1992).

Part II Surface Mining

# 5 Surface Mine Development

Surface mine development embraces a significant number of problems and subjects. In this chapter some basic concepts relative not only to the open pit mining of thick or steeply dipping seams or beds but also to the strip mining of shallow, thin and flat-lying deposits. To design a mine one should take into account geometries, ore and waste properties, equipment and economics. This can be achieved in various ways depending on the length of mining time period, supply and demand changes, client requirements, and environmental requirements. Nowadays, the mine planners use computer software and therefore are able to simulate the diverse options for mine planning and equipment selection to elaborate the best design and to support the decision makers (Hustrulid, 1992).

## 5.1 Open Pit Planning and Design

According to Fourie (1992), an open pit mine is an excavation or cut made at the surface of the ground to extract ore which is open to the surface for the mine's operational time. In order to expose and mine the deposit, significant quantities of waste rock has to be excavated and relocated. As the lowest possible costs with maximum profits are highly expected, the planning of an open pit mine is closely related to economics, which in turn is influenced by several geologic and mining engineering conditions (Fourie, 1992).

Surface mining is considered more advantageous than underground mining in relation to recovery, grade control, economy, flexibility of operation, safety, and the working environment. However, there are many deposits that are too small, irregular, and/or deeply buried to be extracted economically by surface mining methods. Moreover, even where mineralization goes to a greater depth in open pit mines, the increasing amount of overburden to be handled can make mining unprofitable. In such the situation, mining has to be either abandoned or converted from surface to underground operations (Fourie, 1992). Furthermore, the selection of excavation methods and equipment is more complex for surface mining than for underground mining. In surface mining, blasting patterns have to be properly planned to get adequate fragmentation of overburden.

Open pit design is elaborated in several stages that consist of devising a scheme or set of alternative schemes, followed by an evaluation and selection of the optimum scheme, which Fig.5.1 shows. The most economic final pit design depends on factors that cannot be controlled by the mining engineer such as the geometric outline of the ore body, the distribution of ore within the ore body, topography, maximum allowable slope angles, etc. However, the economics of the mining program is related to the factors determined by the mining engineer such as the choice of mining ratio, production rates, and equipment (Fourie, 1992).
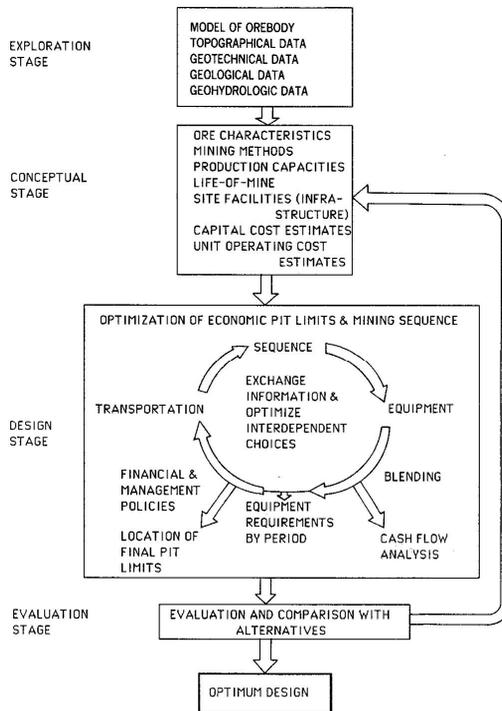
EXPLORATION STAGE

MODEL OF OREBODY
TOPOGRAPHICAL DATA
GEOTECHNICAL DATA
GEOLOGICAL DATA
GEOHYDROLOGIC DATA

CONCEPTUAL STAGE

ORE CHARACTERISTICS
MINING METHODS
PRODUCTION CAPACITIES
LIFE-OF-MINE
SITE FACILITIES (INFRA-
          STRUCTURE)
CAPITAL COST ESTIMATES
UNIT OPERATING COST
          ESTIMATES

DESIGN STAGE

OPTIMIZATION OF ECONOMIC PIT LIMITS & MINING SEQUENCE

SEQUENCE

EXCHANGE
INFORMATION &
OPTIMIZE
INTERDEPENDENT
CHOICES

TRANSPORTATION          EQUIPMENT

FINANCIAL &
MANAGEMENT
POLICIES

BLENDING

LOCATION OF
FINAL PIT
LIMITS

EQUIPMENT
REQUIREMENTS
BY PERIOD

CASH FLOW
ANALYSIS

EVALUATION STAGE

EVALUATION AND COMPARISON WITH
ALTERNATIVES

OPTIMUM DESIGN

Fig.5.1. Diagrammatic presentation of the design process (after Fourie, 1992)

## 5.1.1 Open Pit Terminology

There are some terms and phrases used for certain operations or for defining geometric parameters in open pit mining. The following are terms that are used in open pit mine planning. These terms are illustrated in Fig.5.2, which shows a pit section through an idealized tabular ore body (Fourie, 1992).

A bench may be defined as a ledge that forms a single level of operation above which mineral or waste materials are mined back to a bench face. The mineral or waste is removed in successive layers, each of which is a bench. Several benches may be in operation simultaneously in different parts of, and at different elevations in the open pit mine.

The bench height is the vertical distance between the highest point of the bench, or the bench crest, and the toe of the bench. The bench height is normally governed by the specifications of operating machines, such as drills and shovels, and by government mining regulations. The bench slope is the angle, measured in degrees, between the horizontal and an imaginary line joining the bench toe and crest.

Pit limits are the vertical and lateral extent to which the open pit mining may be economically conducted. The cost of removing overburden or waste material vs. minable value of the ore is usually the prime factor controlling the limits of the pit. Other factors that can influence pit limits are surface infrastructures, such as rivers, townships, etc.
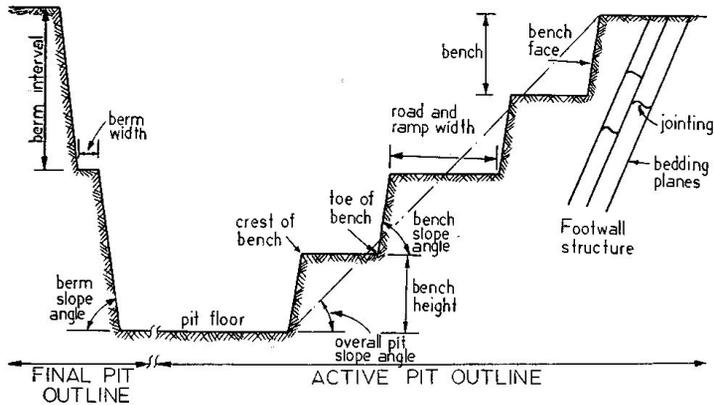
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Fig.5.2. Section through an open pit in an idealized tabular ore body (after Fourie, 1992)

To enhance the stability of a slope within the pit and for safety reasons, berms may be left. A berm is a horizontal shelf or ledge within the ultimate pit wall slope. The berm interval, berm slope angle, and berm width are governed by the geotechnical configuration of the slope. The overall pit slope angle is the angle at which the wall of an open pit stands, as measured between the horizontal and an imaginary line joining the top bench crest with the bottom bench toe.

During the life of open pit mining, a haul road must be maintained into the pit. A spiral system is an arrangement in which the haul road is arranged spirally along the perimeter walls of the pit so that the gradient of the road is more or less uniform from the top to the bottom of the pit. A zigzag or switchback system is an arrangement in which the road surmounts the steep grade of a pit wall by zigzagging, generally on the footwall side of the pit. The choice of spiral or zigzag is dependent on the shape and size of the ore body, truck economics, and pit slope stability. Haul road width is governed by the required capacity of the road and type of haulage unit. The grade may be defined as the inclination of the road in terms of degrees from the horizontal or percentage of rise to the horizontal.

The angle of repose or angle of rest is the maximum slope at which a heap of loose material will stand without sliding.

The suboutcrop depth represents the depth of waste that has to be removed before any ore is exposed. This waste is often referred to as pre-production stripping.

### 5.1.2 Exploration Input for Open Pit Planning

The process that lasts throughout the life of a mine is estimation of the characteristics of a reserve including quantity, grade, and thickness. This process is connected with the preplanning stage. The data necessary to define the extent and value of a deposit are obtained from exploration phase. Moreover, exploration provides data essential to mine development. A pit size and layout, production rate, and the mineral processing flowsheet are based on the exploration data input. An exploration program involves drilling and sampling to obtain information used to establish mine operation parameters, geotechnical design, geohydrologic conditions, and mineral processing or metallurgical extraction (Fourie, 1992).

Geotechnical Investigations. Soil masses are often homogenous and relatively isotropic, whereas rock masses are not. The main reason for this is the existence of ubiquitous discontinuities in rock masses compared with soils. The behaviour and stabilities of rock

masses are governed mainly by the nature and orientation of these discontinuities, while the stability of soil masses is controlled mainly by the strength of the intact material.

Requirements for effective geotechnical drilling are as follows (Fourie, 1992):

1. Boreholes must be cored from as close as possible to surface through the overburden to at least 10 m into the footwall. If permeable zones at greater depth are possible to exist, the boreholes should be extended.

2. Cores should preferably be at least 60 mm in diameter and drilled with adequate core barrels to ensure minimal core disturbances.

3. It may be necessary to drill inclined rather than vertical holes to: a) orientate the core, either from the bedding traces or by the use of a core orientation, b) intersect steeply dipping points, which cannot be properly sampled by vertical holes.

The density of geotechnical drilling depends on the degree of variability of the material over the mining area to analyse. An exploration geologist should supervise drilling to record hole depths, core losses, and other significant aspects (e.g., water losses). The exploration geologist and the mine-planning engineer have to ensure that all mining-related data are collected.

Each recovered core run should be transferred to the core tray and allowed to dry before being: photographed in colour, ensuring that the core trays are adequately labelled and oriented; logged in detail for: rock type, degree of weathering, description of weak rocks and weak zones, and location, description and orientation of any breaks in the core; sampled for materials testing. Geologic data should be presented in graphical form to indicate the similarity of variations in properties across the proposed mining area. Rock hardness and bedding, or joint frequency, can be illustrated along the borehole. Material testing is necessary to determine significant material properties and to confirm visual estimates during core logging. The relevant testing for open pit mines should embrace (Fourie, 1992):

1. Slaking and classification tests on weathered or suspected weak materials that could influence dump and pit stability.

2. Shear strength tests on soft or weak layers, especially in the floor, and on overburden materials manifesting strong slaking behaviour, for application in analysis of pit stability.

3. Compressive strength, point load strength, and drillability tests to provide data to evaluate overburden and ore excavation characteristics.

Geohydrologic Investigations. The presence of ground water in the rock mass surrounding an open pit influences mining operations for the following reasons (Fourie, 1992):

1. Water pressure decreases the stability of pit slopes by reducing the shear strength of potential failure surfaces. Water pressure in tension cracks, or near-vertical fissures increases the forces inducing sliding, which reduces stability.

2. High moisture content increases unit weight of the rock and in consequence transport costs. Moisture content changes in some rocks, primarily shales, can cause fast weathering and hence decrease in stability.

3. Freezing of groundwater during winter can cause wedging in water-filled fissures because of temperature-dependent volume changes in the ice. Freezing of surface water on slopes can block drainage paths, causing a build-up of water pressure in the slope resulting in decrease in stability.

4. The velocity of flow of groundwater can cause erosion of surface soils and fissure infilling. This erosion can reduce stability and also silt up drainage systems.

5. Discharge of groundwater into an open pit increases operating costs since the water must be pumped out, and also because of the difficulties of operating heavy equipment on very wet ground. Wet boreholes increase blasting problems and blasting costs.

6. When water pressure within the material rises to make uplift forces exceed the mass of the soil, liquefaction of overburden soils or waste dumps can happen. This can be observed if

drainage channels are blocked or if the soil structure undergoes a sudden volume change, for example under earthquake conditions.

The most important effect of the presence of groundwater is the reduction in stability resulting from water pressures within discontinuities in the rock mass.

There are two possible ways to collect data on water pressure distribution within a rock mass: a) deduction of the overall groundwater flow pattern using the permeability of the rock mass and sources of groundwater, b) direct measurement of water levels in boreholes or wells or of water pressure with the use of piezometers installed in boreholes (Fourie, 1992).

5.1.3 Bench Plan Preparation and Presentation

The design of a modern mining operation needs the construction of an ore body model or block model. This model is constructed from predicted information using exploration data from boreholes. The blocks involved are subsets of the overall model that makes it possible to manipulate the contained information on a local scale. Block models enable mining planners to select the best physical and economical means of extracting the ore. The uses of a block model can be quite diversified, but it is obvious that a single model that satisfies all curiosities and forms of expertise is difficult to build. Because material removal or mining takes place along a particular bench, it is firstly important to prepare a series of bench plans that, when combined, makes the overall block plan of the modelled ore body. This is illustrated in Fig.5.3. Modern computer facilities enable one to prepare complex bench plans (Fourie, 1992).
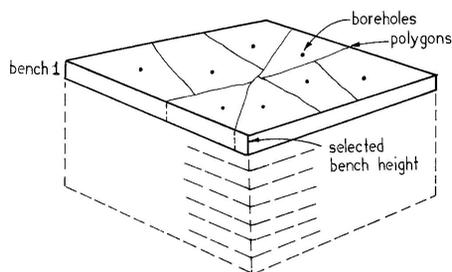


Fig.5.3. Block model of an ore body (after Fourie, 1992)

5.1.4 Stripping Ratio

The parameter known as the stripping ratio represents the amount of uneconomic material that must be removed to uncover one unit of ore. Fig.5.4 shows an idealized open pit ore body, dipping at angle $\alpha$. The ratio of the total volume of waste to the ore volume is defined as the overall stripping ratio (Fourie, 1992):

R=volume of waste removed to depth d/volume of ore recovered to depth d      (4.1.1)
=ABD/BCED

It is a volume relationship, calculated in cubic meters/cubic meter, but more often expressed as tonnes/tonne. However, in mining certain minerals, stripping ratio is expressed in units of cubic meters/tonne.
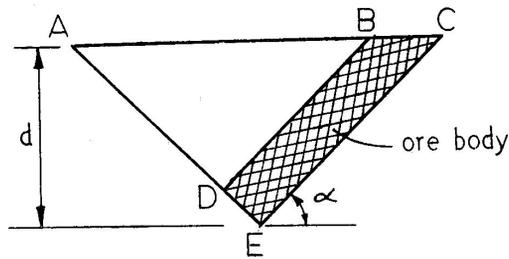
Fig.5.4. Overall stripping ratio representation (after Fourie, 1992)

Having determined the final pit limits and overall stripping ratio, the mining plan can be performed in a number of ways such as Declining Stripping Ratio Method, Increasing Stripping Ratio Method, and Constant Stripping Ratio Method. Declining Stripping Ratio Method requires that each bench of ore be mined in sequence, and all the waste on the particular bench is removed to the pit limit. In Increasing Stripping Ratio Method, stripping is performed as needed to uncover the ore. The working slopes of the waste faces are kept parallel to the overall pit slope angle. Constant Stripping Ratio Method tries to remove the waste at a rate given by the overall stripping ratio. The working slope of the waste faces starts very shallow, but increases as mining depth increases until the working slope equals the overall pit slope. This method, from an advantage and disadvantage point of view, is a compromise that avoids the difficult conditions of the former two stripping methods (Fourie, 1992).

5.1.5 Mine Plan Frequency

Mine plan frequency can be short- to long-range. The most popular are daily, weekly, monthly, yearly, and life-of-mine. The main difference between these plans is the degree of detail. The shorter the time period covered by the plan, the greater the degree of accuracy and confidence that the actual mining operation conditions will meet the forecast. For example, the daily mine plan details the production schedule for 24 hours, while long-range plans can be used to establish financial forecasts concerning purchase of new equipment and variations in operating costs (Fourie, 1992).

5.2 Strip Mine Planning and Design

5.2.1 Planning Procedure and Reserve Estimation

5.2.1.1 Planning Procedure

Many variables closely interrelated are needed to plan a surface mine using the strip (open cast) mining method. Hrebar (1992) distinguishes four possible phases of planning procedure, each of which utilizes different sets of the variables, as follows:
I. Information on deposit
   A. Geology of overburden: (1) stratigraphy, (2) geologic structure, (3) physical properties (highwall and spoil properties, degree of consolidation), (4) thickness and variability, (5) overall depth, (6) topsoil parameters
   B. Geology of coal: (1) quality (rank and analysis), (2) thickness and variability, (3) chemical and physical characteristics, (4) structure (particularly at contacts)

C. Hydrology (overburden and coal): (1) permeability, (2) porosity, (3) transmissivity, (4) extent of aquifers

D. Geometry: (1) size, (2) shape, (3) attitude, (4) continuity

E. Geography: (1) location, (2) topography, (3) altitude, (4) climate, (5) surface conditions (vegetation, stream diversion), (6) drainage patterns

F. Exploration: (1) historical (area, property), (2) current program, (3) sampling (types, procedures)

II. Project information

A. Market: (1) customers, (2) product specifications (tonnage, quality), (3) locations, (4) contract agreements, (5) spot sale considerations, (6) preparation requirements

B. Transportation: (1) property access, (2) coal transportation (methods, distance, cost)

C. Utilities: (1) availability, (2) location, (3) right-of-way, (4) costs

D. Land and mineral rights: (1) ownership (surface, mineral acquisition), (2) acreage requirements (onsite, offsite), (3) location of oil and gas wells, cemeteries, etc.

E. Water: (1) potable and preparation, (2) sources, (3) quantity and quality, (4) costs

F. Labour

G. Governmental considerations: (1) taxation (local, state), (2) royalties, (3) reclamation and operating requirements, (4) zoning, (5) mining legislation

III. Development and extraction

A. Compilation of geologic and geographic data: (1) surface and coal contours, (2) isopach development (thickness of coal and overburden, stripping ratio, quality, costs)

B. Mine size determination: (1) market constraints, (2) optimum economics

C. Reserves: (1) methods of determination, (2) economic stripping ratio, (3) burned, oxidized areas

D. Mining method selection: (1) topography, (2) refer to previous geologic/geographic factors, (3) production requirements, (4) environmental considerations

E. Pit layout: (1) extent of available area, (2) pit dimensions and geometry, (3) pit orientation, (4) haulage, power, and drainage systems

F. Equipment selection: (1) sizing, production estimates, (2) capital and operating cost estimates, (3) repeat for each unit operation

G. Project cost estimation (capital and operating): (1) mine, (2) mine support equipment, (3) office, shop, and other facilities, (4) auxiliary facilities, (5) manpower requirements

H. Development schedule: (1) additional exploration, (2) engineering and feasibility study, (3) permitting, (4) environmental approval, (5) equipment purchase and delivery, (6) site preparation and construction, (7) start-up, (8) production

IV. Economic analysis

A. Sections III and IV are repeated for various alternatives

Having gathered the information on deposit and project, mine planners start preparing the development and extraction plan. Project economics is evaluated, and an economic analysis is conducted to estimate project viability. Steps II and IV are iterated until an optimum plan is found. These iterations take into account various mining method/equipment combinations, mine size/equipment combination, mining method/pit layout combinations, etc. The variables and information that should be considered indicate that the problem of surface mining is strictly connected with geology, engineering, environmental sciences, and economics. Therefore, the planning and development process has interdisciplinary character and needs a lot of individuals with diverse training and qualifications to take part in it. If the mine plan is correct, an operating plan will be feasible and will optimise the economic return that depends

on legal, environmental, and contractual factors connected with the given mining area (Hrebar, 1992).

Jones (1977) outlined the ten major steps involved in planning and developing a surface coal mine. In his opinion, this process can last even 10 years and requires millions of dollars of expenditure apart from that for actual mine preparation and equipment purchase. These steps are the following:

I. Assembly of the minable coal package: (1) lease acquisition, (2) mapping the area, (3) drilling program, (4) surface drilling rights acquisition, (5) drilling, sampling, logging, analysis, (6) mineral evaluation (determination on commercial quantities present), (7) drilling on closer centres (development drilling), (8) sampling, logging analysis, (9) surface acquisition

II. Market development: (1) market survey, (2) potential customer identification, (4) contract negotiations III. Environmental and related studies: (1) initial reconnaissance, (2) scope of work development, (3) consultant selection, (4) implementation, (5) environmental impact report, (6) environmental monitoring

IV. Preliminary design, machine ordering: (1) conceptual mining development, (2) economic size determination, (3) mining system design, layout and development, (4) equipment selection, (5) stripping machine ordering, (6) mine plan development

V. National environmental policy: (1) environmental impact statement (EIS), (2) EIS review and comments, (3) council on environmental quality filling, (4) mining and/or reclamation plan approval VI. Permits

VII. Design and construction: (1) preliminary design and estimation, (2) material ordering and contracting, (3) water well development, (4) access road and site preparation, (5) railroad construction, (6) power supply installation, (7) facilities and coal handling construction, (8) warehouse building and yards, (9) coal preparation and loading facilities construction, (10) overland conveyor construction

VIII. Mining preparation: (1) stripping machines erection, (2) loader erection, (3) support equipment readying, (4) manpower recruitment and training IX. Production build-up X. Full production

## 5.2.1.2 Reserve Estimation and Stripping Ratio Calculations

The reserve estimation and construction of the geologic model of the deposit are the prime tasks of the planning process. These estimates can be preliminary or made in detail, which depends on the stage in the exploration/development program. There can be two estimation methods i.e., sampling methods and calculation methods (Hrebar, 1992).

Sampling Methods. A combination of rotary and diamond core drilling methods is used to obtain sample data from the deposit. Rotary drilling is used to determine overburden depth and coal thickness. A rotary rig is used to drill through the coal seam(s). Then geophysical methods are applied in the hole and a log of the hole is made. Gamma logs and density logs are used to pick the coal/overburden interface. Quality data (e.g., %ash, %S, etc.) are gathered by diamond core drilling. The solid cores of the total coal seam are processed in a lab. Sample spacing will vary depending on the variability of thickness and quality parameters of the seam. The initial rotary grid spacing (e.g., 600 m) may precede a closer-spaced grid (e.g., 300 m) or a hole at the centre of the first grid. The coal core drilling usually is performed on a lesser spacing than rotary drilling. Closely spaced fences of holes are drilled perpendicular to the cropline to evaluate the extent of the burned or oxidized zone. This contact usually correlates with some depth of overburden, but in areas where overburden is badly fractured, the burned or oxidized zone may not correspond to depth of overburden, and more extensive

drilling is needed. Hole density for overburden coring is about four to five holes/section (Hrebar, 1992).

Calculation Methods. The method of calculation used should be a function of the seam variability and to some extent the mining history of the seam in the particular area. There are three such the methods i.e., Tons/Acre Method, Iso-Line Method, and Computer Models (Hrebar, 1992).

Tons/Acre Method. The tons-per-acre method is used where there is little seam variability or where a quick estimation of reserves, overburden, and stripping ratio is needed. The method involves calculating the tonnes of coal/hectare-meter and then multiplying this figure by the average thickness of the coal to determine the tonnes/hectare of coal. The area of reserves in hectares-metres is then multiplied by the tones/hectare of coal to obtain the reserve. Two reserve figures are presented: (1) the in-place reserve and (2) the recoverable reserve, which includes allowances for mining and for preparation plant recoveries. Similarly, overburden/hectare is calculated using the average overburden height, and the reserve area is multiplied by this figure to get total overburden. The average stripping ratio (i.e., bank cubic meters of overburden/tonne of coal) is calculated by dividing total volume of overburden by total weight of coal, on a recoverable basis. In these calculations, average coal and overburden thickness dimensions are obtained from the drillhole data. The area is determined by the boundaries of the outcrop, subcrop, and/or property lines. Techniques such as the polygon or triangular approach can be applied to more precisely assign areas of influence to the drillholes. A detailed reserve and overburden analysis can be carried out by means of the iso-line method (Hrebar, 1992).

Iso-Line Method. This method relies on construction of lines of equal values for overburden thickness and coal thickness to present a model of these parameters representing the deposit. The iso-overburden map is prepared with the surface topography map and a topographic map of the top of the coal seam. The latter is constructed by using drillhole data (depth to top of coal) and the surface drillhole collar elevations. The two maps are then overlaid, and by subtraction of the surface and top of coal elevations, the thickness of overburden can be established either at the topography intersections or on a regular grid basis. These values are then contoured to generate the iso-overburden thickness lines. The iso-coal thickness map is produced either by using the top and bottom coal contours or by interpolation of actual drillhole thickness intercepts. When coal structure contours are used, the top and bottom coal elevation maps are overlaid and the values calculated at either the line intersections or at regular grids. These values are contoured to produce the iso-coal thickness lines. If coal intercepts are used directly, the values are interpolated to construct the thickness iso-lines. The coal structure topography is used very often, because these maps are used in pit haulage and drainage planning. The coal thickness and overburden thickness iso-lines are then applied to calculate the amount of overburden and coal. The data are summarized by overburden range. Iso-stripping lines can also be constructed by overlaying the overburden and coal thickness map, calculating the stripping ratio at the line intersection or on a regular grid, and then interpolating and contouring the stripping ratios. The iso-stripping map is useful to evaluate the economic limits (Hrebar, 1992).

Computer Models. Many operators use computer software to perform the previously depicted reserve calculations and to generate the graphics. A typical software system will have the following capabilities: (1) drillhole data loading and editing with data including drillhole identification, location, lithology on a from-to basis, rock type, seam codes, coal seam analysis, etc; (2) drillhole data listing and abstracts; (3) drillhole data quality summaries by seam, list, and area with averages for specified parameters; (4) polygonal reserves with reserves by hole and seam with maps generated showing polygonal shapes; (5) topography loading by drillhole or by digitising contour maps; (6) vertical cross sections, to scale,

showing overburden, coal, and other input data; (7) plan maps showing drillhole locations, crop lines, property lines, etc.; (8) interpolation of data to produce a gridded model of overburden thickness, coal thickness, top and bottom coal elevations, etc.; (9) seam reserves based on gridded data for any area or series of areas with ability to limit reserves based on numerous parameters or combinations of parameters (e.g., minimum coal thickness, maximum overburden thickness, maximum percent sulphur, etc.; (10) mining cut data generated on a cut-by-cut basis to allow simulation of alternative mine plans and comparison of coal and overburden amounts, coal quality, etc. Such software is available from numerous companies (Hrebar, 1992).

Limitations. Various limitations and losses that will be met should be taken into account. There can be economic limits, equipment limits, and mining and barrier losses. These factors will result in coal left in place or lost in the process of excavation. The minable reserve is usually less than the in-place or geologic reserve (Hrebar, 1992).

Economic Stripping Ratio. The economics of stripping increasing amounts of overburden is a factor that can be controlled by mine management. The basic premise involves setting a minimum acceptable profit/tonne of coal recovered and then adjusting the amount of stripping to satisfy that profit constraint. Such an approach results in a maximum allowable stripping ratio and establishes a boundary on the area that can be mined economically (Hrebar, 1992).

Equipment Limitations. Stripping equipment digging-depth limitations or coal-loading equipment height limitations may limit the minable area. In certain deposits, highwall or spoil instability may make use of deep-digging techniques impossible and limit the minable area with a maximum digging depth. In this situation, extended bench or spoil-side pullback may not be possible, causing a depth limit on the reserve. A minimal coal thickness is required to economically recover thin seams using surface techniques. In very thin seams, dilution from top and bottom material, will unable recovery of a profitable product. In consequence, depending on the coal loading technique used, areas with coal thickness below that minimum can become unminable (Hrebar, 1992).

Mining and Barrier Losses. Mining losses are related to the extraction process. These losses include the following: (1) top of coal: losses at upper coal contact with overburden as a result of cleaning coal after stripping; (2) bottom of coal: losses at lower coal contact with bottom as a result of loader losses; both top and bottom losses, as a percentage, are a function of seam thickness; (3) rib: losses at side of seam adjacent to spoil as a result of spoil piled on coal rib during stripping or slides of spoil on coal; function of pit width; (4) other: includes fly rock (blasting) and transportation losses (dust and spillage). Barrier losses are caused by coal left in place. They include outcrop (coal left as a low-wall barrier or for quality reason), right-of-way, stream, underground mine, oil and gas wells, property line, building, and cemetery. These losses depend on the geologic and geographic setting of a mine. Mining and barrier losses are significant and therefore, vital in estimating minable reserves (Hrebar, 1992).

## 5.2.2 Stripping and Pit Development

Since the overburden removal system takes the greatest capital and operating costs, the surface coal mining method selected for a given mining area is strongly related to this unit operation. When the stripping method and complementary equipment have been chosen, compatible equipment systems are selected for the other unit operations such as coal fragmentation, coal loading and hauling, etc. Development for the major systems used in surface mining of coal, especially at large operations like in western part of the USA, is depicted below. Various alternatives have been elaborated for the stripping systems, as well as for the other unit operations. It is important to ensure maximum resource extraction at a minimum both capital and operating costs (Hrebar, 1992).

5.2.2.1 Alternative Stripping Methods

Three alternative stripping procedures are described: Area/Dragline Method, Modified Open Pit/Shovel-Truck Method, and Block Area/Dozer-Scraper Method (Hrebar, 1992).

Area/Dragline Method. This method presented in Fig.5.5 involves opening an initial box cut, removing the coal exposed in the box cut, and then placing the overburden from the next longitudinal cut into the mined-out, box-cut area. The procedure is then repeated on a cut-by-cut basis. The method is also referred to as 'deep ploughing'. The method can be employed in flat to moderately dipping coal seams with relatively constant overburden depths. It can also be used in areas where coal dip or overburden slope will allow reaching the economic limit in few cuts and such the method is called a modified area or box-cut contour operation. The difference is the rate at which the limit is reached. The advantages and disadvantages of the area/dragline method combine the properties of the dragline excavator and the method. The alternative piece of equipment used to conduct stripping is the stripping shovel. Draglines are chosen because they are flexible in varying operating conditions. The dragline can handle varying overburden depths and characteristics, and multiple seams by changing the operating mode. These changes allow mining under the complex set of conditions without auxiliary equipment that would be needed with a stripping shovel. However, they may cause some loss in machine productivity (Hrebar, 1992).

Modified Open Pit/Shovel-Truck Method. Modified open pit or terrace mining is used in thick-seam deposits with low stripping ratios. Seams are flat-lying, moderately dipping, or rolling. The method shown in Fig.5.6 involves opening an initial pit and placing the overburden in temporary off-site storage. Coal is then removed from the initial pit area. The next cut is made in the direction of mine advance and the overburden is hauled around the pit and dumped in the mined-out area. Coal is removed and the haul-back process is repeated as the relatively small pit advances. In steeply dipping seam operations, the initial pit is opened and coal removed. The initial box cut is opened to the economic limit down dip. Next, subsequent cuts are advanced on-strike, and overburden hauled back into the mined-out initial pit (Hrebar, 1992).

Block Area/Dozer-Scraper Method. The block area method was first an alternative to the area/dragline method because draglines were not readily available. Alternative methods that would use available construction-type equipment such as dozers and scrapers were looked for. This method takes advantage of the dozer's ability to move material over short distances at low costs and the scraper's ability to elevate material over steep grades for short distances at reasonable costs. However, as the degree of consolidation of material increases, wear and tear on the scrapers makes the method very expensive because of high repair, maintenance and supply costs, and reduced machine productivity. One version of this method is presented in Fig.5.7 (Hrebar, 1992).

BOX CUT METHODS

END CUT:



END CUT
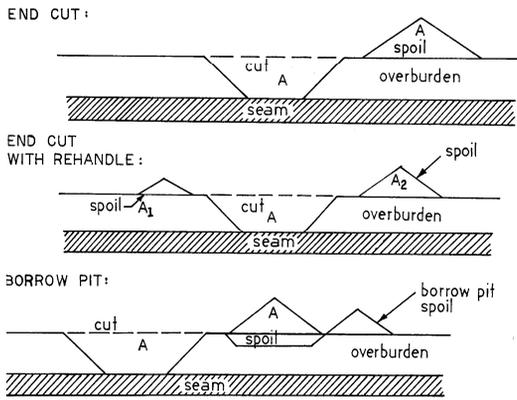WITH REHANDLE:



BORROW PIT:
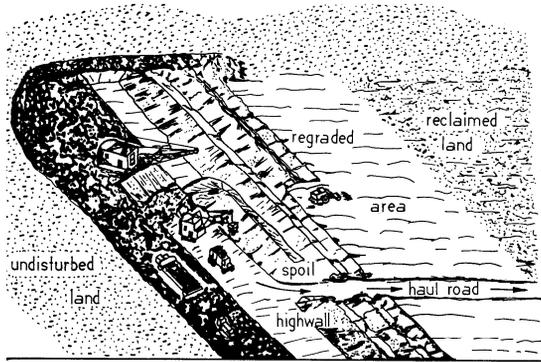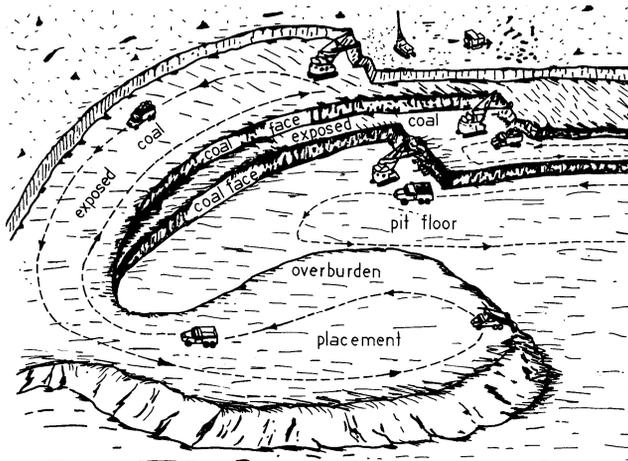


Fig.5.5. Area/dragline stripping (after Anon, 1975)



Fig.5.6. Modified open pit: flat-lying seams (after Anon, 1975)

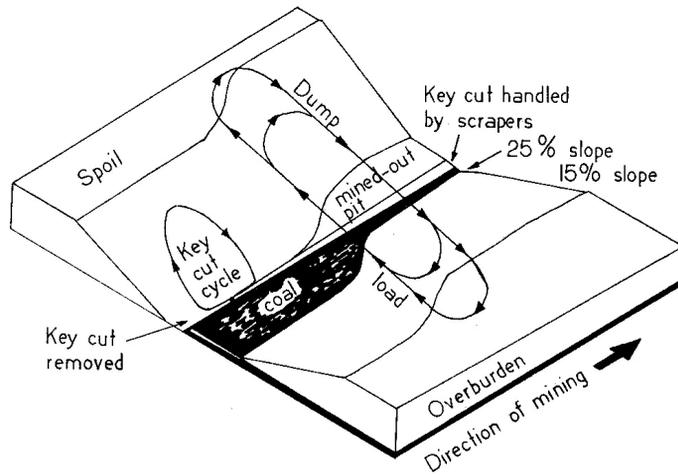Fig.5.7. Block area/dozer-scraper method  (after Ryan, 1982)

### 5.2.2.2 Dragline Pit Design

Pit parameters and configuration chosen for a given mining area are influenced by deposit geometry and regulations. The following factors have to be focused on: pit length, pit width, pit orientation, and haul road layout (Hrebar, 1992).

Pit Length. Pit length depends on the following parameters: (1) property dimensions (can limit maximum length); (2) dragline efficiency (longer pits, over 900 m, are preferred); (3) coal inventory (A pit inventory may be kept to meet seasonal demand; amount depends on seam thickness, pit width, and length of block in place.); (4) spoil instability (Shorter pits are preferred to limit spoil failures if spoil stability is a function of time); (5) pit flooding (In high-rainfall areas, short pits limit water quantities in pit.); (6) power system (In low mine-voltage systems, voltage drop is more frequent with long pits) (Hrebar, 1992).

Pit Width and Orientation. The following factors influence the pit width: (1) stripping machine reach (Strippers are limited by reach, which in consequence limits the pit width and overburden depth pattern if rehandling is not to be involved.); (2) overburden depth and dragline productivity (Narrow pits should be selected in shallow overburden and wide pits in deep one to improve dragline production.); (3) reclamation (Narrow pits minimize spoil peak strike-off, while wide pits increase spoil area and strike-off quantities.); (4) coal loading and hauling (Some minimum width is required for efficient equipment operation.); (5) coal inventory (Width influences tonnage/unit length of pit). The choice of pit orientation and sequencing is crucial to the mine plan, because determines equipment selection and operating costs (Hrebar, 1992).

Straight vs. Curved Pits. When there is a curved outcrop or subcrop where stripping will start, it must be decided if the subcrop will be followed or whether straight pits will be used. Where curved pits are selected, a series of inside and outside curves exist. The inside curves, where the advancing highwall arc is greater than the spoil side arc, can cause significant spoil room problems, especially when overburden depth increases with the mine advance. In this situation, each new cut contains more overburden volume than is available in the prior cut spoil volume. Excess spoil is soon generated and costly rehandling is required. Outside curves

with the advancing highwall arc less than the spoil side arc can be used in places of increasing overburden depths. Then each new cut is of less area than the adjacent spoil one, and this difference can be used to accommodate the added volume caused by the increase in overburden height. Outside curves can be used to advantage where an operation meets ridges perpendicular to the cropline or subcrop. Moreover, curved pits are difficult to lay out in the field and difficult to conduct from an operating side. They also require coal haulage on curves in the confined pit. Therefore, straight pits are formed to lessen these problems. Straight pits are developed either by a series of short pits or by varying pit width. A series of short pits on the cords of an outside curve can be used to straighten the pits. Auxiliary equipment such as scrapers is used to strip these pits so that the dragline is applied when long straight pits are completed. The other method is to use the dragline and employ wide pits at the outside curves and narrow pits at the inside curves until a straight pit is constructed. This is more difficult to engineer, lay out, and execute (Hrebar, 1992).

Average vs. Minimizing Stripping Requirements. Orientation of pits and pit sequencing are essential to the mine planning. The alternatives are averaging and minimizing the stripping ratio over time. When the ratio is averaged, the same stripping ratio and the same quantity of overburden are handled each year. Overburden depths range from low to high in a given year to balance and average the ratio. The advantages of this method are constant stripping capacity requirements, constant costs/tonne, and constant manpower needs. When the ratio is minimized, areas with the low stripping ratio are mined in early years, mid-range ratio in midyears, and high ratio in later years. These plans must also take the account of minimum pit length, elimination of unnecessary dragline walking by selecting proper sequences of operations, and allowance for efficient haul-road access. The advantage of this method is the best economics achieved due to the postponement of capital expenditure and low direct operating costs in early years of production. However, the necessity to strip increasing amounts of overburden can become a problem. Management of the constantly changing operation is more difficult, but the economic benefits of minimizing stripping ratio are remarkable (Hrebar, 1992).

Haul Road Layout. Strip mine roads may be spaced on 450 to 540 m centres through the spoil to facilitate one-way hauls if possible. In deeper overburden, fewer haul roads through the spoil may be used as a result of significant spoil volume losses. Where contemporaneous haul-road reclamation is obligatory, fewer haul roads through the spoil and steeper ramps may be constructed. Both these situations decrease truck productivity because turnaround areas in the pit and time spent hauling up ramps increase. These roads are located to minimize haul distance and expected to facilitate high-speed haulage. Two other haul-road arrangements are used in certain situations. Highwall side ramps are used where soil stability problems, spoil volume losses, or unusual deposit geometry occur. The disadvantage of the highwall side ramps is the dragline inefficiency and auxiliary equipment time connected with continuous rebuilding as the mine advances. Pit-end haul roads are made where spoil volume losses or spoil instability exist. This requires special pit geometry, cropline on three sides, or deep box cutting to keep the deposit accessible. Long in-pit haul distances are associated with pit-end haul roads (Hrebar, 1992).

5.2.2.3 Pit Development

Clearing and Grubbing. This stage involves clearing the property of trees and shrubs and then removing the stumps and roots to uncover homogeneous topsoil. Grubbing is often done with rake-like grubbing devices on agricultural tractors or dozers (Hrebar, 1992).

Topsoil Removal. The topsoil has to be removed and finally replaced upon graded spoils. Topsoil can either be stockpiled at the side of the pit area for later redistribution or hauled

immediately to the graded area for redistribution. In the latter situation, the topsoil can either be hauled around the pit or across the pit on spoil bridges, which depends on economic factors like topsoil quantities and haul distances. Self-elevating scrapers are used to remove and redistribute topsoil. However, loader/truck or wheel excavator/belt systems are applied where topsoil quantities are large and/or haul distances are long. Topsoil thickness is determined by auger drilling. If multiple soil horizons exist, these layers must be removed separately, and stockpiled and redistributed to keep the integrity of each horizon. Topsoil storage should be revegetated to prevent water and wind erosion (Hrebar, 1992).

Mine Drainage and Erosion and Sediment Control. To eliminate in-pit water problems, it is good to divert surface water from operating pit areas. Diversion ditch systems can be used to deflect the water and channel it into natural drainages. For example, in the USA under regulations, this practice has included the following: (1) surface drainage from disturbed areas should pass through a sediment pond; (2) effluent from the ponds must meet limitations of PH, iron, manganese, total suspended solids, etc.; (3) sedimentation ponds must be constructed to standards on capacity, detention time, dewatering, location, slopes, etc.; (4) discharge compliance must meet 10-year, 24-hour precipitation events; (5) treatment is required to meet effluent standards. Design measures taken to meet these standards economically should involve minimizing the disturbed area, stabilizing backfill, diverting overland flow around or through disturbed areas to lessen pond size, revegetating to reduce sediment load, and separating pit water from other water to minimize treatment (Hrebar, 1992).

Regrading. Regrading or striking-off the spoil is conducted with large-horsepower dozers, and final grading is done with large graders. It embraces: restoration to nearly original contour; elimination of highwalls; restoration of natural drainages as far as possible; construction of final slopes not exceeding original slopes; productivity equal or greater than pre-mining productivity (Hrebar, 1992).

Topsoil Replacement or Redistribution. Scrapers, dozers, or loader/trucks are used in this operation and topsoil is replaced continuously or from stockpile. Some preparation of the graded spoil, ploughing, etc., is done to stabilize the topsoil bed. Traffic patterns are designed to prevent over-compaction of the bed (Hrebar, 1992).

Revegetation. To revegetate the area, planting must be done as soon as possible for example with conventional farm equipment. Comparison with a reference area is done to check the process efficiency. Erosion must be controlled and a period of responsibility is extended (Hrebar, 1992).

5.2.3 Cast Blasting of Deep Overburden

Strip (open cast) mining frequently requires overburden to be blasted to enable its economical excavation by a dragline. In other words, the overburden must be well fragmented and loosened, and its lateral displacement should be small. For many years mining engineers have applied cast blasting (explosives casting) of overburden utilizing chemical energy of explosives. The good heaving effect of ammonium nitrate-fuel oil (ANFO) and its low cost are the prime advantages of this blasting technique. The advantages of cast blasting are especially important in the case of deeper, stronger overburden requiring high powder factors (kilograms of explosive/cubic meters of overburden blasted) that are typical for some very strong sandstones. Under certain conditions, cast blasting in deep overburden can be more economic than conventional stripping. Early attempts relied on reducing the primary overburden cast by the dragline (Fig.5.8). As well as decreasing the dragline duty, this method allows one to cast very large rock fragments from the base of the highwall onto the floor of

the pit to construct the base of the spoil heap, which in turn facilitates spoil heap drainage and stability (Atkinson, 1992).
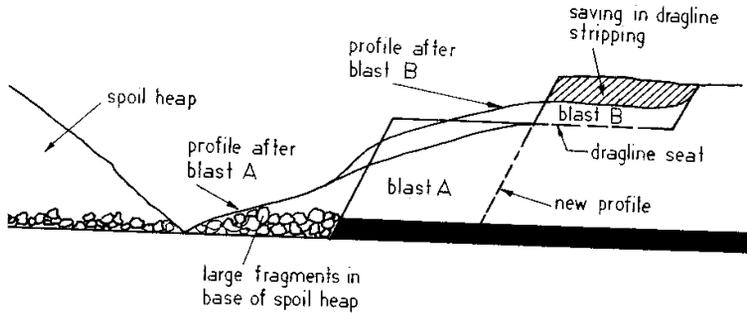


Fig.5.8. Throw blasting for a standard dragline operation (after Atkinson, 1992)

### 5.2.3.1 Presplit Blasting

Presplit blasting can be implemented to dewater permeable overburden. Where clay-rich materials do not exist, low-cost, bulk-placed ANFO has been used instead of more expensive water-resistant explosives, such as water-gels, heavy ANFO, etc. Moreover, in strong ground, a vertical face can be made after blasting the total burden whereas the irregular sloping face is formed with conventional blasting (Fig.5.9). It can be seen from Fig.5.9b that the vertical face, with significantly decreased distance from the front row of blastholes to the toe of the highwall, may result in much more efficient cast blasting. The combination of pre-splitting and inexpensive ANFO can make economic cast blasting. It can also be low-cost for non-vertical highwalls, where an inclined presplit line is used and the blasthole pattern adequately changed (Atkinson, 1992).



a) Conventional blasting profile



b) Presplitting profile
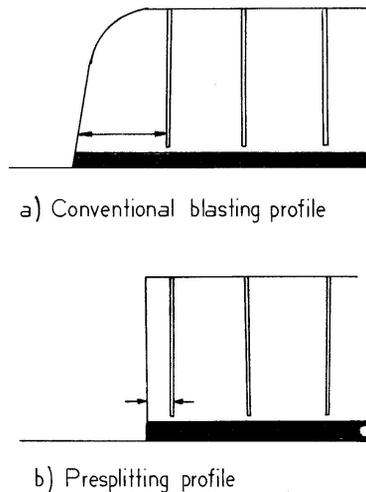
Fig.5.9. Vertical presplit highwall (after Atkinson, 1992)

5.2.3.2 Casting Characteristics

Blasting Patterns. The geometry of the spoil after cast blasting must be under control to construct a seat suitable for dragline operation while moving the most possible amount of spoil across the pit. Conventional blasting patterns in strip mining can be square or rectangular with a 90° tie up, which means that rows of holes at 90° to the highwall are detonated concurrently, then sequentially, so that the spoil can be thrown parallel to the highwall. It can be noticed that if rows of blastholes parallel to the highwall are detonated simultaneously, started from the highwall progressively from front to back, maximum spoil will be cast across the pit. It was observed that slightly more spoil is cast by this tie up, but a section of overburden between the presplit line and the last row of blastholes remains in situ (Fig.5.10) (Atkinson, 1992).
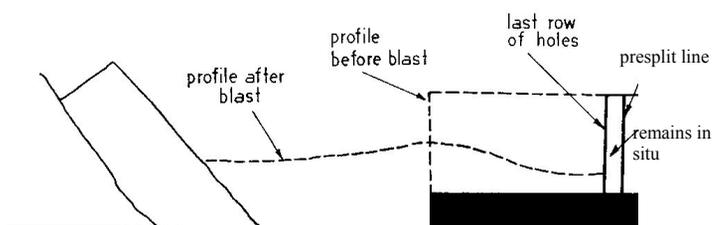


Fig.5.10. Blasting with parallel tie up (after Atkinson, 1992)

The effective blasting is one with about a 30° tie up using a staggered 'V1" pattern (Fig.5.11) with comparatively long, inter-row delays. Fig.5.11 demonstrates a typical section and the desired section. The 'throw depression' can be greater than desirable, and some little rehandling of the thrown spoil is needed to construct the 'seat' for dragline operation. The spacing: burden ratio of the blasting pattern is influenced by site-specific factors and requires detailed investigation to calculate the trajectory of the thrown spoil. Blasthole diameters of 254 to 311 mm can be used. Larger-diameter holes may be more advantageous in strong ground, if more energy can be packed into the blasthole, and closer spacing rather than larger blasthole diameters may be more effective (Atkinson, 1992).

Depth/Width Ratio. The efficiency of cast blasting can be determined with the amount of spoil thrown beyond the edge of the mineral seam. This may be calculated and expressed as the percentage volume thrown by blasting across the edge of the mineral seam that is called the percentage blast-over. When the strip width is constant, the percentage blast-over increases with increasing depth. The percentage blast-over depends on the depth/width ratio, which changes with site conditions and blasting efficiency (Atkinson, 1992).
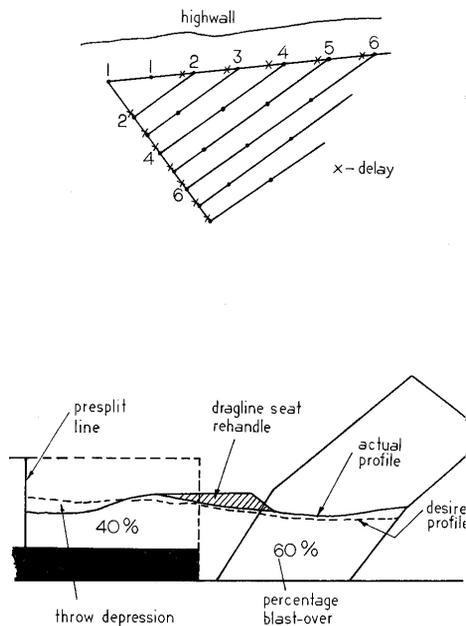
Fig.5.11. Blasting method (after Atkinson, 1992)

### 5.2.3.3 Cast Blasting Methods

The location of the dragline on the spoil side of the pit makes it possible to achieve the highest efficiency. These positive situations include: (1) The access ramp roads can be located on the highwall side of the pit, removing the problems resulting from lack of spoil space at the junctions of the strip with the access ramps (e.g., major rehandling of spoil), because spoil must be piled higher at these junctions and can cause serious spoil-side failures; (2) The stripping rhythm is kept since access ramp junctions do not have to be constructed, and dragline productivity on prime overburden is increased to a great extent; (3) Equipment scheduling is simplified; (4) Access to mineral benches in multi-seam deposits is facilitated; (5) Surface reclamation is eased and land comes back to its planned use earlier because there are not access ramp roads throughout the spoil. The main bad sides of the cast blasting method application are: (1) The dragline may operate in the less efficient 'chop-down' mode; (2) Separate electrical reticulation systems are needed on both sides of the pit-on the spoil side for the dragline and on the highwall side for the blasthole drill rigs (Atkinson, 1992).

Single-Seam, Cross-Pit, Chop-Down Operation. A dragline used in the chop-down mode is about 60% efficient compared with the conventional drag mode. Bucket maintenance costs also more (although this is significantly lessen by the better fragmentation obtained in cast blasting). The depth/width ratio of the pit should be bigger than 0.4 to achieve greater than 40% blast-over for chop-down to be taken into account; it means that the method is more suitable for deeper pits. Fig.5.12 shows the method of operation. The dragline seat bench height in (c) can be fixed so that all the spoil is required for levelling, that is, the need for levelling spoil peaks for land reclamation disappears. This can only be the case if the height of the dragline seat above the top of the mineral seam does not exceed the optimum digging

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depth. Otherwise, the bucket must be dragged to a higher elevation so that it may be hoisted away for dumping, which increases the cycle time. Under these conditions, it can be more economic to level the spoil peaks with conventional mobile equipment. (If a dragline must hoist a full bucket of spoil, the hoist and drag ropes must form an angle, 90°, i.e., be within a semicircle drawn through the fairleads and boom sheaves). The single-seam, chop-down method may be used to strip seam partings where other seams occur beneath the seam being excavated. This operation can reduce dragline productivity up to 50%, but it can be economic if the ratio of parting to lower seam thickness is low (Atkinson, 1992).
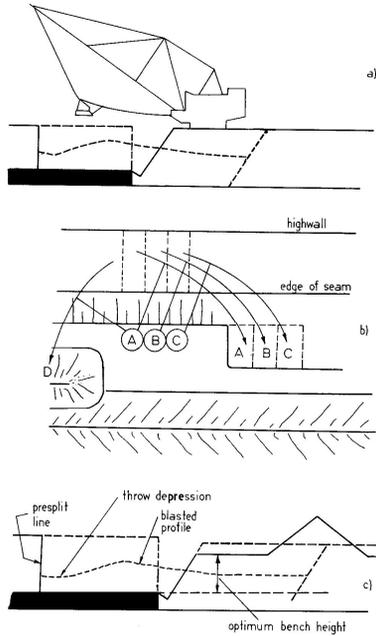


Fig.5.12. Cross-pit, chop-down with single seam (after Atkinson, 1992)

Two-Seam Method. Fig.5.13 shows a sequence of operations (a, b, c, d, e, f, g) for a typical two-seam setting. The operations can be depicted as follows (Atkinson, 1992):

(a) Pit section prior to blasting.

(b) Interburden blasted into the void left after extracting the mineral from the previous strip. Big rocks can form the base of the spoil heap and are not handled by the dragline.

(c) Blasted overburden. The throw depression of the interburden blast is filled by overburden spoil.

(d) The dragline seat is made by bulldozers.

(e) A main cut is formed to the lower seam, and the upper seam is stripped; this spoil is dumped on the lowwall to obtain material for the spoil bench.

(f) The spoil bench is dozed level, and the remaining spoil is 'pulled back', being dumped on the spoil bench.

(g) The lower seam is stripped. The mineral extraction operation can now perform. After the mineral is removed, the cycle can be repeated.
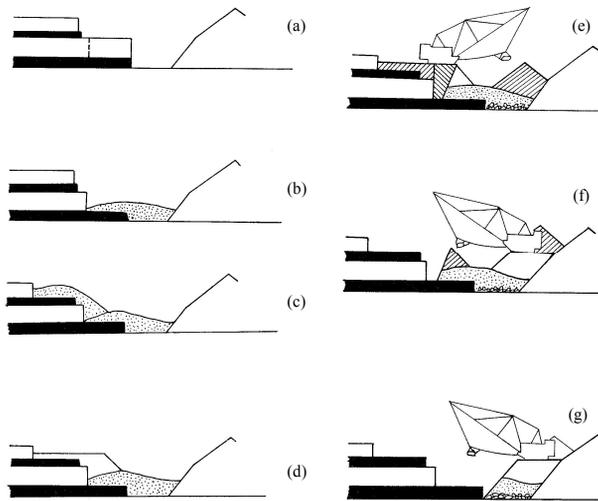
Fig.5.13.Two-seam method. Operations: (a), (b), (c), (d), (e), (f), and (g) (after Atkinson, 1992)

The method increases dragline productivity above that of conventional dragline stripping. There are good sides of this method. (1) Both seams are exposed together, which enables one to extract mineral at the same time. The upper seam can be dozed over the side of the bench on to the lower seam, which can concentrate and simplify mineral loading operations and improve blending. Greater flexibility in operation is possible since the volume of in-pit exposed deposit is increased. (2) Spoil-heap stability is improved because strong rocks from deep within the overburden make the base of the spoil heap. In conventional dragline stripping, weaker weathered material forms the toe of the spoil heap. (3) Both seams are exposed simultaneously. In conventional dragline stripping, two dragline passes are needed. Dead heading is therefore halved. (4) The deeper, stronger interburden is cast by blasting into the void to form the base of the spoil heap, and is not handled by the dragline. Consequently, bucket maintenance costs are decreased. The method can also be used for single-seam, thick-over-burden deposits to eliminate the chop-down operation of the upper overburden. Where weathered material occurs near surface, the upper bench can be presplit with more closely spaced holes than for the lower, more competent overburden. If weathering is not too deep, the upper bench slope can often approach 90° (vertical) because the presplit face was not seriously damaged after blasting and as a result of the enhanced drainage effects (Atkinson, 1992).

Stripping Previously Worked Deposits. In many countries, thick coal seams that have been previously partially extracted by underground methods (e.g., room and pillar mining) can be economically recovered by strip mining. In some situations, the security of the dragline can be in question where conventional dragline stripping is proposed. Cast blasting can be an alternative stripping method where the dragline is not located over the pillared coal, which eliminates any dragline security hazard (Atkinson, 1992).

5.3 Selection of Excavating Equipment

Ground preparation, excavation and loading, transport, and mineral treatment, are the main operations in a surface mine. They are interdependent and cannot be considered solely.

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The optimum cost per tonne may be evaluated by minimizing each of the individual operational costs. Loading, transport, and crushing costs may decrease when more explosives are used. This can reduce ground preparation costs. Adverse rock materials cause a lot of equipment selection problems in surface mining. Where continuous systems are applied, problems result from abrasion and stickiness. Therefore, testing the material properties, prior to selection as well as during excavation, is necessary. The selection of the excavator or loader is very important because it determines the other equipment needed and the mode of operation. Moreover, it is crucial to achieving low-cost production (Atkinson, 1992).

The competence and 'diggability' of ground plays an important role in the selection of excavating equipment. The choice depends on many factors, for example, (1) the intact strength of the ground, (2) the competence of the ground as a whole, (3) the abrasive properties of the minerals, (4) the bulk density of the ground, both in situ (bank) and broken (loose), (5) the flow properties of the broken ground, possible stickiness, etc., and (6) the degree of ground preparation needed to obtain a proper fragmentation. Other factors that count are: the reach of the excavator, horizontal as well as vertical, above grade and below grade, both digging and discharging; the bearing strength of the working bench; manoeuvrability; power supply; capacity; etc. Where the mineral grade varies significantly and stockpiling is uneconomic or not possible due to the nature of the mineral, to feed the mill adequately, a larger number of smaller-capacity excavators may be required than if grade is relatively uniform or if stockpiling can be performed (Atkinson, 1992).

5.3.1 Cyclic and Continuous Excavators

There are a lot of operational systems and machines available. Their selection depends on numerous factors. Fig.5.14 presents such the factors in form of a classification diagram. The diagram shows two methods of working, that is, continuous or cyclic, and also presents the types of materials handling. Continuous operation is applied more often, because it utilizes plant more efficiently and decreases margins added to plant capacity, as well as all peaks (e.g., material flow, mechanical stresses, electrical demands). In open pit mining, the continuous system is classified by the multi-bucket excavator (bucket wheel and bucket chain). The cyclic system is represented by: shovels, draglines, front-end loaders, scrapers, rippers, bulldozers, etc. (Atkinson, 1992). The machines that are most widely used in mining are depicted below.
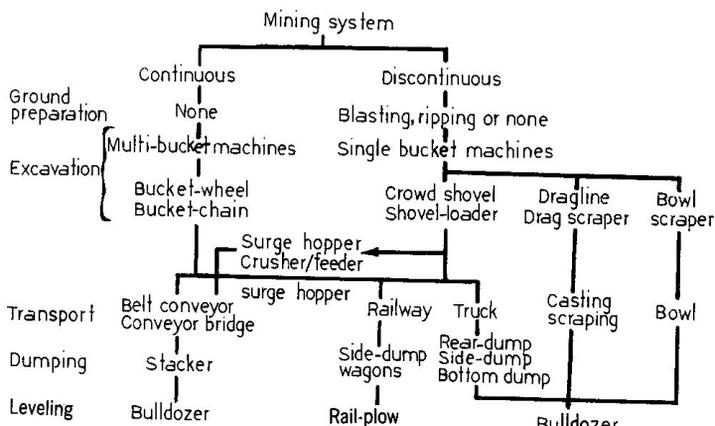


Fig.5.14. Materials handling system classification diagram (after Atkinson, 1992)

### 5.3.2 Loading Shovels

The crawler-mounted loading shovel is the best machine to handle hard, dense, abrasive, badly fragmented ground because it demonstrates positive crowd action and can use a high breakout force. It has good spotting ability when loading into dump trucks, rail cars, loading hoppers, etc. Since it has sturdy construction and its action is simple, it can be highly available; however, it is not very mobile and shows a poor sub-grade digging capability. A competent floor is vital unless oversize crawlers of low bearing pressure are implemented. According to Atkinson (1992) loading shovels can be grouped into three prime types: (1) heavy-duty, mine loading shovels (for dense, abrasive, badly fragmented ground, which occurs in most metal mining operations); (2) general purpose loading shovels (for lighter, well-fragmented materials, e.g., sand and gravel, coal, bauxite, etc.); and (3) hydraulic shovels.

### 5.3.2.1 Size Selection

The first step in shovel selection is the calculation of dipper size. Since in situ volumes (bank volumes) are of the most importance to mine planners, they employ them in the assessment. Dipper size can be expressed as (Atkinson, 1992)

$$B_c = Q/(C \times S \times A \times O \times B_f \times P) \tag{4.3.1}$$

where $B_c$ is dipper capacity (volume), Q is production required (bank volume per hour), C is theoretical cycles per hour for a 90° swing = $60/t_c$, $t_c$ is shovel cycle time for a 90° swing in minutes, A is mechanical availability during the scheduled hours of work, O is the job operational factor, $B_f$ is the dipper factor, S is the swing factor, and P is the propel time factor (Atkinson, 1992).

C: Theoretical cycles per hour. Values of C may be found in manufacturers' literature or from time studies. In most open pit mining operations, shovels are able to load up to their optimum digging depth. However, in strip (open cast) mining, if the mineral bed is comparatively thin, the crowd-hoist time in the shovel cycle will be extended to gain a full dipper. Moreover, in certain strip mining operations, the bed underlying the mineral may be not competent enough for rubber-tired trucks to pass. In this case, the haul roads are located on top of the mineral bed, and the loading shovel can be equipped with a longer-than-standard boom and dipper handle to obtain the reach suitable for loading the trucks. As a result, the loading shovel cycle times will be longer than those given in manufacturers' literature.

S: Swing factor. The loading shovel cycle times are based on a 90° swing. Due to various angles of swing occurrence the shovel cycle time has to be corrected on the base of manufacturer's data.

A: Availability. A is the availability of the shovel for work during the manned hours; it is defined as the mechanical availability during the scheduled hours.

O: Job operational factor. Loading shovels are subject to delays due to management, supervision and labour deficiencies, job conditions, climate, etc. The dipper capacity increase must compensate for these losses in time. If the propel time P is included in O, it must be eliminated from $B_c$.

OA: Operating efficiency. If no experience data can be used to calculate A, O and their product OA, the operating efficiency, may be determined using literature data. Production can be decreased by poor job conditions (poor quality of the labour). The actual time of producing will be high if management and supervision are of good quality. Poor management and

supervision will decrease production, and the dipper capacity must be increased to meet production requirements.

$B_f$: Dipper factor

$$B_f = \text{Fillability/Swell factor} \qquad (4.3.2)$$

Fillability is the loose volume of material excavated in an average load as a ratio of the dipper capacity $B_c$; this is best determined by field tests. Swell factor is defined as weight/unit volume (bank) divided by weight/ unit volume (loose). It shows the degree of swell. The special attention should be paid when calculating the value of swell factors for less-consolidated materials whose volume can decrease during repeated handling,

P: Propel time factor. P depends on the time required to propel the shovel during manoeuvring. In strip mines, where a relatively narrow bed is exposed, the shovel must move more frequently than in the case of loading a high rock pile.

The determination of space needs for a loading shovel has to be preceded by detailed investigations. Operational efficiency requires sufficient bench room. However, in most multi-bench, conical pits (Fig.5.15), the slope angle for a single bench, with double-spot loading, is too flat and several bench levels, which make a machine-loading group, must be mined sequentially. Some space is also necessary for drilling and blasting and to allow trucks to pass. For pits using rail haulage, clearance must be made for rail track, trolley lines, and shovel; the whole sequence of drilling, blasting, and loading; and trolley and track shifting. The essential dimensions of loading shovels are shown in Fig.5.16 (Atkinson, 1992).
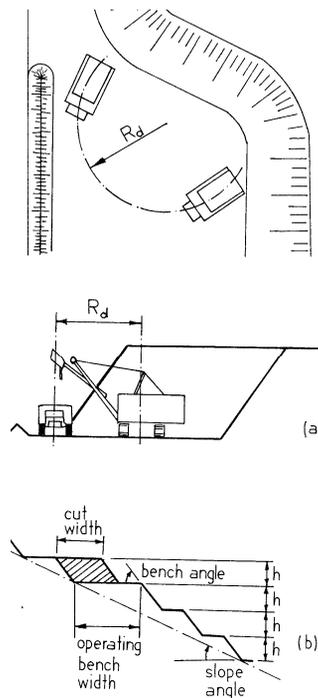


Fig.5.15. Double-spot loading and machine excavation group ($R_d$ is maximum dumping radius) (after Atkinson, 1992)

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h_m = Maximum cutting height
h_o = Optimum cutting height
h_d = Maximum dumping height
R_m = Maximum cutting radius
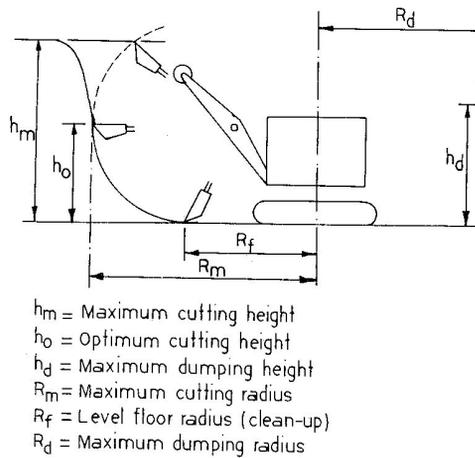R_f = Level floor radius (clean-up)
R_d = Maximum dumping radius

Fig.5.16. Essential dimensions of loading shovels (after Atkinson, 1992)

### 5.3.2.2 Cost Estimation

Ownership as well as operating costs must be estimated. The total cost is the sum of the ownership and operating costs. Where discounted cash flow rates of return are to be calculated, all cash outflows are negative cash flows and must be debited at the time of payment. Initial capital costs for equipment and replacement capital costs must be included in the cash flow at the date of purchase. Ownership costs, other than capital charges must be debited at the date of payment while operating ones, annually. Many shovels have a life excess of 20 years, but this figure is not taken to evaluate depreciation. A company may establish the depreciation period. Freight and insurance costs, erection costs, and interest charges preceded the production period have to be added. It is necessary to take into account the cost of supervision of erection by the manufacturer, which embraces time, subsistence, local transport, labour, erection equipment (e.g., cranes, tools, slings), working costs, etc. Operating costs consist of maintenance and supply costs, electrical power, and labour. However, the estimation described above omits administrative, development, or other charges (Atkinson, 1992).

### 5.3.3 Loading Draglines

The crawler-mounted dragline has its application as an open pit loading machine confined because: (1) it has a less positive digging action, particularly in chopping down operations, than the shovel, and its smaller sizes are not suitable for loading dense, badly fragmented rocks; (2) the cycle time is longer than that of a shovel; and (3) it has poor spotting capability of loading. The good deep-digging characteristics of the dragline may result in reduced gradients for truck and rail haulage operations where the bucket excavates below grade. It can operate from any horizon and, unlike the shovel, can stand on a selected competent bed. Due to its capability of deep-digging, the crawler-mounted dragline can be applied in wet pit operation (e.g., sand and gravel, chalk, etc.) excavating box cuts, recovering remnants from pit floors, etc., and in general work (e.g., digging sumps, extending inclines, etc.). Total costs can be calculated with the procedure used for loading shovels (Atkinson, 1992).

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### 5.3.4 Cyclic Stripping Machines

Large single-bucket stripping machines mainly were developed in the United States to expose comparatively near-surface coal deposits, whereas multi-bucket machines were constructed and improved in German brown coal mines. Nowadays, these machines are applied in all strip-mining operations. Walking draglines, stripping shovels, and bucket wheel excavators are the most important types of cyclic stripping machines. The selection procedure for large stripping machines must be well prepared and performed due to the large capital investments involved (Atkinson, 1992).

Once the production requirements of the mineral loading machines are established and they are used to calculate the output of the stripping machine. The key parameter in the assessment of this output is the maximum volume to be stripped, whereas the maximum depth determines the reach. As the geometry of most strip mining operations makes it possible to expose reserves only to limited extent, stripping and loading facilities must be correctly fitted and machine availabilities must be precisely foreseen. Where necessary machines that can meet the maximum production requirements must be taken into account. If grade varies largely and stockpiling is impractical, small stripping operations may be required to mine a deposit, which involves the application of a greater number of smaller-capacity machines. However, the reach will be the same as for a larger-capacity single machine since the depth of overburden determines the reach of the stripping machines. This increases overall capital cost. Walking draglines and stripping shovels are often defined as single-bucket excavators (Atkinson, 1992).

### 5.3.4.1 Walking Draglines

Walking draglines are used for direct casting operations due to their better (capacity × reach)-to-service weight ratio than that of other single-bucket machines. They can have relatively low bearing pressures when a large tub is applied. However, such the design moves the centre of rotation further back from the edge of the cut, which decreases the reach of the machine. Walking mechanisms are cam-operated, but because weight distribution problems occur, hydraulic systems have been adopted for large machines. Walking draglines were developed in 1980s, and an exemplary machine has a 170-m$^3$ bucket, a 122-m boom, a machine mass (service weight) of 12,700 t, and a connected (ac) load of 36,999 kW. Smaller machines operate efficiently in less consolidated, well-fragmented rocks whereas large draglines are successfully used in strong but well-blasted ground. A highwall is ready for dragline working, when blasting provides satisfactory fragmentation to obtain efficient bucket loading. Moreover, the blasted burden should be sufficiently stable to enable the dragline to dig and move safely. These requirements may lead to buffer blasting and decking of charges where some strata are prone to produce large lumps. Point loading can distort or puncture the bottom plates of the draglines tub therefore in order to 'walk', the machine should have a track prepared. The operation costs should include the price of the bulldozer or other units employed to form the track (Atkinson, 1992).

### 5.3.4.2 Stripping Shovels

The stripping shovel has been used since it is more productive than the walking dragline because of its positive dipper loading action, shorter swing time, and ability to work in strong, dense rocks. The largest machine used in 1980s had a 138-m$^3$ dipper on a 65-m boom with an ac load of 22,500 kW. Self-cleaning crawlers and hydraulic levelling jacks were standards. A competent floor is essential, and large machines may need some preparation of the floor so

that excessive local bearing pressures can be reduced and satisfactory crawler life can be obtained.

Shovel vs. Dragline. The areas of the use of stripping shovels and draglines are nearly the same. The wide application of inexpensive ANFO explosives, the excellent digging depth and reach of the dragline, and its lower capital and operational costs, make the walking dragline more advantageous than the stripping shovel (Atkinson, 1992).

5.3.4.3 Size Selection

To select a stripping machine properly, one should: (1) determine the approximate dipper or bucket capacity, (2) determine the machine geometry, and (3) make selection according to a standard range, and assess the model selected. The approximate dipper or bucket capacity can be assessed using Eq.4.3.1, whereas the mineral production requirements decide the overburden stripping rate determination. Stripping machines are designed to work 22.5hr/day and up to 350 days/yr since enormous capital investments are engaged (Atkinson, 1992).

Machine Geometry. The dumping radius and the dumping height are two prime dimensions to be evaluated. An idealized dragline stripping operation is presented in Fig.5.17. A boom should not be too long. This goal will be achieved when any berms are not applied in dragline operations, the cut width equals the pit width and all exposed mineral is loaded out. If the dragline is placed on the surface (Fig.5.17), the mineral loading and transport equipment requirements can define the minimum width of cut. For small loading shovels, pits can be as narrow as 15 to 18 m; for large shovels, up to 11.5 $m^3$, pits need to be as wide as 24 to 30 m. A narrow cut (and in this case, pit) width not only reduces the boom length but also allows the spoil space to be efficiently used because it makes the valleys between the spoil peaks smaller. Short booms reduce also the dragline cycle time, increasing the capacity of striping. The pit width is also influenced by other parameters. If loading operations need to be more flexible, the pit width has to be increased. Fig.5.18 shows how to solve the problem where the ground bearing strength of the surface is not high enough to support the dragline. A competent bed is chosen in the overburden for the horizon of the working bed. The minimum pit width must enable the dragline boom to swing through 90° when it operates at the place nearest the edge of the highwall (Atkinson, 1992).

An idealized shovel stripping operation is presented in Fig.5.19. The pit must be wide enough to let the shovel swing through 90° when it works in the position nearest the spoil bank. The stripping cut may be less wide than the pit, and a berm of mineral remains. The stripping cut width is equal to the mineral cut width as it depends on the width required for the mineral loading and transport machines (Atkinson, 1992).

 The overburden thickness may differ from place to place where surface topography undulates, mineral seams are steeply inclined or hillside outcrops occur. In this situation, the reach of the dragline cannot be calculated based on average overburden thickness and the maximum one necessitates the machine reach to be extended. The other alternative is to rehandle the spoil by so-called 'haymaking' (Fig.5.20) or by the spoil bridge methods (Atkinson, 1992).

The geometry of stripping machines should be assessed by means of average and extreme pit plans and sections, including both stripping and loading operations. The overburden and mineral thickness, the safe slope of the highwall, and the angle of repose of the spoil are given, and the cut width can be obtained on the basis of the loading and transport needs. Once the width of the berm (if applicable) prepared, for example, for transport or blasthole drilling, is established, the pit width is calculated. Next, the dumping radii and the heights of shovels or draglines are to be determined as well as the bucket or dipper size. Dumping radius can be

increased by a reduction in dumping height, and vice versa. In operations performed in thick minerals, the stripping dragline is the best machine to load out ore (Atkinson, 1992).
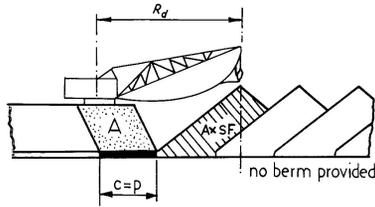


Fig.5.17. Idealized dragline stripping operation (after Atkinson, 1992)
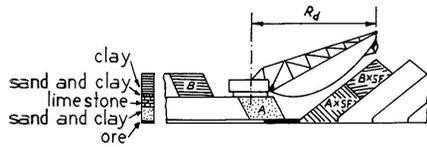


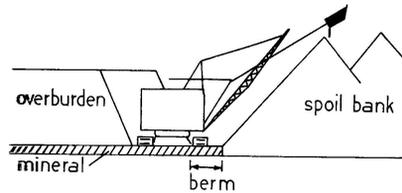Fig.5.18. Dragline on intermediate bench (after Atkinson, 1992)



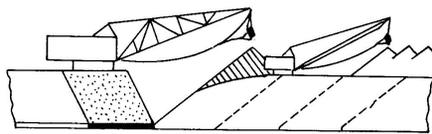Fig.5.19. Idealized stripping shovel operation (after Atkinson, 1992)



Fig.5.20. 'Hay-making' operation: double handling with two draglines (after Atkinson, 1992)

### 5.3.4.4 Cost Estimation

Ownership and Operating Costs. The procedure used to calculate ownership costs for single-bucket stripping machines is the same as for loading shovels. A stripping machine can live usefully longer than 30 years. Large stripping machines are to operate up to 350days/yr and 22.5 hours/day. The erection costs of stripping machines cannot be determined as simply

as for loading shovels. The mass (service weight) may be given in manufacturers' literature. Some care is necessary with very large machines. Operating costs as well as total ownership and operating costs, the cost per unit volume (bank), and the stripping cost per tonne of mineral can be calculated in the same way as for loading shovels (Atkinson, 1992).

5.3.4.5 Selection of Buckets and Dippers

When choosing buckets and dippers one should take their abrasion and wear into consideration to extent their life. This can be done, for instance, by means of liner plates or shrouds that should be located correctly and made of proper materials so that the increase in machine's weight they cause could be reduced. In the case of walking draglines, lightweight buckets can give maximum capacity, but the drag pull per cubic meter excavated decreases. Therefore, they are only suitable for relatively light, easily loaded materials. A lightweight bucket may be economic if a spare one is available. Medium-weight buckets are used for heavier rocks and have the broadest application. Severe duties, or operations where a spare bucket is too costly (e.g., small mines or short-life operations), require heavy-duty buckets of lower capacity. Bucket capacity depends on the difference between the mass of the bucket plus rigging and the maximum permissible suspended load of the dragline. Thus, the bucket capacity may have to be reduced under abrasive conditions, where an oversize drag chain is used to fight wear. The following dipper or bucket sizes can be chosen: up to 15 $m^3$, 16-26 $m^3$, 27-44 $m^3$, 45-57 $m^3$, 58-92 $m^3$, and 93-150 $m^3$ (Atkinson, 1992).

5.3.4.6 Specifications of Stripping Machines

The following specifications of stripping machines can be found in literature (Atkinson, 1992):

For walking draglines: (1) the bucket size can be from up to 11.5 $m^3$ to 85 $m^3$; (2) the dumping radius can be 50-92 m; (3) the dumping height can reach 21-44 m; (3) the width over shoes can be 14-35 m; (4) the machine mass can reach 470-5700 t.

For stripping shovels: (1) the dipper size can be from up to 31 $m^3$ to 140 $m^3$; (2) the dumping radius can be 38-66 m; (3) the dumping height can reach 27-46 m; (3) the width over shoes can be 13.5- 27 m; (4) the machine mass can reach 1300-11000 t.

5.3.5 Continuous Excavators

The most important types of continuous excavators, the bucket chain excavator and the bucket wheel excavator, were developed mainly in the German brown coal fields. Some development has also taken place in Czech Republic, the United States, Australia, Poland, and Russia (Atkinson, 1992).

5.3.5.1 Bucket Chain Excavator (BCE)

Bucket chain excavators are the slewing machines mounted on crawlers, carrying out block-excavating operations. Fig.5.21 shows a large BCE, which was in service in an Italian mine; it has a theoretical output of 1470 $m^3$ bank/hour, a mass (service weight) of 1495 t, and installed ac power of 2285 kW. Moreover, it can be seen from Fig.5.21 how big the machine can be. The BCE is able to mine large outputs in weak unconsolidated ground, but it cannot dig hard ground and is unable to operate selectively. The best and most important characteristic of this machine is its very good downward-digging capability and fairly good upward-digging capability (Atkinson, 1992).

5.3.5.2 Bucket Wheel Excavator (BWE)

The BWE is the most effective machine mainly designed for excavating high outputs in weak unconsolidated ground, but sometimes it can be used in harder formations. It is able to selectively mine faulted or intercalated ground. The machine can cut bands as thin as 0.1 m, but with smaller output. Its downward-digging capability is poor, but a boom sandwich belt, or a special long-boom design can improve this. These designs decrease the boom angle enabling the conveyor to operate successfully when digging takes place below grade. However, in case of handling boulders or large frozen lumps, the sandwich belt does not work as well as expected (Atkinson, 1992).

The BWE is equipped with evenly spaced buckets on the periphery of the wheel. Excavated material is supplied through a transfer point inside the wheel (e.g., a plough, rotating disc, etc.) to the belt conveyor system of the excavator and then discharged. The machines can have outputs of up to 10 000 $m^3$ bank/hour and masses (service weight) exceeding 7000 t.
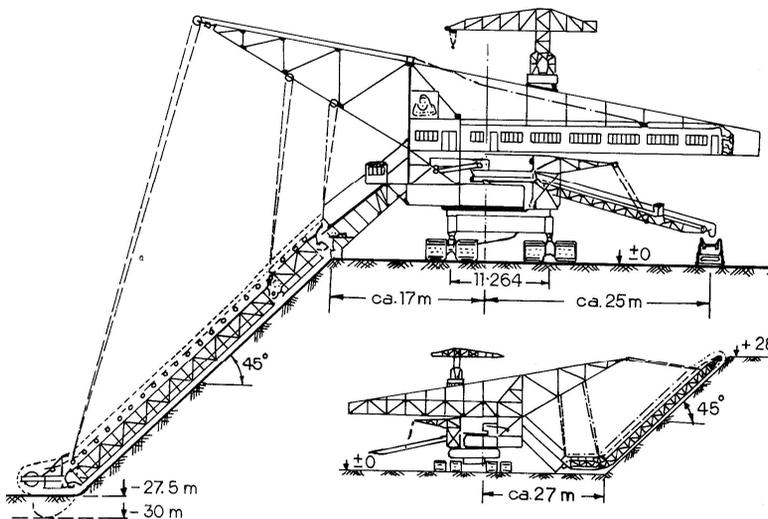


Fig.5.21. Bucket chain excavator (after Atkinson, 1992)

Advantages of Continuous Excavators. Continuous excavators show lower impact loadings than those of single-bucket machines, which can decrease dynamic stresses, machine mass (service weight), maintenance costs, and power consumption. The slower slewing (swing) speed and alleviated digging impacts make load transferring across the crawler gradual, which results in the reduction of ground bearing pressure. Therefore, it is possible to carry out the operation in conditions in which a single-bucket machine cannot operate. Land reclamation is easier with continuous excavators (Atkinson, 1992).

BCE vs. BWE. Since World War II, the BWE has gained wider application than the BCE. According to Atkinson (1992), there are several advantages of the BWE and BCE that should be taken into account while choosing a continuous excavator. They are listed below.

1. The BWE is able to perform selective operation because its action in part-block operation can be horizontal. The BCE cannot selectively mine a deposit.

2. In some circumstances, the BWE can excavate relatively hard ground (sandstone, shales, etc.). Digging hard ground or handling boulders is impossible with the BCE.

3. Maintenance costs of the BWE are lower than those of the BCE, because the BWE has fewer wear parts than the BCE. Therefore, the BCE is less available than the BWE.

4. The BWE has great high-digging capability, while the BCE is of poor efficiency because in high digging it has bucket-filling difficulties. The BCE has excellent deep-digging capability, and the BWE rather bad.

5. The bucket-clearing action of the BWE is much better than that of the BCE, and it can handle sticky materials easier.

The BCE are far preferable to the BWE in the following circumstances, assuming that boulders or hard bands do not occur and mining has not to be selective: (1) Soft, nonabrasive rocks where high and deep cuts must be made with one machine. (2) Situations in which the initial box cut is opened in these conditions. (3) Operations within wet pits. (4) Where reduction in transport gradients is required. (5) Overburden bridge operations where cuts must be taken below grade level of the bridge crawlers. (6) Where exact slope profiles are required. (7) Where there are big undulations in the surface of the mineral bed. Most of these advantages are connected with the better downward-digging capability of the BCE. Most other applications prefer the BWE.

5.3.5.3 Operations

Continuous excavators are designed for the following gradients: excavating 1 in 20, travelling 1 in 10. The BWE can be used in thin mineral/thin overburden strip mining casting and is able to strip deeper overburden or provide a greater pit width than single-bucket stripping machines. In thicker overburden and/or thicker mineral, the advantage of the shortest possible lift and transport distance and lowest stripping costs can be acquired if the direct across-the-pit range of the BWE and the BCE is extended by means of (1) the mobile stacker boom (Fig.5.22) and (2) the overburden bridge (Fig.5.23). Because these units show restricted operational flexibility, the following conditions should exist: (1) sufficient proved reserves to justify the capital cost of the expensive machines, (2) nearly horizontal stratification over the deposit, and (3) uniform geologic conditions (i.e., lack of major faulting, large undulations, great variations in overburden thickness, etc.). Continuous excavation is mostly applied in weak sedimentary deposits of relatively great thickness where direct transport of the overburden across the pit is useless due to excessive pit width. Most of these operations utilize several benches, and a conveyor or rail transport system has to be employed to transport overburden round the pit and then the overburden is dumped in the spoil space prepared by the advancing pit (Atkinson, 1992).
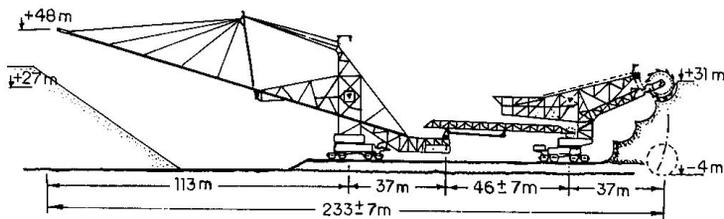


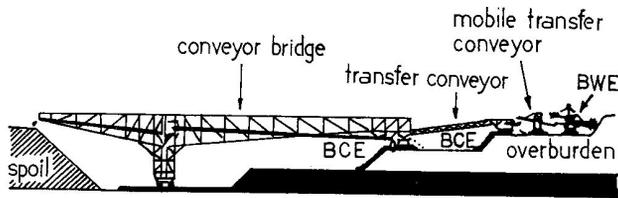Fig.5.22. Mobile boom stacker for BWE (after Atkinson, 1992)

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Fig.5.23. Overburden bridge for BWE (after Atkinson, 1992)

The full-block method with BWEs, presented in Fig.5.24, is applied if mining has not to be selective. The part-block method is less volume efficient since the ground being excavated has the thickness of below the optimum value. The method is designed for selective mining operations because it involves greater travelling time that results in increased crawler wear and reduced digging time. Short booms are not useful for part-block operations. If part-block operation is to be performed at a horizon high up the bench face, an oversize bucket wheel should be employed to make clearance for the boom to keep an adequate fair block width. The mass (service weight) of BWEs armed with booms having crowd action is bigger than that of BWEs with rigid booms. In the case of large BWEs cutting big bench heights, a crowd action boom needs a long crowd travel motion, which can instigate problems with ground bearing pressure (Atkinson, 1992).



Block-digging operation with wheel excavator feeding on to a movable conveyor belt
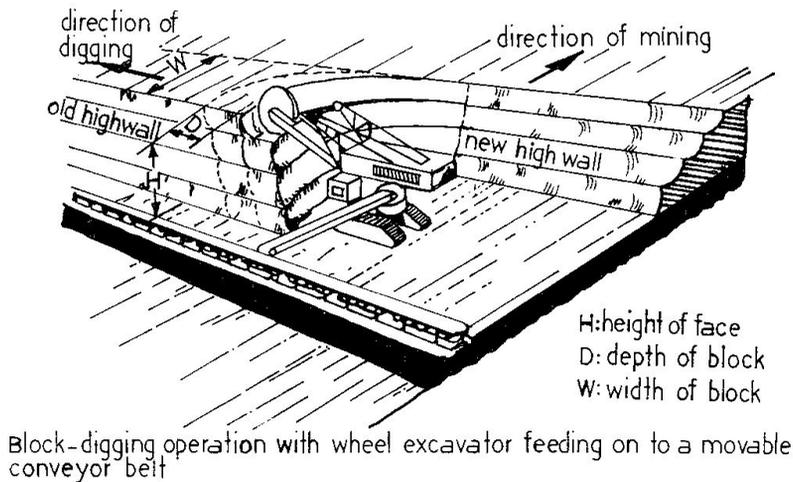
Fig.5.24. Full-block operation, bucket wheel excavator (after Atkinson, 1992)

Transport Systems. The continuous excavators and their transport systems must be planned and engineered as a complete system. The two prime systems used with continuous excavators for round-the-pit transport are (1) high-speed belt conveyors and (2) locomotive rail haulage. The side-slewable, high-speed belt conveyor is becoming far preferable to rail transport because it can operate on steep slopes (1:3). Moreover, large areas used by rail track, especially in deep pits, are eliminated and the time spent by the excavator in waiting for the transport system is reduced, which means that the whole system is continuous. The main disadvantage of high-speed conveyors is their high capital cost and high power consumption.

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Rail transport can be used to travel long distances on horizontal grades. As each block width (Fig.5.24) is mined, the transport system must be moved forward. To carry out this operation less frequently and therefore to avoid the production lost, a mobile, crawler-mounted, transfer conveyor can be used between the excavator and the transport system to extend the range of the excavator (Atkinson, 1992).

Crawlers. As continuous excavators usually operate in less consolidated formations (e.g. lignite deposit), it is necessary to keep a low value of ground bearing pressure, and rigid crawlers that shovels employ should only be used for very small machines (Fig.5.25 (a)). Larger machines need crawler tracks with two wheel bogies so that the crawlers can move on uneven terrain (Fig.5.25 (b)). The largest machines require the crawler design shown in Fig.5.25 (c). The larger excavator and the crawler bearing area, the greater the introduced ground bearing pressure because a pressure cone is formed beneath the crawlers. Large spreaders with the crawler design presented in Fig.5.25 (c) which work on consolidated spoil introduce the largest bearing pressures. Large machines require steerable crawlers to have their manoeuvrability improved (Atkinson, 1992).
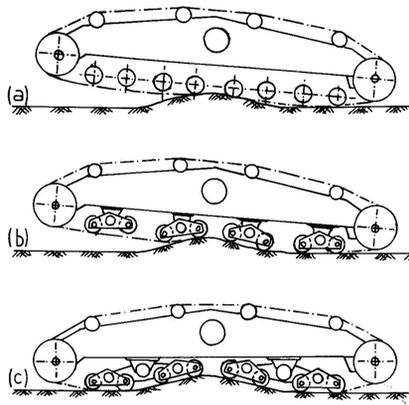


Fig.5.25. Continuous excavator crawlers (after Atkinson, 1992)

Hard Ground Operation. Large bucket wheels excavators with high wheel speeds and slower slewing speeds can mine materials that shovels are able to mine only with blasting (e.g., hard clays, phosphates, frozen oil sands). This is only possible with machines of mass big enough to transfer the cutting forces through the crawlers to the ground. If the rock does not contain large boulders that will not pass through the bucket wheel, ground with a relatively small compressive strength can be excavated without ground preparation. The production rate of the BWE in hard ground is much less than the theoretical output because of loading difficulties and flow difficulties through the bucket wheel. To avoid difficulties resulting from breaking out from the face and spilling of large lumps one should obtain uniform fragmentation (Atkinson, 1992).

Machine Geometry. The machine geometry of a continuous excavator is determined by the pit layout. The overall slope angles involved are usually not steep because (1) the rocks excavated are relatively weak, and (2) the deposits mined may be relatively flat. Furthermore, as advance is lateral rather than vertical, advance stripping is quite economical. Consequently, the benches are wide, and space restrictions are absent. An excavator needs suitable reach to work on both sides of the transport system if it must carry out upward and downward digging. However, in a rotating pit operation the excavator has to be able to operate up the pivot point,

where space is confined. In a parallel advancing pit, where end-bench, lift conveyors may be applied, the excavator has to dig up to the corner of the pit. The machine geometry may limit the pit layout, and detailed planning is required before the selection of final machine geometry. Sometimes, a mobile, crawler-mounted, transfer conveyor has to be used, and for large machines, a telescopic discharge boom conveyor may have to be implemented into the excavator design (Atkinson, 1992).

Ownership and Operating Costs. Continuous excavators can have service lives longer than 30 years, but tax concessions, etc. can change the write-off period. It is difficult to compare single-bucket excavators with continuous excavators because their duties are different, but under the same conditions (production rate × reach), continuous machines are much more expensive. They are also part of a continuous system, which in a multi-bench mine, must be synchronised with other excavator-transport systems, therefore detailed planning of mine operations is essential. Ownership and operating costs can be estimated with the method for loading shovels (Atkinson, 1992).

### 5.3.6. Mobile Equipment

Mobile earthmoving equipment applied in the construction industry is also widely used in mining operations. Front-end loaders (FEL), tractor-scrapers, bulldozers, and rippers represent the prime types of this equipment. Before the implementation of cheap explosives, they have been used only in soils and the weaker, less consolidated rocks. Since then it has been possible to excavate more consolidated and harder rocks (Atkinson, 1992).

### 5.3.6.1 Crawler vs. Rubber-tired Mounting

Crawler-mounted units have the following features: (1) they demonstrates strong digging capability as a front-end loader or as a tractor scraper; (2) their traction effort is very good in severe ground conditions and posses low ground bearing pressure, and; (3) they operate well on steep inclines; (4) their manoeuvrability is excellent; (5) they can be economically employed only on the short distances between digging and discharge points because they are able to travel rather slowly; (6) maintenance costs in abrasive materials are high (Atkinson, 1992).

Rubber-tired units have the following properties: (1) their mobility is very high; (2) maintenance costs in easy/medium conditions are comparatively low; (3) when they move on sharp, broken, or abrasive rocks (e.g., basalt, angularly broken schists, etc.), tyre costs increase to high degree; (4) they are less manoeuvrable than crawler-mounted units, however, centre-articulated units are of higher manoeuvrability than that of conventional excavating and loading machines; they operate well on narrow benches; four-wheel-drive units have much better manoeuvrability than two-wheel-drive ones; (5) their travel speeds are high, therefore, they dig and transport excavated material over longer distances than crawler-mounted machines; this feature has contributed to the development of rubber-tired units as load-haul-dump machines (LHD), e.g., FELs and tractor-scrapers; (6) ground bearing pressures are high, therefore high rolling resistance decreases the units' efficiency (Atkinson, 1992).

### 5.3.6.2 Front-End Loaders

Rubber-tired front-end loaders have been used extensively in open pit mining. Units with rock buckets up to 27 m$^3$ can be met, however, in metal mines, the FEL has been mainly applied for weak overburden stripping and auxiliary operations. They are short-life machines.

FELs are widely used in face loading since they have the following features: (1) mobility, the FEL can quickly travel from one part of a pit to another, move out of blasting zones, etc.; (2) low capital costs; (3) operational flexibility, FEL can carry out many auxiliary operations (e.g., haul road construction and maintenance, stock-pile and blending operations, drainage operations, ripping, truck 'busting' on inclines, etc.) (Atkinson, 1992).

Production Rate. Because of its mobility, the most important factor influencing the production rate of a wheel loader is the operating load (bucket load) rather than bucket capacity, since if a FEL operates at maximum load when handling a light material, this power will be insufficient to handle a dense material. The operating cycle consists of (1) loading, (2) hauling, (3) dumping, and (4) return (Atkinson, 1992).

Cost Estimation. The economic life of a FEL depends on the condition of service, the skill of the operator, and the management supervision. Because of the relatively short life of FEL tyres, their cost is subtracted from the machine cost and considered to be an operating cost element (Atkinson, 1992).

Front-End Loaders vs. Loading Shovels. The advantages and disadvantages of both machines are as follows (Atkinson, 1992).

FEL Advantages: (1) excellent mobility, (2) good versatility, (3) can easily operate on moderate grades, (4) lower capital costs, (5) never outmoded, (6) only one operator is needed, (7) large lumps are not trapped in the bucket, as can happen with a shovel dipper.

FEL Disadvantages: (1) unsuitable for hard, dense rocks that are fragmented badly because of discontinuities such as open fissures, joints, bedding planes, (2) ground preparation must be precise; overall costs can be decreased by increased blasting, (3) tyres require special attention to avoid excessive costs, (4) high operating costs, (5) greater operator fatigue, (6) high ground bearing pressures, highly influenced by bad floor conditions.

Shovel Advantages: (1) it is able to dig hard, dense, badly fragmented rocks, (2) low operating costs, (3) less sensitive to poor quality maintenance than wheel loaders, (4) reduced operator fatigue, (5) lower ground bearing pressures and not so seriously affected by poor floor conditions.

Shovel Disadvantages: (1) Lack of mobility; for multi-bench, sequence operations in open pits with a number of shovels, this need not be a disadvantage, (2) single-purpose loading machine, (3) higher capital costs, (4) cannot travel on steep grades, (5) quickly outmoded, (6) support equipment may be needed to clean up bench, etc.

Crawler-Type Front-End Loaders. The crawler-type FEL in open pit mining is mainly used as a support machine. It is not as mobile as the rubber-tired loader, but it is much more mobile than the loading shovel, and is highly useful for excavation of inclines, sumps, etc. In the face loading operation, the machine can work on a V-shaped path with overall cycle times between 0.7 and 1.0 min (Fig.5.26). For excavating inclines, etc., a fixed time can be 0.65 min, and an overall haul speed can be of 4 km/h. The cost estimation method used for rubber-tired FELs may be used, but there are no tyre costs (Atkinson, 1992).
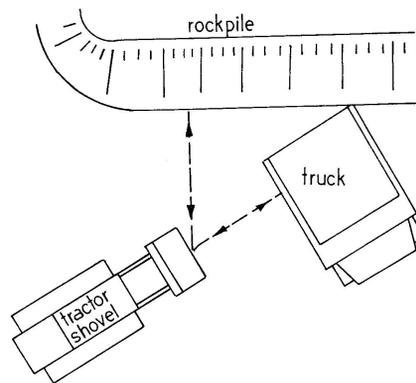
Fig.5.26. V-shaped path: crawler-mounted, front-end loader operation (after Atkinson, 1992)

### 5.3.6.3 Hydraulic Excavators

The hydraulic excavator has been developed mainly in Europe. A large machine is very effective in difficult conditions in large open pits. The machine with highly effective working mass (180-t) and flexible front-end geometry was successfully introduced to metal mining where rope-type shovel had been mainly used. The face shovel machine is usually equipped with an opening dipper bullclam. The hydraulic excavator develops high-crowd, prying, and breakout force characteristics due to its flexible geometry. It can mine selectively and follow the contours of irregular beds much better than a rope-type shovel. In the back-hoe mode, the hydraulic excavator can stand on top of a bench and load a truck spotted on the bench below. In this position, the swing angle is small and therefore the cycle time is short, which results in high production. As the excavator operator looks directly into the truck body he can spot the bullclam over the truck body very easily. The back-hoe mode may also be used to load trucks on the same bench when it is difficult to operate trucks on the bench below. Most hydraulic excavators are diesel powered, but electrical ones (with cable reels for the trailing cable) are also produced (Atkinson, 1992).

Cost Estimation. The usual estimating procedure is used with hydraulic excavators. In calculating ownership costs, the machine life is considered medium (8 to 10 years). The larger units live longer. In estimating operating costs, availabilities of 90% or more are used. The electric drive has lower operating costs than diesel ones (Atkinson, 1992).

### 5.3.6.4 Tractor Scrapers and Bulldozers

After ground preparation, in many medium and strong rocks blocky fracture can appear. Consequently, they cannot be excavated with tractor scrapers. These scrapers can be used mainly for overburden stripping, where their productivity and mobility are significant. However, they are also used in medium rocks that can be well fragmented with blasting or ripping. The tractor scrapers are load-haul-dump machines. There are four main types of them: single-engine conventional, tandem-powered, elevating, and auger tractor scrapers. The single-engine conventional scraper is used most widely. However, it cannot operate in very easy, downhill conditions and a pusher tractor is needed for loading. The tractor then boosts the scraper to haul speed. The conventional machine does not work well where grades are steep and adverse, rolling resistances are high, floor conditions are poor, or where the haul distances are short. The tandem-powered machine has an all-wheel drive from front and rear

engines, which makes a power/weight ratio high and traction effort suitable for conditions with high rolling resistances and adverse grades. It can, therefore, work where other scrapers cannot, particularly in mud or when dumping in wet conditions on poorly compacted spoil heaps. The tandem-powered scraper operates with the assistance of a pusher tractor, but it can partially self-load or even fully load on downhill runs. Pairs of tandem-powered scrapers are used in the push-pull method. The two machines link up through push blocks and a hook and bail assembly and enter the cut; the second scraper push-loads the first, the first then pull-loads the second; they disconnect and drive individually to the dump. Elevating scrapers are self-loading and are suitable for short to medium haul distances where machines have to wait for the pusher tractor. To avoid queuing of scrapers at the start of each cut, the number of scrapers must be correctly matched to the number of pushers. They do not perform well on adverse grades or with high rolling resistances and cannot efficiently handle sticky materials or materials containing rock or boulders larger than 200 mm. Because of its ability to work with no assistance, the elevating scraper is highly suitable for selective mining or for small-fleet operations. The action of the elevating flights also breaks down and helps to blend the material being loaded. It can be easier spotted over dumping hoppers than other scrapers. The auger scraper is a self-loading machine considered an alternative to conventional, push-pull elevating machines. The auger is placed at the centre of the bowl, and as material flows over the scraper's cutting edge, it is fed to the rest of the bowl by the rotating motion of the auger. The auger scraper requires a shorter cut distance, has very good ejection characteristics, increased tyre life, it retains material well when hauling, and can work in a wide range of conditions. It cannot efficiently handle material containing large rock fragments or boulders larger than 300 mm (Atkinson, 1992).

Equipment Selection. To select the equipment well, the following factors should be taken into account: (1) Scraper Cycle Times; (2) Scraper Production (The load hauled is expressed in bank volume and is based on scraper heaped capacity, fillability and swell factor); (3) Operational Efficiency (OA); (4) Fillability; (5) Swell Factor; (6) Pusher Tractors (; (7) Push-pull Operations.

Cost Estimation. This procedure may follow one used to estimate costs of shovels (Atkinson, 1992).

Bulldozer. Although not usually employed as a primary excavator, the bulldozer is very often used in surface mining. The crawler-mounted machine is more frequently employed, and 63-t units have been in operation; but because of its mobility, the rubber-tired bulldozers are more frequently chosen. For short hauls and severe conditions where scraper operation would be difficult, bulldozing can be applied (Atkinson, 1992).

5.4 Design and Layout of Haul Roads

In surface mines, off-highway trucks with capacity of up to 360 t have been used. The capital investment and costs that the trucks involve are of a significant value. The machines can be hazardous when operating within a limited area of a surface mine if haul roads are not designed properly. The haul roads have to ensure transport efficiency and work safety. Haul road design parameters should provide: (1) Minimum costs for the transport of mineral and waste; (2) A minimum amount of traffic congestion and safe, ready access to the mining operations; (3) The exclusion of areas with potential slope stability problems; (4) long-life haul roads rather than short-life ones. Consequently, haul road overall construction and operating costs can decrease. Moreover, the demand for construction materials, which may not be available in sufficient amount from the overburden, can be reduced. Other factors are as follows: the locations of mineral preparation plants, stockyards, external waste dumps, and environmental constraints. Therefore, to make a good design one should consider: (1) haul

road layout, (2) haul road geometry, and (3) haul road construction materials (Atkinson, 1992).

5.4.1 Haul Road Layout

5.4.1.1 Strip (Open Cast) Mines

In these operations, T Junctions (Fig.5.27) play the most important role. These junctions are created at places where the haul roads (access ramps) through the spoil and the strip (pit) cross each other. At these places, the spoil heaps must be piled higher than elsewhere because some spoil space is reduced by the space the access ramp requires. This spoil space elimination results in spoil rehandling. Consequently, spoil heap failures occur since dumping cannot be correctly performed. Because the side slopes are best cut from placed soil, spoil should not be dumped directly to form T Junctions. The use of placed soil produces more stable slopes since consolidation related to self-weight may form substructures within spoil heaps. Therefore, they can be excavated to stable angles steeper than the angle of repose of the spoil, which can decrease the volume of necessary rehandle. Because strip mines are generally flat, the bottom-dump truck is the preferred haulage unit. If gradients are greater than 6% (3.5°), rear-dump trucks may be necessary. Therefore, during layout, special attention should be paid to dips, pavement contours, etc.

There are no serious problems with construction materials where competent materials occur in the overburden. However, where there are more mudstone, silts, clays, etc., the import of materials for haul road construction may be required. Temporary haul roads within the strip need the pavement to be of suitable material. Where it is not, the haul road may be located on the top of the mineral bed. In this situation, the loading shovel (located on the floor) must have its boom length increased or a back-hoe excavator should be located on the top of the mineral bed.

When cast blasting is used, the access ramps can be located on the highwall side of the strip. In this case, there are not spoil stability problems at the junctions of the access ramps with the strip. Furthermore, the prime stripping operation may be less frequently interfered. To be successful, cast blasting requires comparatively strong overburden that will be almost vertical at the highwall. The haul roads are constructed within and outside the overburden on the highwall side and sufficient amount of suitable material should be provided (Atkinson, 1992).
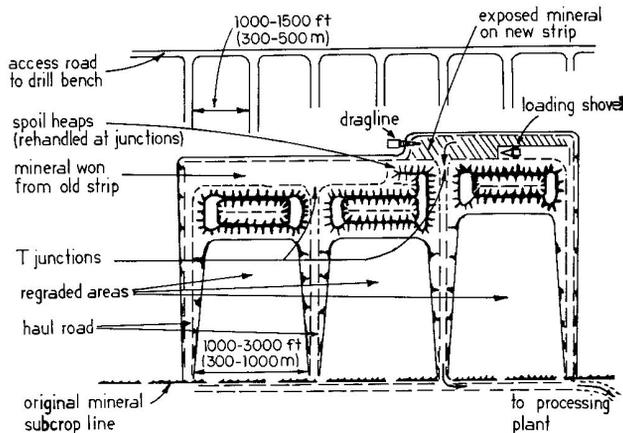


Fig.5.27. Diagrammatic strip mine layout (after Atkinson, 1992)

5.4.1.2 Terrace Mines

In bucket wheel excavator/conveyor mines, haul roads are required for access by service vehicles that travel at medium speeds. Haul road requirements are not an important issue apart from the need for bridges to cross conveyors at key points. Such mines are excavated in unconsolidated deposits (e.g., lignite mines) and there may not be materials suitable for haul road building. The need for volume is small and in the case of lignite mines with a captive power station, the use of fly ash can be satisfactory. Temporary major haul roads are required in shovel/truck terrace mines (Fig.5.28). Terrace mining can be used for stratiform deposits with dips in excess of 14% (8°), where spoil stability problems may occur if down-dip strip mining is applied. Efficient truck haulage on the full dip is impossible because the strata dips are too steep and haul roads must be made in the overlying strata. Forming haul roads in underlying strata can cause instability since it undercuts the footwall bedding planes, as well as increases stripping volumes. Where suitable materials occur in the overburden, haul road construction is not a problem. Where there are not such materials, they must be imported in huge amounts (Atkinson, 1992).
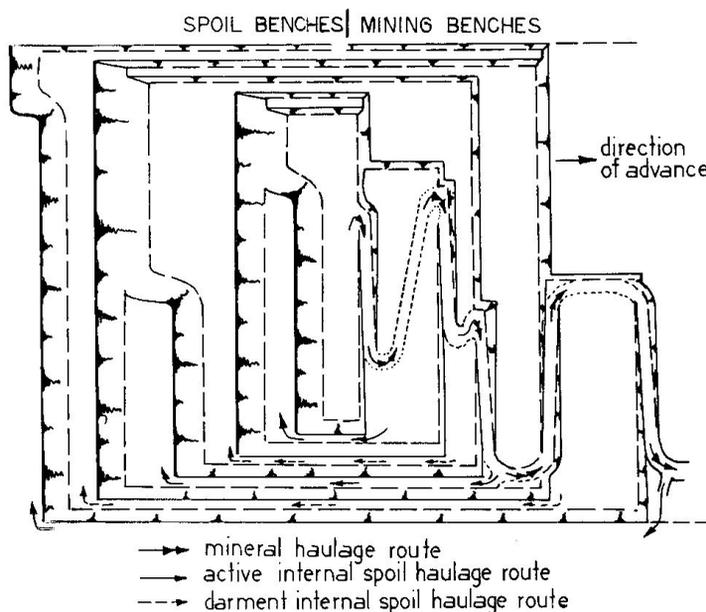


Fig.5.28. Diagrammatic terrace mine layout (after Atkinson, 1992)

5.4.1.3 Open Pits

A benched, inverted conically shaped open pit is designed for excavation of massive ore bodies, steeply dipping stratiform deposits, and coal accumulations. The facilities external to the pit (i.e., mineral preparation plant, waste dumps, etc.) are located with regard to topographical, geotechnical, and environmental requirements. These locations may influence the alignment of the access roads to the top of the pit. The in-pit access haul road (Fig.5.29) may be a clockwise or a counter-clockwise spiral, or a switchback or zigzag (where the haul road turns back on itself). The main factors influencing the choice of layout are: (1) It may be

possible to locate permanent access haul roads on a 'tight' side of a pit (e.g., the footwall of a stratiform deposit) and then switchback layout can be applied; (2) In large pits, if a spiral layout is used, the haulage distance may be too great; (3) Areas where slope stability is endangered should be avoided, which eliminates a spiral layout; (4) The pit walls may be too steep for the switchback layout because suitable bends cannot be formed without the stripping ratio increase. In this case, the spiral layout is the most suitable. Similarly, branching haul roads may influence the choice of layout. Gradients of 8 to 10% (4.5 to 6°) are adopted for rear-dump trucks and a 12% grade may be adopted for trolley trucks (Atkinson, 1992).
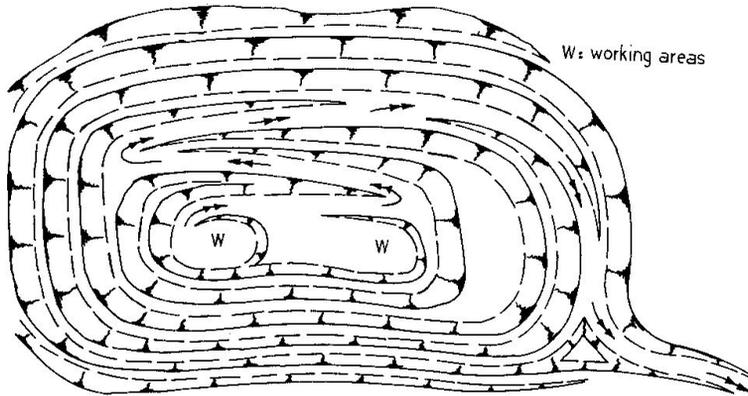


Fig.5.29. Diagrammatic open pit layout with switchback haul road (after Atkinson, 1992)

Many metallic deposits are either igneous or metamorphosed, and haul road can be built without difficulties. Stratiform deposits, including accumulations, have the same materials problems as strip and terrace mines, but the bedding planes often dip into the pit, and the location of haul roads may be difficult. Ideally, overall pit slopes are determined with geotechnical parameters. Bench heights are set by the operating requirements of the excavator. Loose material from minor falls must be contained on safety benches or berms. These safety benches flatten the overall slope and are left after every several lifts (Atkinson, 1992).

Haul roads also flatten the overall slope (Fig.5.30). In some cases, the geometry implemented by safety benches and haul roads flatten overall slopes more than it is required by slope stability factors, which results in stripping cost increase. The location of the haul road within or outside the ore body can also influence mining costs. Fig.5.35a shows the haul road located outside the ore body. The location increases the volume of minable ore but also the stripping ratio. Fig. 4.35b shows the haul road located within the ore body. The stripping ratio is reduced, but the minable ore volume is also reduced by the haul road construction (Atkinson, 1992).
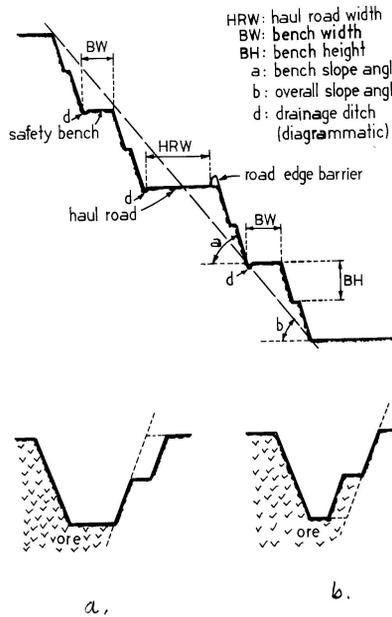
Fig.5.30. Haul road geometric considerations, a. haul road outside ore body, b. Haul road in ore body
(after Atkinson, 1992)

## 5.4.2 Haul Road Geometry

The following items are considered when determining the haul road geometry: (1) number of lanes, (2) safe distance between trucks, (3) road width, (4) super elevation, (5) gradients, (6) safe sight distances, (7) haul road signs, (8) lighting, (9) runaway precautions, (10) cross slope, (11) drains and culverts (Atkinson, 1992).

In-pit roads are constructed for single-lane, unidirectional traffic or two-lane, directional traffic because traffic density may not be high or because of space problems. Haul roads from the pit to external waste dumps, preparation plants etc., may require more than a single lane per one direction. The safe distance between trucks depends upon driver reaction time, the gradient, and the road surface plus an allowance. The widest vehicles applied determine the haul road width. For straight, regular grade roads, the rules of thumb are adequate. For sharp curves, additional width must be included, both on the curve and the tangent to the curve, to cover the front and rear overhangs of the vehicle and the difficulty of negotiating the curve. Trucks negotiating tight curves are subjected to an outward centrifugal force, which is opposed by the side friction between the tyres and the road surface. A good surfacing material is essential on sharp curves, and super elevation of the road surface is included in the haul road design. There are limitations to super elevation, since trucks driven at slow speeds on sharp curves could overload the tyres on the inside of the curve, and in areas of ice, snow, and heavy rain, tend to slide towards the inside of the curve. Where possible, all the super elevation should be uniformly introduced in the tangent to the curve. Maximum gradients may be limited to between 8 to 15% (5° to 8.5°) for sustained gradients, but when considering the economics of uphill haulage, as well as downhill safety, the optimum gradient for most situations is about 8% (4.5°) but up to 12% (6.8°) for trolley-assist trucks. Runaway trucks

can be a serious hazard on steep downhill gradients, and safety provisions to guard against these hazards must be provided as part of haul road design. Where possible, a level surface between road edges is preferable, since this provides more even tyre loading and less driver fatigue. Where heavy rain is experienced, a cross slope is required. A degree of cross slope must be a compromise that provides adequate drainage without adding adverse tyre loading conditions and driver fatigue. Single- and two-lane haul roads may have the cross slope in one direction, while on benches, it should be inwards. However, three- and four-lane haul roads may have a centre high point with the cross slope applied in both directions. Road edge barriers (rock boulders etc.) should be located between roads and adjacent excavations. Run-off water can create major problems due to washouts, mud slides, and saturation, which makes drains and culverts essential. The degree of drainage is dependent on rainfall, catchment area, ground conditions, depth of road base, storm water disposal conditions, etc. The drains are more easily constructed and maintained. Culverts are required to conduct run-off water beneath and away from haul roads (Atkinson, 1992).

### 5.4.3 Haul Road Construction Materials

Many mine operators, in the interest of capital cost, simply cut or fill haul roads with the materials existing at the location. This can raise various problems since (1) in weak materials, the road surface will deteriorate and (2) where competent rock is exposed, it is probable that irregular and jagged edges may form. Both problems may result in reduced production and increased costs of fuel, maintenance, and tyres. The primary task is to construct a roadbed or pavement that allows one to transfer wheel loads over the subgrade (i.e., foundation materials) without an excess of the subgrade bearing capacity. Both subgrade and roadbed materials are to be tested to estimate the bearing capacity. Conventional design methods for flexible pavements are inadequate for mine haul roads, since they are concerned with on-highway trucks having axle loadings less than 10 t. Mine trucks may have axle loadings that exceed 60 t. Assessments should be made of the quantities required and the availability of materials within the scheduled mining operation; it is possible to find the most suitable materials in the later stages of working. Roadbed thickness, widths, lengths should be given a special attention particularly when materials are to be imported (Atkinson, 1992).

Subgrade must be graded to an even surface. In many surface mines, this is the only stage in road construction. Short-life haul roads require conditions inadequate for long-life haul roads. The subbase width should exceed that of the surface and base course. On-site materials compacted in layers should be used. The most suitable materials for base courses are those having high bearing capacity such as crushed rock or high-quality gravels. Large lumps of rock are excluded, as well as organic materials, clay, and silt. Materials should be compacted as for the subbase. The best surface courses are those that give hard, even surfaces and little dust with no penetration under load. High adhesion and low rolling resistance coefficients are preferred. The best surface materials that also have a low permeability are concrete and asphalt. Finer crushed rock, coarse sand-size mill tailings, clean gravel, and clinker are suitable for low cost materials if compacted when moist. Only carefully constructed haul roads will permit safe, high running speeds that will both improve production and reduce maintenance, tyre and fuel costs (Atkinson, 1992).

## 5.5 In-Pit Crushing and Conveying

In-pt crushing is performed by means of mobile crushers. The mobile crushers allow the quarry operator to haul the excavated material with continuous belt conveyors. They also

make it possible to avoid high-cost road construction and operations in wet soft ground, which reduces overall cost. Mobile in-pit crushing and conveying systems are mostly applied in quarries because of ground conditions in such mines. These installations are fed directly by a mine face excavator (Frizzell and Martin, 1992).

5.5.1 In-Pit Crusher Systems

Crushing is done in-pit so that the material could be transported out of the pit by means of a conveyor system. The units used for primary crushing are jaw gyratory, gyratory, hammer, impact, roll, and jaw crushers. In-pit crusher systems may be fully mobile units used continuously or fixed units. Fixed crushers make haulage trucks travel a gradually increasing distance from the mining face. The term 'mobile' describes any type of crusher that can be moved (i.e., is movable). The system is mounted on a frame base, enabling the complete unit to be moved by skids, or a transporter such as a crawler system or a walking mechanism. A mobile crushing plant consists of a hopper, feeder, crusher, and discharge conveyor, as well as the necessary auxiliary and electrical devices. Several types of this equipment such as the mobile, semi-mobile, movable, portable, modular, and semi-fixed crushers are available. The mobile crusher works at the mine face, is directly fed by an excavator, and moves with the excavator on its own transport mechanism as mining progresses. The semi-mobile unit works close to the mine face but is moved less frequently than a mobile crusher. The movable crusher is centrally located in a mine near the same level as the mine's working face. It is relocated, as required. The portable unit has temporary support foundations to be moved in sections. The modular crusher station can be disassembled into modules and reassembled in a new location in about a month. The semi-fixed crusher is mounted on a steel platform, which reduces the need for a concrete foundation (Frizzell and Martin, 1992).

5.5.2 Transporter Technology

Mobile crushers with weight of up to 725 t can be moved by crawlers, tyres, and walking mechanisms. The smaller and more mobile units are moved by skids, tyres, or crawlers, and the larger semi-mobile units are displaced with crawlers or walking mechanisms. Rail wheels can be used as a transport system because of their low rolling friction, which provides great efficiency. However, their use is limited to grades of 10%. Crawlers are widely used in moving heavy equipment, such as mine shovels and dozers. Single sets of crawlers can be teamed to move heavier loads. Crawlers are also able to move at high speeds when loaded. They are usually diesel-powered and have hydraulic lift mechanisms. The weight of a 900-t lift crawler transporter can be about 180 t. It can also travel on uneven ground. However, they are limited to grades of 10% or less when transporting a crusher. Crawlers are very durable, but they are prone to shock and vibration if they are left loaded under a working crusher for long periods. They can almost complete a turn in one spot by running one set of treads while locking the other. Wheels can be tired or rail and are the most commonly used machines to transport heavy loads. Multiple wheels can carry huge loads, but at highly limited speed. Drive mechanisms for tyres embrace electric motors like those used in haulage trucks and hydraulically operated wheels. Walking mechanisms are the most widely used transporters to move loads in excess of 900 t. Machines have been built that can lift up to 4500 t and travel at speed of 60m/h. These systems are less expensive than crawlers. They require careful ground preparation so they will not be impeded or the load redistributed by small bumps. Spillages around a crusher from dumping or from the conveyors must be cleaned up before the units can be moved. The walking mechanism is very durable and can be left for long times under load (Frizzell and Martin, 1992).

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### 5.5.3 Conveyor Systems

The conventional conveyor haulage has problems in open pits since the maximum vertical slope angle a conveyor can reach is 17 or 18°, which is highly different from the typical slope of an open pit wall, achieving 38 to 45°. Therefore, the conveyor must be run out of the pit in switchbacks (similarly to truck haulage roads) or a trench must be cut through the side of the pit. Transfer systems must be installed to enable the change in direction of the conveyors at each switch point or a huge amount of material must be removed from the pit to form a lower slope angle. To solve these problems, high-angle conveyors were developed to remove material from a pit. There are many types of belt systems available to create a conveying system. These belt systems can be divided into the following categories: (1) fabric (canvas) belt systems, (2) steel cable belt systems, and (3) cable-supported belt systems (Frizzell and Martin, 1992).

### 5.5.4 Auxiliary Systems

Auxiliary items include the feeder to the main conveying system and rock transfer from one conveyor flight to the next. There are three major types of feeders used under the primary crusher ore pocket: (1) Overlapping flight apron feeder (It looks and functions like a crawler tread where the crushed output falls directly on top of the tread. The ore is then carried out over the end of the feeder and dropped on to the conveyor); (2) Vibrating feeder (It may have a flat pan and may be vibrated by vibrating heads); (3) Belt feeder (Belts are used under primary crushers as feeders) (Frizzell and Martin, 1992).

Transfer Chutes. If the conveyor flights must leave the pit in switchbacks, the means of transferring material from one conveyor to another will be of the most importance. The transfer height between one belt and another should be reduced by nearly laying the material on the receiving belt. A transfer system stops the forward movement of the material being transferred in one direction, turns it, and starts it in a new direction. Transfer is accomplished by dumping the conveyed material into a rock box, reducing the material energy. Then the material is transferred by chute or slide onto the next conveyor in the direction of that conveyor movement (Frizzell and Martin, 1992).

# 6 Surface Mining: Mechanical Extraction Methods

Surface mining is involved in extraction of minerals from water and sediments of rivers, lakes, seas, and oceans; from solutions flowing through fragmented rock and soil; and from rock and soil excavated from the earth. Surface mining contributes to the most of mineral production. This chapter deals with extraction technologies that apply mechanical methods to obtain minerals from the earth in comparatively dry conditions. Extraction methods used in coal surface mining are briefly depicted. Since reclamation is of vital importance in all the operations that have an environmental impact and damage or even devastate the earth, mechanical extraction methods can be classified as employing either deferred reclamation or concurrent reclamation. Deferred reclamation is employed when excavation should remain open throughout the mine's life. Once economic reserves exhaust, the opening is reclaimed with regard to all the standards. Methods employing deferred reclamation are referred to as open pit or quarry mining. Metallic ores, industrial minerals, and coal are extracted with open pit methods. Typical examples are presented in Chapter 6.1. Dimension stone is also produced in surface mines which remain open throughout the economic life of the deposit. Quarry mining is dealt with in Chapter 6.2. In many countries, concurrent reclamation is obligatory for most coal and industrial mineral deposits excavated with surface methods. These methods may be classified as area mining (open cast, strip, or furrow mining), modified open pit mining, contour mining, and auger mining. In all four methods, extraction and reclamation are performed simultaneously and strongly relate to each other. Area mining deals with vast tracts of land and is designed for nearly horizontal bedded deposits. In modified open pit and contour mining, the excavations are longer in the direction of advance in relation to the width. To reclaim the destroyed piece of land the rearward part of the pit is back-filled as the forward part advances. There are many variations in area and contour mining, which deposit geometry and equipment used for excavation and haulage influence. Typical examples for coal extraction are presented in Chapter 6.3. In auger mining, parallel, horizontal openings are bored into thin coal seams. It is employed where area or contour mining as well as conventional underground methods cannot be economically used or technically feasible. This method uses the unique specialized excavator to bore the holes and convey the coal to the surface. This technique is presented in Chapter 6.4. In order to provide a brief overview of practice especially in coal mining, several case studies are presented with special attention paid to typical operations (McCarter, 1992).

## 6.1 Open Pit Mining

### 6.1.1 Surface Mining-Deferred Reclamation

Open pit mining is a term related to surface methods in which reclamation is postponed until all of the deposit is removed within economic limits. Deferring reclamation is introduced to ensure the access to work areas during the whole production life. If excavation advances to greater depths, backfilling the upper levels will either block the haulage or damage work areas at the base of the slope. In some countries, 'open pit mines' (usually metal) are distinguished from 'strip mines' (usually coal) and therefore, mining permits for some metallic deposits, and other minerals exclusive of coal may not require backfilling the pit. In these countries, such the pits may never be fully backfilled, and reclamation is put off until the economic limit has been reached. Overburden removed from an open pit is transported to an adjacent waste embankment or to a previously mined-out pit. Waste is not dumped at its place of origin. In some operations, waste is treated to recover values that are too low in grade to be processed in

standard milling. This process known as 'leaching,' is applied for copper, gold, and uranium wastes (McCarter, 1992).

In some open pit operation, the pits are large enough to be partially backfilled before the production stops, which may be done to decrease haulage and reclamation costs or simply to facilitate operating. This handling of waste, however, is not required as part of the mining process as in other methods. Mining procedures in which back filling is done voluntarily should be considered open pit rather than area mining or contour mining, in which excavations are filled and reclaimed simultaneously with mineral extraction (McCarter, 1992).

All surface mining of coal is considered strip (area) mining however, the method applied to steeply dipping coal seams is rather open pit mining. In these circumstances, excavation advances through one or more seams. The deepest coal seam is not accessible unless all overburden is removed. Therefore, the reclamation has to be deferred (McCarter, 1992).

Mountaintop removal is practised in the coal regions in mountains and can be classified as open pit mining in which overburden is transported to a waste embankment located outside the area of excavation. In such pits, it is not important whether or not reclamation of the uncovered surface is simultaneous with mining. The phrase 'open pit,' depicts depression below the natural ground surface. Mountaintop removal may be termed 'area mining' without concurrent reclamation, or 'open pit mining' without a pit (McCarter, 1992).

Open pits can be in the form of inverted, truncated, circular cones where the radius of each circular bench decreases with depth. Such the shape is typical of big disseminated deposits such as copper. In deposits that are stratiform, shallow, and large, open pits may be formed in the shape of footprints with steep sides and flat bottoms. Material can be removed in one of two ways termed sequential pushbacks or pushbacks. One or more pushbacks make a phase or intermediate plan. The sequential pushback pattern is presented in Fig.6.1. In this example, there are three zones A, B, and C, and one excavator (an electric shovel) is assigned to each zone. If zone C is ore and zones A and B are waste, it is necessary for cuts to be made through the areas B1 through B4 before ore can be mined in areas C5 through C8. Similarly, areas A1 through A4 must be removed before mining can move to areas B5 through B8. These sequences of activities have to be closely coordinated so that shovel A finishes its final level and begins to move up to the final cut in zone A before shovel B moves up to begin a new series of cuts in zone B. Proper scheduling makes it possible for waste stripping to be accomplished so as to prepare the 'mining room' needed to keep the required amount of ore production in the lower part of the pit (McCarter, 1992).
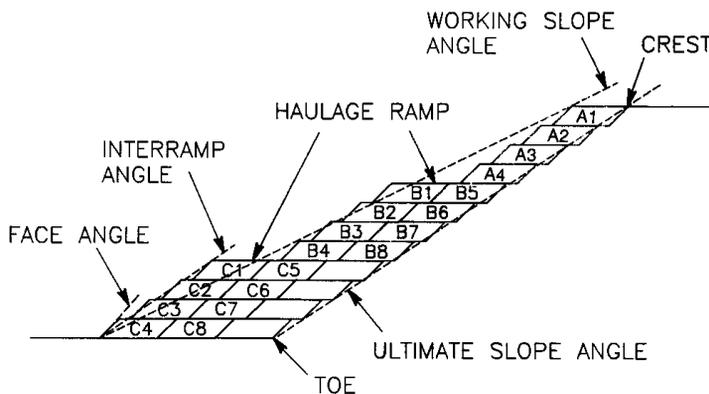


Fig.6.1. Cut sequence for sequential pushbacks (after McCarter, 1992)

Working slopes flatter than the ultimate have operational advantages (Fig.6.1). They enable one to form additional production faces in both ore and waste. Such the faces increase operational flexibility and make production more uniform. Additional production faces in ore also allow blending of ore types to optimise mill performance. Moreover, working slopes reduce the size and effects of major slope failure. In addition, an ultimate slope angle is chosen based on geotechnical analysis. Using a shallower working slope to reach the pit limits allows one to obtain more geotechnical data before an ultimate slope angle is committed. Selection of either too large or too small angle will be more dangerous for pits with a depth of 300 m than for pits with depths of 30 to 90 m. In the case of deep pits, if angles are too steep they may threaten the entire operation since they can cause massive instability. If angles are too shallow, the stripping may be costly and recovery of natural resources, very low (McCarter, 1992).

Conventional pushback. This mining pattern is more popular than the former and is applied in shallow pits and in larger mines. As presented in Fig.6.2 a pit sector is mined from the top down to the lowest ore level excavating one whole bench at a time. Scheduling is not as complicated as for the sequential pushback method. The larger plan area also allows more working faces within a given mine sector, which reduces overall costs. The backslope of a conventional pushback is cut to the ultimate slope angle. This means that the ultimate slope design is tested early. However, if slope instability appears, there may be temporary decrease in production. The major stages in development of an open pit mine embrace (1) evaluating three-dimensional distribution of mineralization and grade; (2) determining the economic limits for the pit; (3) choosing sites for waste embankments and soil stockpiles; (4) clearing vegetation; (5) locating processing, maintenance, office, and transport facilities outside the pit limits but close to the pit; (6) selecting equipment; laying out haulage roads; and (7) performing initial cuts. These initial cuts may be in a form of 'box cuts' or access roads on steep hillsides that are extended to build mine benches (McCarter, 1992).
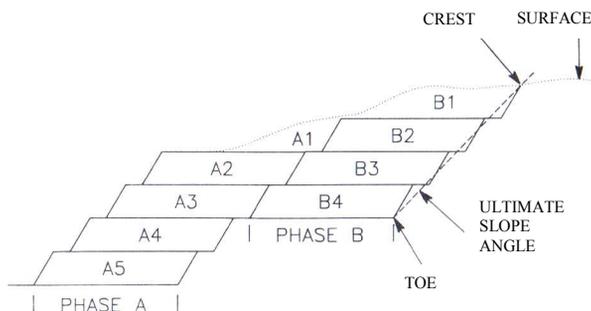


Fig.6.2. Cut sequence for conventional pushbacks (after McCarter, 1992)

Standard unit operations (drilling, blasting, excavating, and haulage) are involved in removing both waste and ore. The type of drills can be rotary and down-hole percussion rigs. If the material cannot be ripped, ANFO explosives are used is for dry blastholes and emulsion for wet ones. After breakage, material is excavated using electric shovels, backhoes, front-end loaders, scrapers, and, less frequently, draglines and bucket wheel excavators. The reach of the prime excavators to be used decides the height of a bench. The height should be within the maximum digging range so that the slope and the inclination toward caving of the face and highwall could be better controlled. The width of individual benches is determined by the convenient loading into the haul units, by stability of the pit wall, and by the break-even stripping ratio. Haulage employs trucks, scrapers, conveyors (in-pit crushers and conveyors

are also used), and less frequently with rail, skip hoist, and hydraulic transport (McCarter, 1992).

The following case study provides an example of mining techniques for coal open pit practice.

6.1.2 Steeply Dipping Bituminous Coal: Kemmerer Mine (Case Study)

6.1.2.1 Mine Description

The Kemmerer mine located in the United States produces high-quality subbituminous B coals. Previously, underground mining was applied at this property from 1881 until 1964. Underground production was replaced entirely by the surface operations, which began in 1951. In 1989, the 40 square kilometres consisted of a series of active and proposed pits that extended for about 11 kilometres. About two-thirds of the production was transported by conveyor to a mine-mouth power plant; the balance was shipped by train to smaller industrial clients. The mine used shovels and trucks to move 11.5 to 15.3 million bank $m^3$ of waste rock. There were three mining areas that included 1-UD started in 1971, 2-UD in 1983, and I-Area in 1988. Each of these areas was separated from other areas by either geologic features or conditions left by earlier mining activities (Whitman, 1992).

6.1.2.2 Deposit Description and Geology

Two coal-bearing formations exist in the immediate area (Fig.6.3). The first formation has three major seams ranging from 1.5 to 4.6 m thick and the second one has up to 30 different seams ranging from 0.6 to over 30 m thick. These two formations are separated by the formation, which is about 1500 m thick. The mine is situated within the second formation. The sediments strike generally north-south and dip to the west from 18 to 28°. Located approximately 1.6 km west of the mine, the major thrust fault displaced up to 4600 m of sediments and is one of several faults of this nature in the area. At the same time, the faults were active, and folding also took place. As a result, the second formation was placed into its present position (Whitman, 1992).
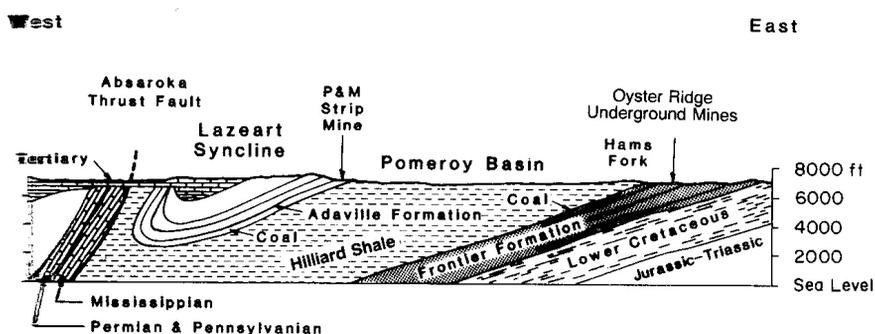


Fig.6.3. Geologic cross section (after Whitman, 1992)

6.1.2.3 Mine Development

The removal of the entire upper horizon of topsoil must precede mining and overburden disposal. The topsoil is stockpiled for later reapplication to reclaimed areas, or topsoil is

replaced at once if adjacent reclaimed sites are prepared. Each mining area includes a series of pits, and each pit has horseshoe-shaped cuts open to the east. The cuts extend vertically from the surface to the lowermost seam. Since most of the coal is deposited in the three lowest seams (i.e., No.1 to No.3), these seams must be available when developing less-productive parts of the deposit (Fig.6.4). Each cut must be scheduled carefully to achieve balanced coal production by combining high-ratio benches with low-ratio benches that cross the bottom three seams (Whitman, 1992).

1-UD Area. This area (Fig.6.4) is a special bituminous operation with total out-of-pit spoil and minimum in-pit reclamation. The entire pit was intended to be left open. However, an acquisition of the adjoining reserves resulted in using the original 1-UD pit for disposal. Production benches are at 10-m intervals with 15-m safety benches at 30-m vertical intervals. The overall pit wall is designed at 45°. The 1-UD area has nine coal seams, numbered in sequence from bottom to top. Their thickness varies, for example, from 26.5 m (No.1 Seam), 16.8 m (No.3 Seam) to 1.8 m (No.6 Seam). The total thickness of the nine seams is 75.9 m. There is wide variation in individual seam thickness along strike. In the south wingwall of 1-UD, there is local faulting. Overburden waste is disposed of entirely out-of-pit. The dump development plan includes the initial construction of a three-terrace outer ring with 15-m benches, variable vertical intervals, and 17° slopes. These maximum slopes are necessary for stability and to facilitate revegetation (Whitman, 1992).
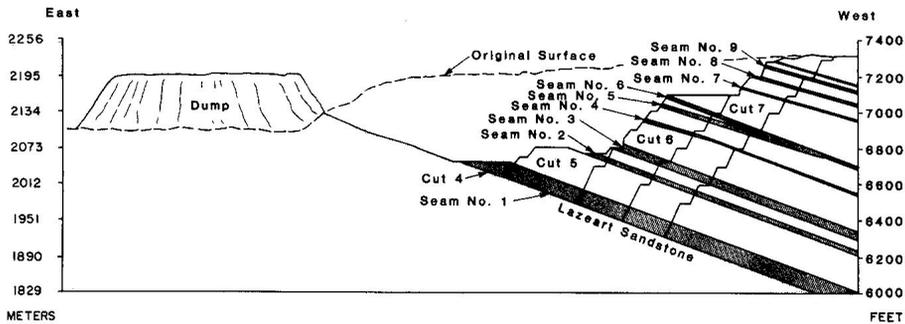


Fig.6.4. Typical cross section through the 1-UD pit (after Whitman, 1992)

Fig.6.5 shows a typical mining sequence in the south wingwall. The bench width changes from 107 to 122 m. A section line is drawn to divide the bench into halves. The inner half of the bench is used as the haulage road for coal and overburden exiting the area to the east. There are five coal seams to be excavated (there is not No.4 Seam in Fig.6.5). Since the seams dip to the west, all development must progress from west to east, which alternatively removes overburden and coal. The relative location of each seam determines the sequence in which the seam can be removed. Access ramps are limited to 8%. Therefore, sufficient horizontal distance is required between adjacent seams to drop the bench interval (10 m). Seams 6, 2, and 3 can be removed in this way. Because space is confined, the access to seams 5 and 1 has to be horizontal, as Fig.6.5 illustrates. After all seams have been removed, the ribs that remain are taken, moving from east to west. During this operation, the overburden haulage route is replaced to the outside of the bench and extended to the east pit limit. Finally, the inner half of the bench is removed in a similar manner. However, all development is horizontal, and work progresses from the outside inward (Whitman, 1992).

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6.1.2.4 Unit Operations

Drilling and Blasting. Four units, two electric and two diesel-powered drills, are used for blasthole drilling. The highly flexible diesel drills are necessary in limited areas and on the steep slopes connected with 20° dip of the formation. In dry overburden holes, heavy ANFO is used as the bottom load and standard ANFO as top load. Emulsions are used in wet holes. The density is adjusted depending on energy needs (Whitman, 1992).

Loading. Two 19.1-m$^3$ and one 11.5-m$^3$ electric shovels are the main overburden loading machines. Auxiliary equipment includes a 9.2-m$^3$ backhoe for use in confined areas adjacent to the highwalls, and a 9.2-m$^3$ front-end loader (FEL) employing an ejection-type bucket. Coal is loaded using two 16.8-m$^3$ FELs, also with ejection buckets (Whitman, 1992).

Haulage. Fifteen 154-t rear-dump trucks are employed to haul overburden. These units are used occasionally for coal haulage when there is a lack of coal trucks. Coal is hauled in six 109-t rear-dump trucks and one 136-t rear-dump truck. Coal is transported the whole distance to the tipples (2 to 8 km) by truck from the three active pits. The pit/tipple combination is dependent on the quality and availability of coal (Whitman, 1992).
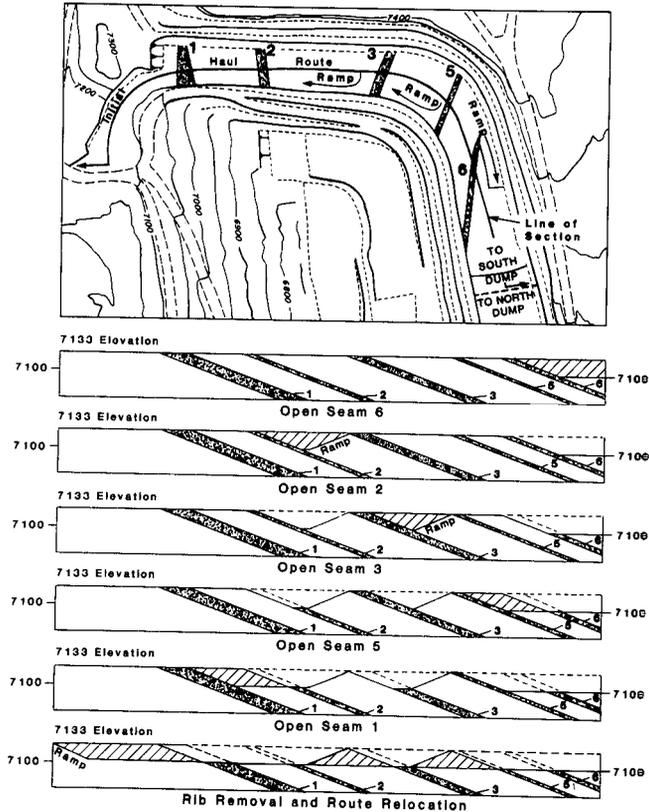


Fig.6.5. Bench development for the south wingwall, 1-UD pit (after Whitmann, 1992)

6.1.2.5 Ancillary Facilities

The shop is located near the north end of the area. It contains large equipment and light-fleet maintenance shops, as well as the warehouse. The large equipment shop includes bays. The steam cleaning bay, electrical shop, and unheated storage are separate and detached (Whitman, 1992).

6.2 Quarrying

Mining of dimension stone or aggregate is considered to be quarry mining. This term is commonly limited to production of dimension stone only. Quarries produce prismatic blocks of rock such as marble, granite, limestone, sandstone, slate, etc. Quarries have benches with vertical faces whose height may range from a few metres to 60 metres. The overall depth of a quarry may reach 300 m. Material is removed selectively using time-consuming and expensive methods for freeing the blocks from the surrounding rock (Hartman, 1987).

Quarrying involves four major operations: overburden removal and cutting, splitting, and handling stone (Fig.6.7). Overburden is loosened by ripping or blasting and then removed by dragline, scraper, or front-end loaders. Hydraulic monitors can be used to clean the deposit. An opening cut or slot is made across the quarry width after the surface has been cleared. This slot makes an additional free face which facilitates removal of subsequent blocks until, for example, the first lift, running across the whole width and length of the quarry, has been removed (Meade, 1992).

Each stone quarry is unique and requires its own unit operations. The upper surface and key slot give two free faces (the top and front of a prismatic block). Three additional faces are made to determine the ends and the back. These faces can be created by line drilling with pneumatic hammers mounted on a 'quarry bar.' Soft formations may be cut with wire saws, carbide tipped chain saws, or circular saw blades impregnated with diamond or other abrasive materials (Fig.6.8). Hard formations can be cut with diamond line saw (Fig.6.6). Hard formations, which contain minerals with contrasting thermal expansion properties, can be cut by means of a flame jet known as the Browning torch. This method is the best one for granite. Other formations can be cut with a high-pressure water jet or diamond line saw (Meade, 1986a).

When the block is freed on five sides, the bottom can be freed by wedging. Short holes may be drilled at the base of the block to insert 'feathers and plugs.' Hand hammering of these closely spaced wedges can produce a horizontal fracture (usually along a bedding plane) that detaches the block at the base. The wedges can be used to produce a fracture along the back of the block if there is bedding plane or joint at its bottom. The feathers and plugs are replaced with hydraulic wedges, air bags, or partially stemmed explosives for line splitting. Blocks are often tipped 90°, split along the bedding to obtain the desired size, and then removed from the quarry (McCarter, 1992).

Derricks are widely used for hoisting of blocks (Fig.6.8). Mobile equipment is being used more often because it is more versatile and flexible, which provides an increase in productivity. Large quarries have mill and finishing shop facilities adjacent to the mine (Meade, 1986b).

The following example of a quarry depicts a typical operation involving both quarry and finishing plant (Meade, 1992).

6.2.1 Dimension Stone: Brown Quarry (Case Study)

6.2.1.1 Mine Description

New cheaper quarrying methods contributed to reactivation of previously closed quarries in the Tennessee Marble District near Knoxville in the USA. The Tennessee marble quarries are located in a limestone known as the Holston Formation. This material is classified as marble because it can be cut into slabs and well polished. The marble is known for its pink colour and uniform texture. It was a desirable marble, but the depth of the quarry, limited development work, and high production costs resulted in abandonment in the 1950s. After about 30 years the quarry was reactivated. The reopening of this quarry in 1985 and subsequent purchase of MSI by one corporation reactivated the property. Since then, new equipment, overburden stripping, waste rock removal, and demand for the stone have contributed to the development of the quarry. Approximately 0.16 km$^2$ of the 0.21-km$^2$ of property cover acceptable marble. Along with the existing 3700-m$^2$ opening, three smaller openings were developed into a permanent working face of multiple benches accessible by rubber-tired loaders. There are places for block and slab inventory, finished material, storage, and a diamond gang-saw plant with an adjacent finishing shop. Operations are mainly removal of overburden and waste rock in the new development area. When this phase was complete, approximately 115,000 m$^3$ of material were removed (Meade, 1992).

6.2.1.2 Deposit Description and Geology

The marble deposit occurs in easterly dipping (10 to 15°) layers. The grey, shaley limestone underlies the marble. High-angle, east-west-trending joints allow for easy extraction of blocks. In some parts of the quarry, diagonally intersecting joints minimize the recovery of usable blocks. These areas are either avoided or removed and wasted. The strike of the marble layers extends across the property in a north-south direction. Overburden is variable but averages 2 m in thickness. A 6-m underlying zone of cap rock is wasted due to mechanical properties and colour. The commercial layer is 15 to 21 m in thickness. The operating additional quarry is about 24 m deep with working benches 9 to 12 m above the quarry floor (Meade, 1992).

6.2.1.3 Mine Development

The first years of operation consisted of predevelopment activity. During this time, the existing quarry was dewatered. The machines such as the derrick, hoist and compressor were reactivated. A series of stairs and ladders for the men and a boom-fall derrick for the equipment and supplies enabled one to access the working area. Old working benches were cleaning up with a small front-end-loader, which was removed with the derrick. Skid-mounted quarry bars with chain-feed air hammers were used for vertical drilling and broaching. Small pneumatic layer drills on horizontal quarry bars were used for break-line cuts. In new developed part, cap rock is removed by a combination of wire sawing, shooting, and wedging. The loosened rock is removed with the use of a dozer, excavator, and loader. The waste is dumped in a grout pile or used for ramps and roadways. The careful preparation and maintenance of all road and haulage ways has kept vehicle maintenance costs to a minimum (Meade, 1992).

6.2.1.4 Unit Operations

A crew of eight utilized three line drills for channel and line cutting, one penetrating drill for wire-saw pilot-hole development, and three wire saws. The crew of eight used a layer hammer for horizontal drilling, a loader, as well as a hydraulic jack system for raising and tipping blocks to produce what a crew of eighteen could produce with the older system. The restricted space in the quarry has made the retention of the horizontal floors and benches necessary. The initial key-way cut, made with the wire saw, is wide enough (2.4 m) to allow access for the layer hammer, if the horizontal break lines are needed. As soon as the key blocks are removed, additional strips are cut. Depending on the height of the strip, they are tipped into the keyway for splitting. Splitting can be performed in the vertical position (Fig. 6.7 and 6.8). The newly developed quarry area has benches of greater height (4.6 to 7.6 m), which allow tipping the strips of cut marble for splitting into required block sizes. Full use of the wire saws and mobile equipment allows for more efficient continuous cutting (Meade, 1992).
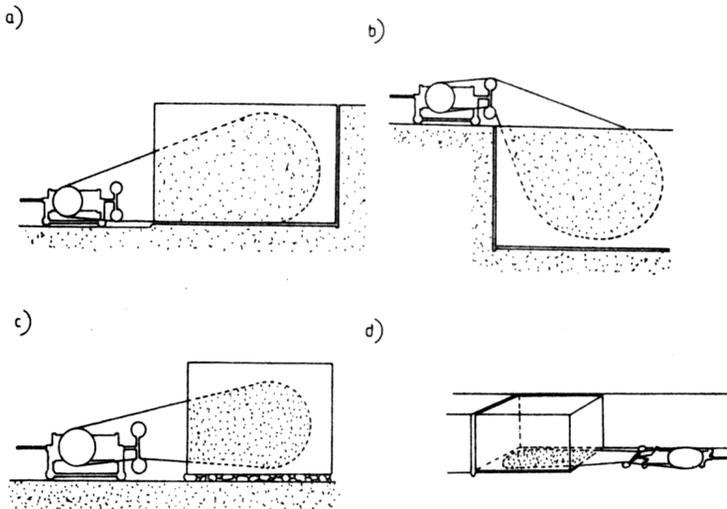


Fig.6.6. Cutting with a diamond line sawing machine
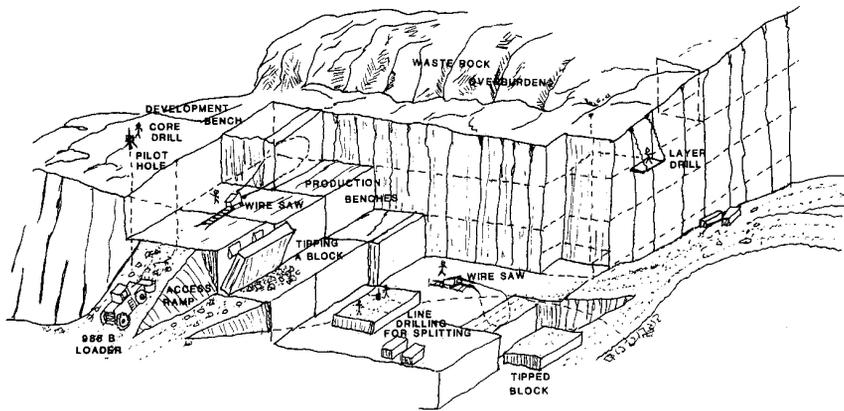
Fig.6.7. Schematic representation of quarry development (after Meade, 1992)
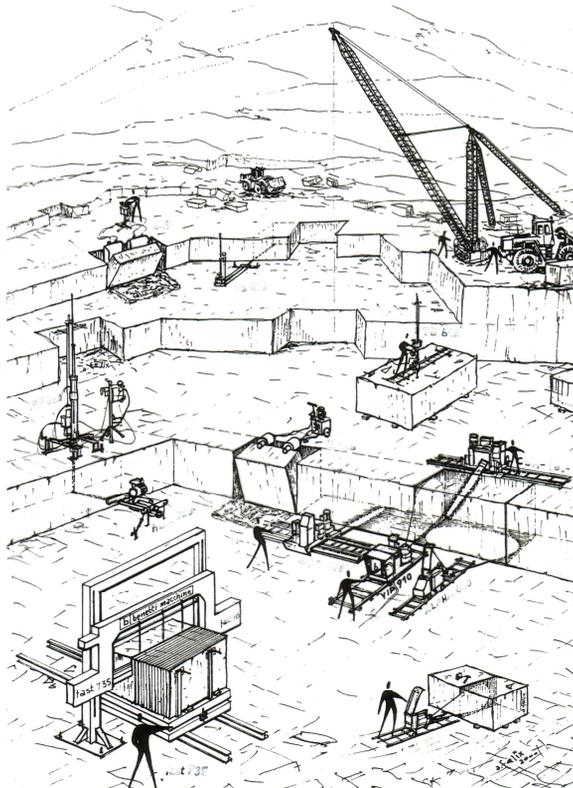


Fig.6.8. Marble opencast equipped with modern machines for breaking stone blocks

6.3 Open Cast (Strip) Mining

6.3.1 Surface Mining-Concurrent Reclamation

Surface mining in which reclamation is carried out simultaneously with extraction may be classified as either area mining or contour mining. Area mining is often called open cast or strip (furrow) mining. This type of surface mining is used on a large scale, which results in low unit cost, high productivity, high recovery, and greater safety than that of underground mining. Contour (or collar) mining, is applied to excavate a narrow zone following the outcrop of a coal seam in mountainous conditions. Haulback, box-cut, and block-cut mining are methods used in contour mining (Anon, 1979). The defining characteristic of both area and contour mining is that overburden, removed to make the mineral accessible, is immediately placed in the previously mined area (McCarter, 1992).

The main products of area mining are bituminous coal rank and lignite. Phosphate and oil sands are also mined in this way. Anthracite, bauxite, and bentonite are also mined by area methods but on a less scale (Hartman, 1987). Contour mining is used for removing thin coal beds (McCarter, 1992).

In area mining, large draglines, stripping shovels, or bucket wheel excavators (BWEs) remove overburden. These machines excavate and deposit material in one continuous operation. The same operation can be performed with the use of explosives (cast blasting). Soil and overburden is often removed using a combination of casting machines so that the mining area could reach the maximum width. Conventional excavation and haulage techniques (truck and shovels, scrapers, or dozers) are also used. They can also cooperate with one or more casting machines. Conventional excavation and haulage may be designed for topsoil and overburden (McCarter, 1992).

Area mining usually operates in parallel, deep trenches called furrows or strips that may be 1 km or more long (Fig.6.9). A deep trench is formed with material removed from the undisturbed area on the highwall side. This material is placed in the open pit at the angle of repose forming a spoil ridge. The width of the open pit may reach 23 to 46 m, the height of the highwall may be up to 61 m, and the height of the spoil ridge may be up to 76 m. Highwall angles may vary from 50° to vertical. Spoil-pile angles may change from 30 to 45° measured from the horizontal. Angles steeper than the angle of repose for most materials, which is 37°, can be acquired by compacting soil under its self-weight then excavating the material once again to an oversteepened condition. It has occurred that oversteepened spoil, can be highly durable (McCarter, 1992).

Haulback mining is applied in a series of almost square rectangular pit following the outcrop for at least several kilometres (Fig.6.10). The economic stripping ratio and steepness of topography determine the pit width. Since mining approximately follows a contour, the route of mining is sinuous with both outside and inside curves. Mining develops by first removing a box cut and hauling the overburden to a disposal site. Next, overburden is broken, excavated, and hauled by truck, scraper, or conveyor to fill previously mined-out pits and only little or no waste is deposited on the outslope. In this way, waste moves in a direction opposite to the direction of mining, which was given the name 'haulback' (McCarter, 1992).

Box-cut contour mining (Fig.6.11) is used in moderately sloping areas. It is completely the same method as area mining where there are only two or three seams. The strips are oriented parallel to the outcrop line and run uphill into the highwall. The first strip is a consecutive series of box cuts, which makes an open pit for spoil cast from the second strip. A dozer is used to clear vegetation over the box cuts and down-slope from the outcrop line. Overburden is then pushed from the box cuts to the outslope area. When the coal is uncovered, it is removed, and the dozer makes a level bench upslope from the box-cut line. A

dragline standing on this bench excavates the overburden and deposits it into the adjacent box cut. Instead of the dragline, a shovel can be used to remove overburden from the second cut thereby eliminating the need to form an upslope bench. When economic limit of mining is reached, dozers push the spoil uphill to cover the highwall before revegetation begins (McCarter, 1992).



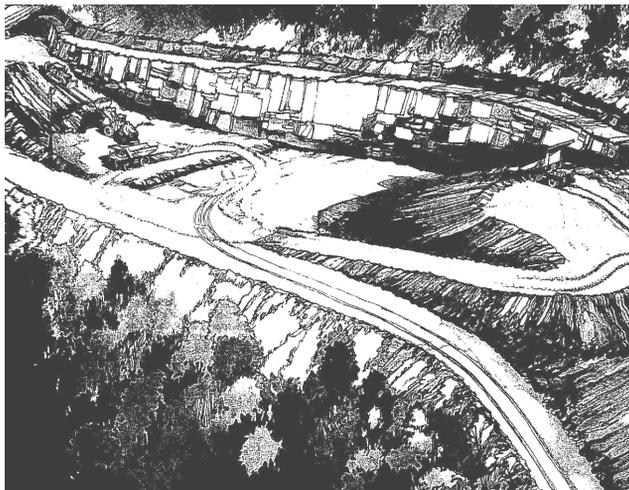Fig.6.9. Conventional area mining utilizing a dragline (after Anon, 1979)



Fig.6.10. Conventional haulback mining employing trucks, dozers, and front-end-loaders (after Anon, 1979)

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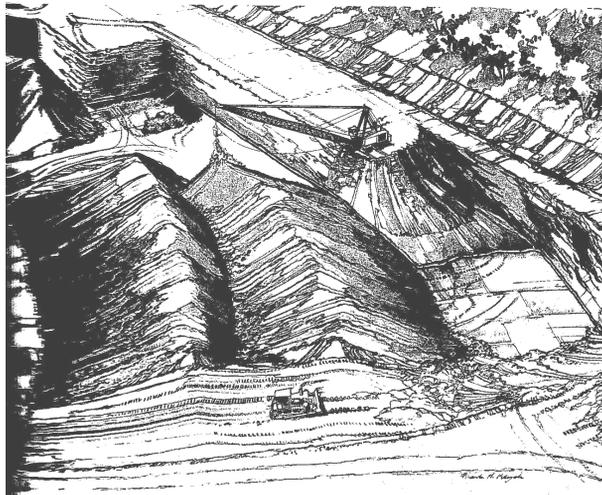
Fig.6.11. Box-cut contour mining utilizing dozers and dragline (after Anon, 1979)

Block-cut mining can be applied to either area or contour mining. In this method, blocks of overburden wider than 20 m are removed as a unit and deposited in mined-out areas either perpendicular to the direction of advance (area mining) or along the direction of outcrop (contour mining), which differs block-cut mining from conventional area and contour mining. The overburden can be removed by front-end loader (FEL) or pushed by dozer to the disposal site in one operation provided that the mineral deposit is shallow and has relatively uniform cover (McCarter, 1992).

If a coal seam is nearly flat and lies not far from the mountain's top, the deposit can be mined from outcrop to outcrop in sequences of strips as performed in conventional area mining. In some conditions, the spoil is not contoured but levelled to follow the original topography. This technique is called mountaintop removal (Anon, 1979), which in the previous chapter, was referred to as a 'deferred reclamation' method since overburden was hauled to a disposal site located out of the mining area. In the above case, overburden is deposited in the open pit located next to the seam being uncovered and reclaimed as mining advances (McCarter, 1992).

The sequence of operations consists of clearing vegetation, soil removal, drilling and blasting overburden, stripping, removal of coal or other minerals, and reclamation. In area and contour mining, reclamation is repetitive and simultaneous with mining therefore it is a unit operation. Clearing vegetation can be done with track-mounted dozers. Soil is removed by dozers, but more often with scrapers and dozers, or trucks and FELs. If overburden is weak, it can be broken utilizing dozers equipped with rippers. If the overburden rock is more competent, it should be drilled and shot. Drilling may be done with drag bits, rotary bits, or percussion. Holes are usually vertical. However, inclined holes are used to improve highwall stability and to obtain uniform fragmentation when the highwall is not vertical. ANFO explosives are used if conditions are not wet (McCarter, 1992).

The properties of the overburden and mineral rock as well as the deposit geometry, determine the choice of a suitable excavator. Stripping shovels can achieve maximum digging effort and are the best choice for blocky material. They also have a shorter cycle time compared to draglines. However, it is necessary for them to operate on top of the seam to be mined. If this material is friable, moving excavator creates fines, which may contribute to poor recovery. Moreover, shovels must reach from the top of the seam to the top of the spoil

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pile whereas draglines reach from the original ground surface to the top of the spoil pile. A dragline is effective while excavating well-broken and disaggregated material. The dragline is more flexible than a stripping shovel, has greater reach, and can operate on both the highwall and the spoil side. In addition, it can work better in adverse pit conditions such as slides and flooding that would eliminate a shovel. It can also dig a deeper box cut and can dig both above and below its position on the overburden (McCarter, 1992).

Bucket wheel excavators work effectively in material that is well fragmented and without large boulders, especially in hard pottery clays, phosphates, tar sands, and bauxite. Blasting is often unnecessary for them to handle these materials. The BWEs are able to produce more stable pit slopes, more uniform spoil piles, and to control well selective mining of interbedded deposits. They can operate both above and below the base of the machine. However, their capital cost is higher and their manoeuvrability is lower (McCarter, 1992).

Overburden 'bank' volume and the equivalent volume of broken waste determine the selection of a casting machine (or combination of casting machines). It is essential for planning to ensure that a spoil pile of given width, height, and slope angle can take the volume produced by a highwall excavation of given width and depth of cover. After the mineral has been exposed by stripping, loading of coal or other materials may be done immediately with BWEs, FELs, electric shovels, hydraulic shovels, or backhoes. Stronger material may need ripping or blasting for successful loading (McCarter, 1992).

The application of surface mining with concurrent reclamation can be classified on the basis of the deposit geometry and type. Area mining can be applied to deposits where (1) the overburden is thick and uniform down to a single, nearly-horizontal coal seams separated by thin parting; (2) the coal seam is thick and nearly horizontal and the overburden is shallow; (3) numerous nearly-horizontal coal seams spread in different directions between the layers of overburden; and (4) the overburden has variable thickness and covers coal seams which are dipping. Contour mining is used where almost horizontal, thin seams are exposed in mountainous areas. Area mining can also be applied to lignite, tar sands, and phosphate (McCarter, 1992). Several examples of coal and lignite mining are presented in Chapters 6.3.2 through 6.3.6.

6.3.2 Monolithic Overburden, Horizontal Coal Seams: Jim Bridger Mine (Case Study)

6.3.2.1 Mine Description

The Jim Bridger mine is located in Sweetwater Country near Rock Springs in United States. The mine produces subbituminous coal and has an area of 81 km$^2$. Coal is utilized on site by the power plant. Annual production averaged 4.5 to 6.4 Mt of run-of-mine coal. The prime operations are dragline-stripping operations. The mine employs four draglines and one shovel for casting operations and two front-end loaders (FELs), one shovel, and one hydraulic backhoe for five to six active production faces (Bricker, 1992).

6.3.2.2 Deposit Description and Geology

The mine is located on the uplift, a broad, asymmetrical anticline about 97 km long and 56 km wide with a north-running axis. Normal faults with less than 30-m vertical displacement cut through the uplift. The dip of strata on the flanks varies from 3 to 15°. The coal seams occur in the formation which is 460 m thick and Paleocene age. There are five coal seams exposed in the mine; they are designated D5 through D1 from top to bottom. There are single-seam, two-seam, three-seam, and four-seam areas. The seams dip 2 to 5° to the northeast.

Overburden and interburden materials consist of sandstones, siltstones, claystones, shale, and limestone stringers. The area of the deposit is about 75 km$^2$ (Bricker, 1992).

### 6.3.2.3 Mine Development

Stripping operations start after soil has been removed from areas. In the early years of the mine, soil was placed in stockpiles at various locations, which was not acceptable because the use of stockpiled soil requires fertilization. Whereas soil applied directly highly decreases fertilization costs. Direct-applied soil is hauled from the highwall across or around the pit and placed on reclaimed areas. A buffer is kept from the highwall for operational flexibility. When soil is removed, the overburden is drilled and blasted. Outcrop development is performed by a scraper or truck and shovel fleet. The outcrop overburden material is placed to lessen out-of-pit spoil. After outcrop has been developed, the draglines remove overburden by simple side casting or extended bench methods up to about 24 m in depth. A multiple-pass, spoil-side, dragline stripping method has to be used for depths greater than 24 m. Highwall angles vary from 50 to 70° and the angle of repose for the spoil material is 36° (Bricker, 1992).

The pit develops from northwest to southeast following the strike of the coal beds. Each successive pit is further down the dip of the coal seams. Pit widths vary greatly throughout the mine; outcrop pits are the widest (61 to 76 m). Where the overburden depth is less than 46 m, the pit width varies from 46 to 61 m. When the overburden is deeper than 46 m, the pit width ranges from 37 to 46 m. This means that the deeper the overburden is, the narrower the pit, and overall average pit width is 46 m. The active pit is about 15 km long. There are four draglines, a truck fleet and shovel operation, and a scraper fleet operation, which allow six faces to be operative at any time. Ramps or entries spaced at roughly 1200-m intervals make the pit access available. The mine is located to the northeast of the power plant, therefore the ramps start at the pit floor and run across the spoil side of the pit up a 3 to 5% grade to a haul road system ending at the power plant truck dump. As the southern expansion is started, final reclamation begins in the currently active area. This development concept minimizes the total area disturbed and allows parts of the mine to go back to its original condition and land use as soon as possible (Bricker, 1992).

### 6.3.2.4 Unit Operations

Drilling and Blasting. Six different makes and models of drills are used for drilling. Blast patterns are rectangular or square and vary with depth and type of material. An emulsion-product mix is used, which is a blend of 33% emulsion and 76% ANFO. In wet areas, the product mix is 50% emulsion and 50% ANFO. Powder factors average between 0.36 to 0.47 kg/m$^3$ for overburden, 0.24 kg/m$^3$ for coal, and 0.47 kg/m$^3$ for parting (Bricker, 1992).

Overburden Removal. After outcrop development, draglines remove overburden by simple side casting (Fig.6.12) or extended bench methods (Fig.6.13) up to 24 m. Depths greater than 24 m are obtained with a multiple-pass, spoil-side, stripping method (Fig.6.14). The overburden is split into two lifts of equal depth and the first (upper) lift is drilled and stripped by simply side-casting the material into the empty pit. Next the dragline moves from the highwall elevation down a ramp (cut out of the upper lift) to the lower lift elevation. The lower lift material is drilled and blasted before the dragline cuts the ramp. The dragline strips a key cut that takes the lower part of the highwall and places that material behind the upper lift spoil. Finally, the entire upper lift and lower lift key-cut spoils are levelled, and the dragline moves onto the spoil-side pad. From this position, the dragline strips the remaining lower lift material and spoils the material in its ultimate place (Bricker, 1992).
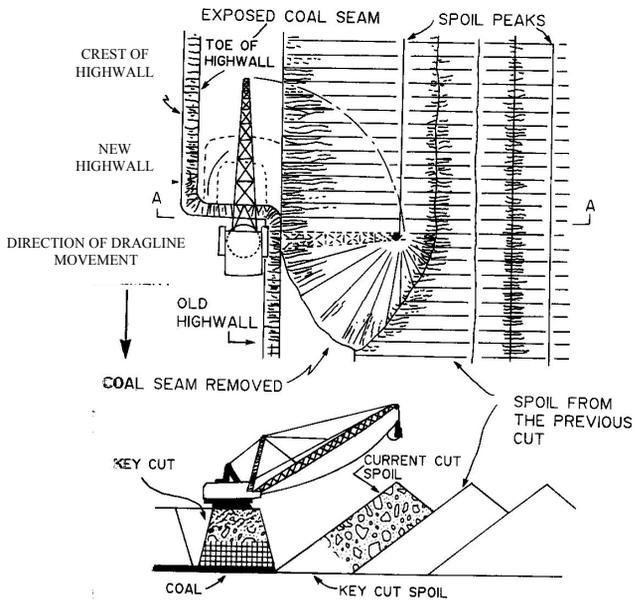
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Fig.6.12. Simple side casting (after Bricker, 1992)
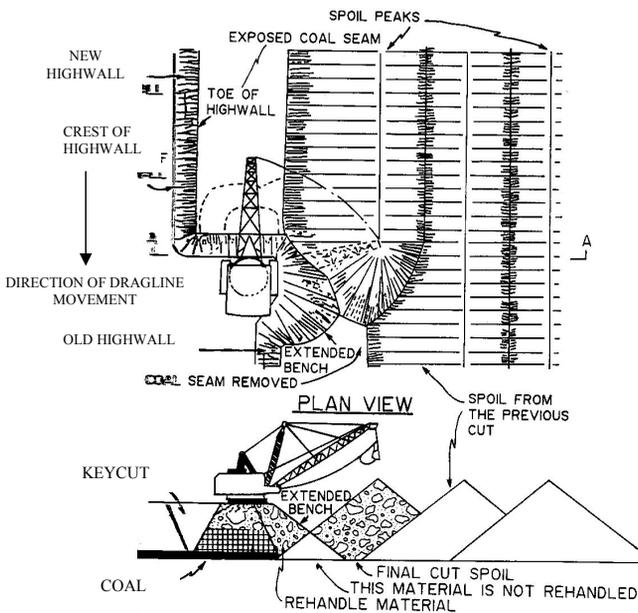


Fig.6.13. Extended bench stripping (after Bricker, 1992)
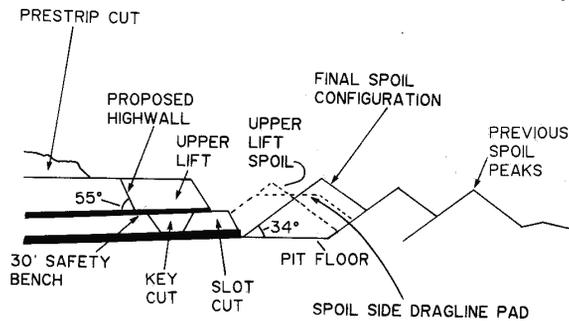
83

Fig.6.14. Multiple-pass, spoil-side stripping (after Bricker, 1992)

The excavation of multiple-seam deposit needs supplemental stripping operations to remove the interburden materials. The dragline can remove interburden materials that are equal to overburden materials, which can be accomplished in the multiple-pass dragline operation described above. Supplemental stripping operations involve a truck-and-shovel fleet or a scraper fleet. (Trucks and scrapers may be of a caterpillar type). These operations provide added flexibility to the mine by selectively handling toxic materials, performing advance benching of the highwall for the dragline, or removing interburden materials in multiple-seam areas (Bricker, 1992).

Coal Production. When the overburden, and/or interburden, has been removed and the top surface of the coal seam uncovered, the coal surface is cleaned by rubber-tired dozers, blades, or scrapers. The coal is then drilled using twin-mast auger drills with 133-mm bits. The material is lightly blasted to ease loading operations. Coal is loaded with FELs, electric shovels, or hydraulic backhoes (Bricker, 1992).

Haulage. In early years of the mine coal was hauled 17.7 km from the mine to the power plant by a fleet of thirteen 109-t bottom-dump trucks. Such the long haul made the operator install a conveyor system to lessen the distance for truck haulage. Five 82-t end-dump trucks replaced the bottom-dump trucks. These new trucks transport coal in 6.4 km round trip from the pits to two truck dump sites feeding the conveyor system. Each truck dump site uses a feeder-breaker to size the coal before the belt is loaded. Coal is discharged at the power plant's transfer building (Bricker, 1992).

Reclamation. Reclamation involves spoil pile levelling and grading, soil application, fertilisation, and mulching. Spoil-pile levelling is started with dozers and completed by the scraper fleet. The regarded area is ripped deeply with a dozer before soil is applied. In order to meet approximately original contour (AOC) criteria, spoil material must be moved several hundred metres using the scraper fleet. When an area has been regarded, soil is applied directly from the highwall or from stockpiles. Soil is applied to regarded areas at an expected thickness. Before soil application, a survey of elevations and results of chemical analyses of samples from the regarded area are submitted for evaluation. After soil has been placed, the area is ripped with a blade or a chisel plough along the contour, and seed mixes are drill-seeded. Finally, most areas are mulched to enhance protection from erosion (Bricker, 1992).

6.3.2.5 Maintenance Facilities

A large diesel shop and small gas vehicle shop are used for maintenance onsite. These facilities are located next to the power plant and main mine office. The diesel shop consists of repair bays, one wash bay, a warehouse, welding bays for dragline buckets, and offices. Additional facilities are located closer to the pit for large shovels and drills (Bricker, 1992).

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6.3.3 Monolithic Overburden, Thick Horizontal Coal Seams: Jacobs Ranch Mine (Case Study)

6.3.3.1 Mine Description

Jacobs Ranch mine is a two-pit operation located about 84 km southeast of Gillette Wyoming, and 18 km east of Wright, Wyoming in the USA. It is one of 15 coal mines operating in 1989 in this part of Wyoming. The mines in this Basin have thick coal seams and thin overburden. This mine employs the truck-shovel method for overburden stripping and coal mining. Two pits are mined so that coal can be blended to meet coal quality needs of customers (Fig.6.20). Construction began at Jacobs Ranch mine in 1975. The first shovel was erected in 1976 and then overburden stripping began. Annually (1989), the mine produced 13 Mt of subbituminous, low-sulphur coal that is used to generate electricity in Texas, Louisiana, Arkansas, and Oklahoma (Anderson and Kirk, 1992).

6.3.3.2 Deposit Description and Geology

The deposit is located in a basin in the form of syncline of 31,000-km$^2$. The sediments that fill the basin were deposited over a 500-m.y. period, but the structure itself is a result of folding that began about 70 million years ago. At that time, the land folded forming rivers, flood plains, and swamps. Streams coming from the highlands carried sand and mud into the basin, where they were deposited in channels and on flood plains. The swampy areas between the channels caused a dense growth of grasses. The remains of these plants make the thickest coal deposits in the world. The deposit at this mine consists of recoverable subbituminous coal. The 20-km$^2$ coal deposit is located on the 60.7-km$^2$ mining area. The mined coal seam is horizontal and continuous within the lease except near the eastern and southern limits. The coalbed consists of up to three separate coal seams referred to as Upper, Middle, and Lower seam. In the southwest part of the lease the three seams form a coal unit 15-18 m thick (Fig.6.15). The Upper coal is the uppermost coal being excavated. It is a separate unit throughout the lease, except where it merges with the Middle seam. The Upper seam ranges from 0 to 2.2 m in thickness. Split A, a carbonaceous shale parting 0 to 12 m thick, separates the Middle seam from the Upper seam. The Middle coal, approximately 12 to 17 m thick, is below Split A. The Lower coal seam ranges in thickness from 0 to 2.6 m over the lease. It merges with the other two coal seams in the southwest and in the northwest a shaley parting separates it from the Middle seam. This parting, Split B, is of thickness averaging between 1.5 to 3.0 m thick over most of the lease. Almost entire coal lease is rolling upland. The rest of the lease area is either shale slope or alluvial terrace. The overburden-to-coal ratio varies throughout the basin. Overburden is thinner in the northern part of the basin but stripping ratios are different in each mine and within individual deposits (Anderson and Kirk, 1992).
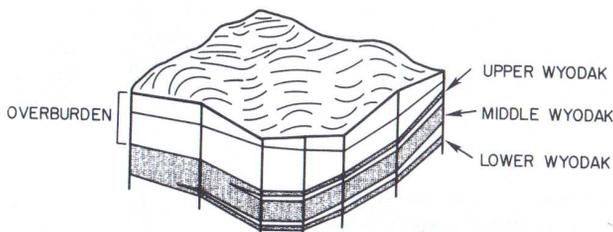


Fig.6.15.Geologic cross section of exemplary mine (after Anderson and Kirk, 1992)

### 6.3.3.3 Mine Development

Topsoil must be categorised, removed, stockpiled, and planted so that the land can be successfully reclaimed after mining. To categorise the topsoil types and quantities augering is performed. An environmental technician analyses the topsoil and defines the depth to which it should be removed. Scrapers then remove the topsoil and transport it to stockpiles or place the topsoil directly on re-contoured overburden surfaces. If the topsoil is placed in a stockpile, it is seeded with a quick growing to hinder wind and water erosion (Anderson and Kirk, 1992).

The initial stripping at this mine was made in an area of low-overburden depth. The 'box-cut' material was placed in permanent overburden piles next to the initial openings. The pit and backfill design must be made concurrently. Optimal pit design minimizes haul distance, while regarding the after-mining contours. The overburden highwall stands at an angle of 1:1, horizontal to vertical. The backfill angle of repose is 1.5:1. The three-seam coal deposit is of a complex nature therefore the pit is of 'star-stepped' geometries in both lateral directions from the pit floor. The coal thickness averages between 15 to 18 m. The pit width averages roughly 610 m. The overall disturbed length varies between 730 to 1200 m along the axis of progress. Overburden is hauled on roads built into overburden benches (Fig.6.16) whereas coal is hauled to the preparation plant on semi-permanent haul roads (Anderson and Kirk, 1992).
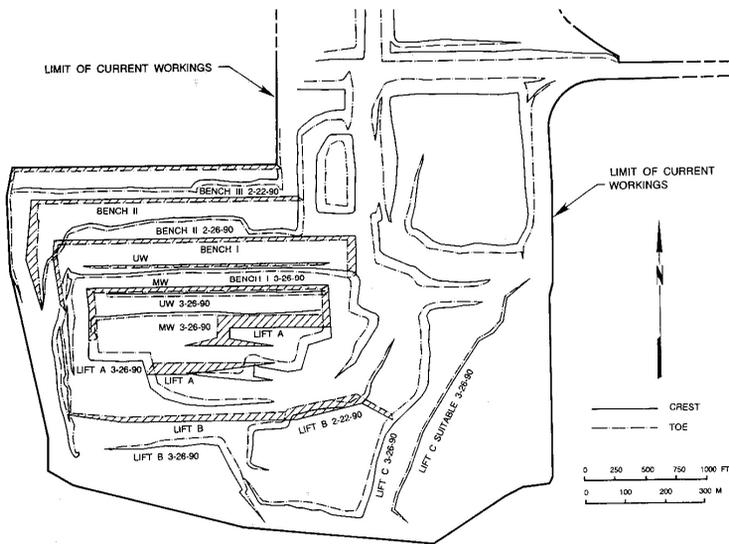


Fig.6.16. Monthly mine plan (after Anderson and Kirk, 1992)

### 6.3.3.4 Unit Operations

Drilling and Blasting. The coal and overburden is drilled and blasted so that high truck and shovel productivities and low operating costs could be kept. Drillholes are spaced on approximate 15-m centres and are loaded with ANFO explosives if the holes are dry, and ANFO-based slurries if the holes are wet (Anderson and Kirk, 1992).

Loading. The overburden is loaded with stripping shovels that have bucket capacities up to 27.5 $m^3$. Moreover, front-end loaders (FELs) remove waste on an as-needed basis. The diesel/electric loaders can load 11.5 $m^3$ per pass. The stripping equipment is used on an

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around-the-clock basis. The thick coal seams and homogenous overburden contribute to high productivities and economies of scale (Anderson and Kirk, 1992).

Coal mining is conducted with coal shovels. One coal shovel is assigned to each of the two pits. Bucket capacities for the coal shovels are up to 31 m$^3$. To blend the coal properly the output from each of these shovels should be regulated on a shift-by-shift basis. As well as the shovels and FELs, the mine uses a continuous loader. The loader has a rotating drum that cuts coal in horizontal slices. The coal is discharged directly into trucks by a movable conveyor belt. Support equipment embraces rubber-tired dozers, crawler tractors, road grades, and water trucks. Overburden and coal shovels operate simultaneously (Anderson and Kirk, 1992).

Haulage. All material is transported with trucks. 154-t end-dump trucks with a capacity of 80 m$^3$ are used to haul overburden. The mine is also using 218-t trucks. Mainly 154-t bottom-dump trucks haul coal (Anderson and Kirk, 1992).

Reclamation. At this mine reclamation is continuous. The mining process resembles a 'moving hole,' because overburden removed ahead of mining is placed behind the advancing pit. When the backfilled overburden is contoured to match the adjacent land, topsoil is spread on the site. The ground is planted with grasses and shrubs. Fences protect the lands until the grasses are established. In addition, natural rock conditions are created for small animals and birds. Before completing reclamation, ditches and ponds are formed to prevent run-off water from eroding topsoil (Anderson and Kirk, 1992).

### 6.3.3.5 Ancillary Facilities

Maintenance. The mine has two teams of diesel mechanics, welders, and maintenance electricians. The maintenance shop is large enough for tall haul trucks. Equipment too large and slow to be brought to the shop is serviced in the field. This equipment includes bulldozers, drills, and shovels (Anderson and Kirk, 1992).

### 6.3.4 Monolithic Overburden, Inclined Coal Seams: Trapper Mine (Case Study)

### 6.3.4.1 Mine Description

The Trapper Mine is located 9.7 km south of Craig in the northwest corner of Colorado, in the United States. The mine has a property of 39 km$^2$ and gives an annual coal production of 1.9 Mt. Trapper ranks as the third largest producing coal mine in the state in 1991 year. Three electric draglines strip waste material from five dipping coal seams. (Stubblefield and Fish, 1992).

### 6.3.4.2 Deposit Description and Geology

The mine is situated in a geologic setting of gently rolling, east-west-striking anticlines and synclines. The mine is on the southern the big syncline where the coal dips at 16% to the north, about 2% steeper than the surface of the land. There are few faults on the property. Overburden ranges from 0 to 43 m. Fig.6.17 presents a cross section of the mining area (Stubblefield and Fish, 1992).
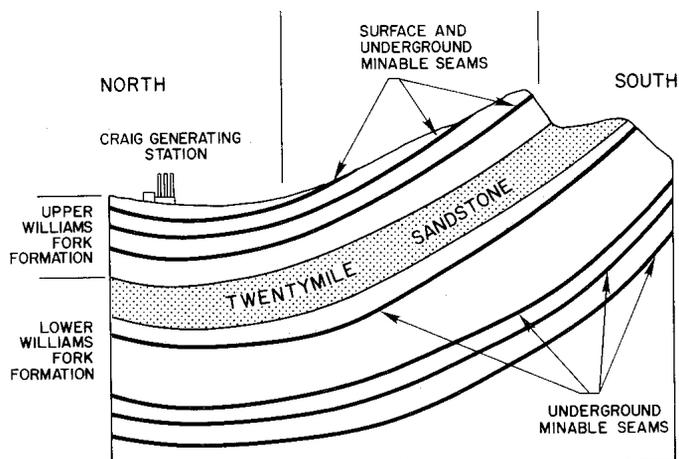
Fig.6.17. Typical Trapper mine cross section (after Stubblefield and Fish, 1992)

Seven coal seams that can be mined with surface mining methods occur within the permit limit. Five of these seams are mined from three active pits. Coal is of high-volatile subbituminous rank and is dedicated to steam coal application. The coal seams thickness ranges from 1.0 to 3.5 m. Waste material (overburden and interburden) consists of interbedded shale, siltstone, and sandstone layers. The waste is soft (Stubblefield and Fish, 1992).

### 6.3.4.3 Mine Development

Topsoil Removal. Topsoil removal operations begin the mining cycle in summer. Firstly, caterpillar (Cat) dozers windrow surface vegetation onto an adjacent area where the topsoil has been removed in advance. Next, six to eight Cat scrapers remove topsoil (range is 152 to 610 mm) from an area large enough to allow mining to perform until the following summer. The topsoil is loaded and then it is either hauled directly to areas previously mined and recontoured or stockpiled for later deposition. Topsoil is transported directly using either across-pit ramps or haulage routes around the end of the pits. (Stubblefield and Fish, 1992).

Layout and Access to Pits. The mine has four active pits: A, B, C, and D. Each extracts two major coal seams. As the C pit progresses to the west, it is mined together with the B pit. An E pit was a single-seam operation that employed cast blasting. Operations at E were moved to C in order to recover coal before an adjacent underground longwall mined below the reserves. 18-m wide main haul roads access the pits. Ramps into the pit are placed about 300 m apart and may run through the spoil or highwall, depending on which side the main haul road accesses the pit (Stubblefield and Fish, 1992).

Overburden Removal. At the mine, the seams are dipping and the topography is sloping, therefore the overburden can be removed with a dragline using three possible techniques strikeline, dipline, and oblique stripping. Fig.6.18 presents the layout of the strikeline and dipline methods. Oblique stripping is a combination of the first two methods where the pits are located between the strike and dip of a seam (Stubblefield and Fish, 1992).
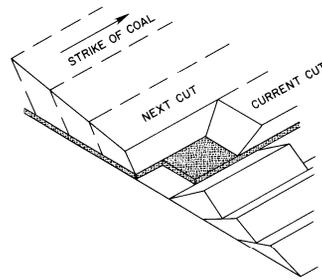
**STRIKELINE STRIPPING METHOD**
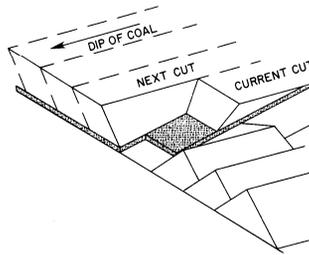


**DIPLINE STRIPPING METHOD**

Fig.6.18. Stripping methods for dipping seams (after Stubblefield and Fish, 1992)

The strikeline method allows the dragline to advance the cut along the strike of the coal. Strikeline pits can be laid out either starting at the outcrop or at the point of maximum overburden depth. The latter choice involves the highest initial stripping costs, due to maximum cover at the beginning, highwall instability, and poor utilization of equipment in later years. Starting at the outcrop does not have high initial production costs. However, as the pits become deeper, smaller areas are available to spoil the increasing amounts of waste. In this situation, called being spoil-bound, waste rehandling is substantial and costly (Stubblefield and Fish, 1992).

Oblique stripping is defined as aligning the pits at an angle to the strike of the coal. The technique is the same as the dipline method when the orientation of the pit reaches the dip direction. When the orientation approaches the strike direction, it is the same as the strikeline method (Stubblefield and Fish, 1992).

The exemplary mine chose the dipline method due to the greater highwall stability and lower rehandle quantities compared to the strikeline method. To make this stripping plan work, the mine had to develop an efficient method of moving the draglines up and down the dip slope (Stubblefield and Fish, 1992).

Dragline Movement. There are two procedures available for the dragline moving along the pit length: a switchback ramping system and a benching or 'see-saw' method. In the switchback method, the dragline digs a ramp to an intermediate bench about 45° away from the pit and then makes a switchback to the next bench above or below the original bench. This

method requires considerable walk-road construction and rehandle. Whereas the "see-saw' method can reduce the rehandle and give the dragline a longer digging time on the same bench. The sequence for the 'see-saw' method moving uphill on the spoil begins with the dragline at position 1 on pad 1. Using interburden material from the pit, the dragline builds a bench extension, area A, and moves onto this extension to position 2. While stripping from this position, the dragline spoils waste into area B (situated at the higher point than position 2) and a dozer builds an 8% ramp and another bench (pad 2, position 3) 6 m higher than pad 1. When complete, the dragline walks up the ramp to pad 2 at position 3 and begins the sequence again. The reverse procedure is used for moving downhill (Stubblefield and Fish, 1992).

Stripping Sequence. The mine has developed a method that allows the draglines to remove overburden from the highwall as they move downhill and to extract interburden as they dig back uphill on the spoil side. The dragline begins at the top of the hill near the outcrop and digs overburden to the bottom seam on the highwall side. As the dragline moves down the hill, it meets the upper seam outcrop and switches to removing overburden above the upper seam until it reaches maximum economic cover at the end of the pit. Using an in-pit spoil bridge, the dragline crosses over to the spoil side of the pit and waits until the upper seam is mined out and the interburden to the lower seam is drilled and blasted. Once blasted, the interburden is removed by the dragline, exposing the lower seam as it continues back up the hill using the 'see-saw' method. As the dragline reaches the point where the upper seam originally out-cropped, it bridges back over to the highwall side to begin the sequence again. Thus the dragline strips down and then back up the hill to complete one cut (Stubblefield and Fish, 1992).

Pit Dimensions. The coal dips more steeply than the topography, therefore, the maximum strip limit of 43 m is reached quickly at the bottom of the hill. This situation requires the pits to be short, from 240 to 1800 m. Since scheduling is difficult in short pits 53-m pit widths are designed on purpose to keep the dragline on the same working bench as long as possible before it moves up or down to the next digging position. Highwall angles vary from 26 to 63°, depending on overburden competency and the highwall conditions. An average slope of 37° for the spoils ensures satisfactory stability for the broken material (Stubblefield and Fish, 1992).

6.3.4.4 Unit Operations

Drilling. Waste materials are prepared by three rotary drills using 270-mm tricone bits. The drills are positioned on a series of level benches cut into the hill slope. Patterns for blastholes vary depending on the type of material but usually average 6.1 by 9.1 m for overburden and 6.7 to 7.6 m for interburden. A twin-masted auger drill is used for drilling 114-mm blastholes in coal. Coal patterns are drilled on a 3.7- by 4.6-m grid (Stubblefield and Fish, 1992).

Blasting. A mixture of ammonium nitrate and 6% fuel oil (ANFO) is used almost exclusively for overburden and interburden blasts. The gas-generating characteristics of ANFO provide better digging conditions relative to the more expensive, high-velocity slurry explosives. Bagged emulsion and bulk slurries are used occasionally when blastholes are wet. However, when only small quantities of water are encountered in the blastholes or when the shot will proceed in a short period of time, a plastic hole liner is used to sleeve the bulk ANFO. Blasting operations use the system with down-hole and surface delays for blast initiation and sequencing. All waste materials are shot (Stubblefield and Fish, 1992).

Overburden Removal. Surface mining (open cast or strip mining) is the only mining method used at the mine. Three long-range draglines remove overburden to expose the coal.

Arched 23-m$^3$ buckets are used in areas of dense and poorly fractured overburden and where chopping is required. This bucket is well suited to these goals, due to its heavier construction. Archless buckets with a capacity of 24.5 m$^3$ have been used in overburden that is lighter and well fractured. The long-range draglines applied in the mine can move on 8% uphill and 10% or greater downhill grades, which is important for the steep slopes at this mine. Computerized monitoring and control system is installed on each machine. The goal of these systems is to assist the operator to improve performance and reduce costs through better preventive maintenance. Radio links are used to transmit data from each dragline to the engineering department. Scrapers and dozers remove thin horizons of separable waste material between seam splits. Dozers rip these parting layers and assist by push-loading the scrapers. Scrapers are needed because they allow greater recovery of the top of the coal seam instead of dozers that can break the coal surface (Stubblefield and Fish, 1992).

Coal Loading. Coal loading operations are performed by three hydraulic backhoes with 11.5-m$^3$ buckets. One backhoe is stationed at each pit and loads from atop the coal seam. The crawler-mounted backhoes were selected over rubber-tired loaders because of the backhoe's excellent performance in a pitching environment and possible wet floor conditions. Loading is performed from 7.6-m wide slots directed down dip to provide coal trucks a path following the dip of the coal (Stubblefield and Fish, 1992).

Coal Haulage. Coal from the mine is delivered under a long-term fuel supply contract to a steam generating plant located adjacent to the mine. Six trucks from a fleet of 45-t end-dump coal haulers are employed to haul the coal an average of 4 km one way to a crusher that feeds the power plant. Since the crusher is located at the bottom of the hill, the haul profile is suitable for loaded trucks. Coal is dumped onto an adjacent stockpile if the crusher is down or surge capacity is reached. A Cat front-end loader moves the stockpiled coal to the crusher when needed (Stubblefield and Fish, 1992).

Reclamation. After all coal is mined, overburden from the next adjacent pit is spoiled by the dragline into the just completed cut. After the overburden is graded and contoured by Cat dozers and Cat graders, the scraper fleet redeposits the topsoil; the area is seeded in the autumn of the year. A mixture of grasses, forbs, and shrubs is planted. Where possible, surface water above the mining area is diverted into settling basins to control suspended solids in water being discharged from the mine area. Ground water encountered in the mine pits is pumped out and allowed to run through the surface drainage system. Due to the mine's diverse topography, drainage maintenance and reconstruction are important in reclamation (Stubblefield and Fish, 1992).

6.3.4.5 Ancillary Facilities

The maintenance facilities are located near the main haul road to the power plant and adjacent to the mine administration building. The maintenance complex includes large equipment bays, bucket bays, wash and lube bay, welding shop, light duty shop, and electrical shop (Stubblefield and Fish, 1992).

6.3.5 Contour Mining, Horizontal Coal Seams: Martin County Coal Corporation (Case Study)

6.3.5.1 Mine Description

Martin County Coal Corporation's mine is located in the Appalachian coalfields, Martin County, Kentucky (KY), near the West Virginia border in the United States. Wolf Creek, the major watershed within the leasehold, bisects it. The total leasehold covers more than 134

km$^2$. There are two surface mines (45% of the annual production), one of which uses exclusively mountaintop removal mining, and five underground mines. Rugged hills and ridges as well as V-shaped valleys are typical of the topography of the area (the cliff and slope topography). There are alternating benches of outcrop sandstone and slopes. The deep valleys have been carved by small streams, which occur within the steep slopes of mountain ridges. The ridges are narrow, sharp and rocky. There are many small streams in the area. These streams are contained within the steep slopes of narrow, twisted mountain ridges. The ridges are underlain by massive sandstone, causing them to be sharp and rocky. The height of the hills above the valley floors ranges from 120 to 240 m (Tussey, 1992).

6.3.5.2 Deposit Description and Geology

General Geology and Stratigraphy. The stratigraphic sequence of the property consists of sandstones, siltstones, shales, coals and underclays deposited cyclically. The strata overlying the coals, is sandstone. A structural dip (of about 1%) across the property is to the northwest (Fig.6.19). The two surface mines excavate six coal seams, which in descending order, are as follows the first, second, third, fourth, fifth and the sixth seam. All of these seams are high-quality bituminous steam coals. The first seam occurs only near the crests of the higher knobs on the property and ranges in thickness from 1.8 to 3.4 m. This seam is mined exclusively by the mountaintop removal method. Because it is close to the crests, its recoverable reserves are restricted. This seam consists of many benches with numerous partings. The second seam underlies the first by a depth of 3 to 12 m, ranges in thickness from 15 cm to 15 m, and has no partings. This seam is exclusively mined with the mountaintop removal method. The third seam occurs only locally is surface mined in along with the first and the second. The fourth seam underlies the second by a depth up to 64 m. Its thickness averages from 1,3 to 1,5 m. This coal is mined by a combination of mountaintop removal, contouring, and underground mining. It consists of a single bench with a small parting. The fifth seam lies from 0.6 to 12 m above the sixth and they are surface mined together. It is typically less than 610 mm in thickness and is a high-grade coal with few impurities. The sixth seam underlies the fourth by 6 to 40 m across the property and is of the most complex stratigraphy there. It consists of three individual, distinct splits (upper, middle and lower). Total seam height ranges from 50 cm to several metres. However, the seam contains a lot of partings. Three splits are mined by both surface and underground mining methods (Tussey, 1992).
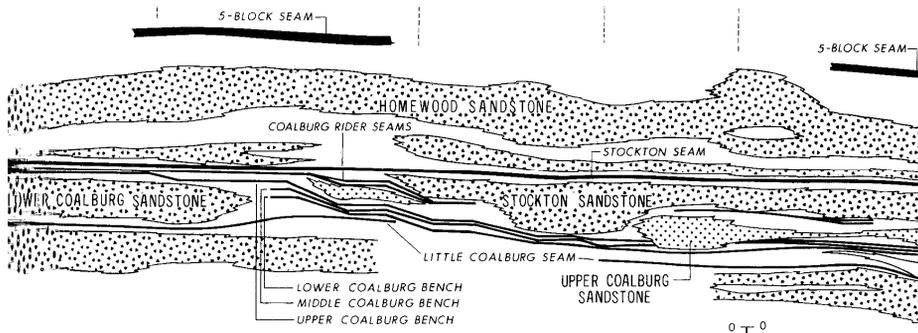


Fig.6.19. Geologic cross section of exemplary deposit (after Tussey, 1992)

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### 6.3.5.3 Mine Development

Contour Surface Mining. Contour surface mining is a technique of mining that enables one to uncover the coal by removing the soil and rock (overburden or spoil) directly overlying the coal with bulldozers, trucks, shovels, and other earthmoving equipment. The mined area is then reclaimed to its approximate original contour (AOC) by depositing overburden from the next successive cut in the area and then finishing by grading and seeding. This procedure is repeated along the outcrop of the coal seam as successive cuts are taken. The exemplary surface mine operation is engaged in contour surface mining. In a mine plan, designs for these surface and underground mines are developed paying attention to the safety and efficiency of each operation and least interference with each other (Tussey, 1992).

Pond and Access Road Construction. At the beginning of mine development sedimentation ponds have to be constructed. Their purpose is to trap any runoff from the area being mined and to maintain water quality. The ponds have to be properly constructed because they should remain intact during mining and reclamation activities. They are built in streams at places where major streams meet smaller ones. After all trees and other organic materials are removed from the site, bulldozers push earth from the valley slopes into the streams and compact the material in lifts to the elevation of the prime spillway. A metal pipe, placed at proper elevation, makes the prime spillway. Additional earth is then placed on the dam and compacted to the proper elevation. The emergency spillway is then built by cutting into solid rock on one side of the pond. After the sedimentation pond is finished, the construction of roads to the area begins. Roads are built on as moderate a gradient as possible and are made of durable high-grade, shot-rock, so that they can be used permanently after reclamation is completed. The roads cross an existing haulage road and are constructed to a head-of-hollow fill where mining begins. Road width averages 30 m (Tussey, 1992).

### 6.3.5.4 Mining Operations

Mining starts with cutting the timber overlying the proposed cut on the coal seam. Bulldozers drag this timber to the outcrop of the coal seam. A ditch is concurrently built to prevent rock or spoil from falling below the outcrop line. Timber in the head-of-hollow fill is also cut and removed. Because of the steep slopes and timbering activities, little or no topsoil is present on the property. Consequently, removal and redistribution of topsoil is redundant. Alternate topsoil material is located in close proximity to the coal seam. This material is removed at the end of mining. Therefore, it is used in backfilling operations and for planting plants (Tussey, 1992).

Overburden Removal. At the beginning, a drill bench for the highwall is prepared with a bulldozer. Next, a crawler-mounted rotary drill begins drilling presplit holes. Presplitting is accomplished by utilizing ANFO, cast primers, and non-electric caps and lead line. Because hill seams and other geological discontinuities occur randomly, presplitting the highwall is necessary to minimize their potential bad effects on safety and productivity. All presplitting is made vertically, which results in vertical highwall angle. Benching the wall is not attempted, in order to obtain and a cleaner, smoother, and safer wall. Highwall height and subsequent pit widths vary and are dictated by seam thickness, existing topography, equipment size, and other factors. After the presplit shot has been detonated, the overburden is drilled with a crawler-mounted rotary drill. The drillholes are loaded using cast primers and non-electric blasting caps. After the overburden shot is detonated, bulldozers push the spoil into the head-of-hollow fill and simultaneously construct another drill pad. The crawler-mounted drill is again moved back to the site and drilling is repeated. This shot is drilled on a given pattern. Bulldozers move a large part of this shot into the head of hollow fill. Rubber-tired loaders and

trucks assist the bulldozer and haul the remaining spoil into the head-of-hollow fill. The highwall is cleaned of any small rocks with the loader. After this spoil has been removed, the crawler-mounted drill again prepares another presplit shot to the coal. After detonation of this shot, drilling the remaining overburden is conducted with drillholes going down to the coal seam. When this shot is detonated, the loader and trucks move the blasted material to the head-of-hollow fill (Tussey, 1992).

Coal Removal. Once the spoil is completely removed, the coal is cleaned and piled with small rubber-tired loaders. Because the coal is friable, blasting is not required. Dozers rip or break the coal, when needed, to facilitate loading (Tussey, 1992).

Backfilling. Backfilling operations begin when the coal is removed. The pit is directly backfilled with overburden from the next adjacent pit to be mined. This spoil material is moved into the pit with bulldozers, loaders, and trucks. The overburden removed from the coal is placed into the previous pit to restore the mined land to AOC. When a sufficient amount of spoil is placed in the pit to eliminate the highwall, bulldozers start final grading operations. The spoil is pushed to the top of the wall and 'walked in' to the toe of the backfill. A road is left at the bottom of the backfill to provide access to the mine. This activity is repeated as the operation moves along the periphery of the coal seam. The final slope configuration is about 30°. Because overburden removal and backfilling operations are performed in sequential order, it is difficult to maintain more than one pit in either coal seam. When it is possible to access the coal from two directions, two or more pits may be open at the same time (Tussey, 1992).

The head-of-hollow-fills are sized as small as possible to make disturbed area as small as possible. They contain only the volume of rock coming from the swell of the blasted rock and the volume included in retention of "face-up" areas for underground mines. The 'face-ups' are built near the head-of-hollow fills to have the additional area for coal stockpiles, mine management areas, etc. Because the highwall has to be eliminated, contour mining must end on a point of a ridge so that spoil rehandling can be also eliminated. On this point the higher cut is made, and the excess spoil is used to eliminate the wall from the previous cut. By ending mining operations on this point, rehandle of spoil material is eliminated, and the remaining highwall is reclaimed as it is mined. After the lower seam is mined, the overlying seam is mined in the same manner. This method leaves a long, narrow, flat area on a ridge that originally was rocky and steep (Tussey, 1992).

Auger Mining. Auger mining follows contouring in some cases but is not an integral part of the mining operations. Augering is used only when mountaintop removal cannot be conducted or where underground mining cannot be accomplished because of inaccessibility. This inaccessibility may result from narrowness of the ridges, property lines, or previous mining operations (Tussey, 1992).

Reclamation. It begins when the pit is backfilled with spoil from the next adjacent pit. This operation is continued until the area has of its nearly original contour. Bulldozers then prepare slopes to final grade. Following final grade work, the slopes are sprayed by a hydroseeder with a mixture of grasses, legumes, fertilizers, and mulch. Trees are then planted. The head-of-hollow fills are reclaimed after they have reached their planned capacity. The outslope of the hollow-fill is graded to 2:1 (horizontal to vertical). Benches or terraces are graded to a 3% slope toward the fill and are not higher than 15 m. Surface runoff is carried across these terraces to drainage channels through the fill, the top of which is graded to 5% toward these drainage channels (Tussey, 1992).

6.3.5.5 Ancillary Facilities

A four-bay shop area is located next to the mine. It is designed for major repair of trucks and other mobile equipment. Drills and other large equipment are serviced in the field (Tussey, 1992).

6.3.6 Lignite: Big Brown Mine (Case Study)

6.3.6.1 Mine Description

The Big Brown mine is a mine-mouth operation operated by Texas Utilities Mining Co. The mine produces lignite as fuel for the two nearby units of the Big Brown Steam Electric Station. The mine is located 144 km south of Dallas, Texas, in the United States. Initial lignite production began in 1971. The mining area covers about 60.7 km$^2$ (Rand, 1992).

6.3.6.2 Deposit Description and Geology

The lignite at the mine is recovered from two minable seams within the formation, which is the uppermost one, Early Eocene. The reserves trend southwest to northeast and dip to the southeast at about 1°. The overburden and interburden material consists of unconsolidated fine-grained sand, silts and clays that do not need drilling or blasting to be removed. The overburden contains some groundwater that must be drained in advance of stripping operations. Lignite is excavated from two mining areas, which are about 1.6 km apart. Two lignite seams occur in the southernmost area, the B area. The southern two-thirds of the northernmost mining area, the C area, has two lignite seams, and the northern third of this area has a single seam. The profile of this area is shown in Fig.6.20 (Rand, 1992).

6.3.6.3 Mine Development

Initial mining operations began with a box cut along the sub-crop at the southwest edge of the deposit and progressed down-dip. As the overburden became deeper, sand channels with groundwater were found in the overburden, which resulted in highwall and spoil instability. Dewatering wells have been built to remove the groundwater in advance of stripping operations. Surface water run-off from disturbed areas (including haul roads) is contained and processed by a network of sedimentation ponds and then discharged into the natural drainage. In the B area, all overburden and interburden materials are removed with a dragline working in a pit of 3200 m in length. Haulage ramps go into the pit at each end and in the middle from the highwall side. The C area contains one pit (8250 m long) in which two draglines and a cross-pit spreader remove overburden and interburden. In this pit, haulage ramps are at each end and along the highwall at about 1200-m intervals. In both mining areas, highwall angles are 70° while spoil angles 30°. Pits are about 37 m wide. Slopes on haulage ramps cannot exceed 8°. Occasionally, scrapers are used to remove overburden and interburden material (Rand, 1992).
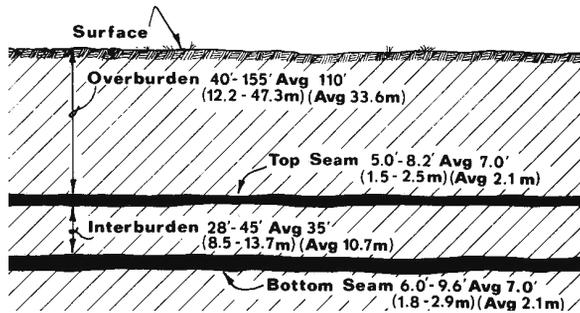
Fig.6.20.Cross section of C area (after Rand, 1992)

### 6.3.6.4 Unit Operations

Overburden Removal. In the B area, stripping operations employ a dragline with a 54-m$^3$ bucket, which uncovers the upper lignite seam by means of conventional side-casting from the highwall. The dragline working from the spoil side of the pit removes the interburden. Fig.6.21 presents a range diagram of this operation. A caterpillar (Cat) dozer works around the dragline, making the walkway ready, moving the power cable, and cleaning the top of the lignite seam in advance of loading. In the C area, overburden is removed (up to 30 m) with a cross-pit spreader (XPS) system (Fig.6.22), a dragline with a 54-m$^3$ bucket and a dragline with a 45-m$^3$ bucket remove the rest of overburden and interburden (Rand, 1992).
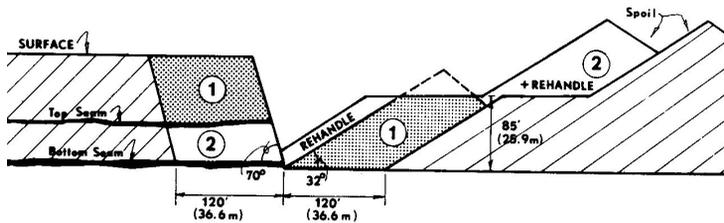


Fig.6.21. Range diagram B area (after Rand, 1992)

The XPS system is a pre-stripping system that is used to help to uncover deeper lignite seams. The XPS system is able to remove unconsolidated overburden. The material is transported across the pit and deposited behind the second dragline spoil pile by means of the 200-m discharge boom. This system allows construction of a bench for a dragline so that it can remove the rest of the overburden above the uppermost lignite seam from the highwall side of the pit. Once the upper seam is loaded, a dragline located on the lower bench can remove the interburden. The XPS system and dragline combination is presented in Fig.6.22 (Rand, 1992). The XPS system involves two bucket-wheel excavators (BWEs) that discharge bank material onto a mobile across-the-pit conveyor. Overburden material is transported (about 300 m) from the digging face to the spoilpile (Rand, 1992).

Lignite Loading. Lignite loading in the B area is performed using a Cat front-end loader (FEL) with a 12.2-m$^3$ bucket. In the area C, lignite is loaded with a backhoe with a 10.3 m$^3$

bucket as well as a Cat front-end loader (FEL) with a 12.2-m³ bucket. The backhoe and FELs operate at satisfactory availability (Rand, 1992).

Lignite Haulage. Lignite is hauled from the pits by means of 91-t bottom-dump coal haulers. 30 kms of haul road are serviced with five Cat motor-graders and four 30.3-m³ water-trucks. The haul road width of 15.3 m provides safe passage of haulage trucks and service equipment. Water is used for elimination of the dust on haul roads. It is taken from holding ponds used for surface water control (Rand, 1992).

Lignite Processing. Haulage trucks dump lignite into a double-compartment hopper. The lignite flows from the hopper through two reciprocating feeders into two double-roll crushers. The lignite is crushed and dropped onto a belt conveyor, which transports it to the power plant stockpile (Rand, 1992).
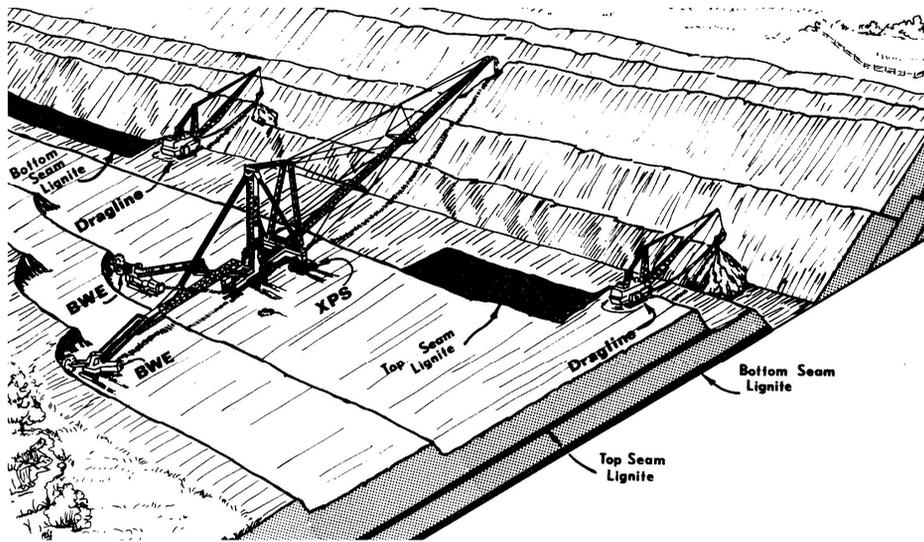


Fig.6.22. Cross-pit spreader and dragline combination C area (after Rand, 1992)

Reclamation. The mined area is levelled to the nearly original contour, as soon as the stripping process finishes. The spoil areas are levelled with three dozers. Next, various cover crops are planted and grasses are seeded. Ponds can take over 2% of the reclaimed area. The ponds facilitate pasture management and provide the required environment for certain shrub and tree species (Rand, 1992).

6.3.6.5 Ancillary Facilities

The shop has a tool room, welding machines, air compressors, a lubrication tank, drive-through bays, and overhead cranes. Repair and maintenance of mine vehicles are performed in the shop. Routine maintenance and small repairs on mobile equipment working in and around the pit area are conducted in the field. Other usable equipment and vehicles are parked in the ready area when not being mined. The warehouse is located in a separate building. The outside storage area provides space for tyres, bucket parts, wire rope, large shafts and gears (Rand, 1992).

6.4 Auger Mining

Continuous Surface Mining with Augers

Auger mining is the term for method used to recover coal from thin seams, 0.6 to 2.4 m thick, beyond a highwall formed using conventional open cast or strip mining (area mining or contour mining), or from seams which deep trenches make available. To excavate coal, openings are bored into the seam beneath the overburden. Where conventional surface or underground methods cannot be economically or technically used, this method is applied (Treuhaft, 1984).

An auger consists of three parts: (1) cutterhead, (2) auger flight(s), and (3) prime mover. They are mechanically connected and make an integral power train. The coal is cut and fractured with the cutterhead, which is a barrel-shaped device. The auger flight is a screw conveyor transporting the coal to the surface, which is in form of a spiral wound about a drill stem. In the case of a single auger, the turns of the drill stem push towards the open end of the hole the coal fragments trapped by the sides of the hole (McCarter, 1992). The cutterhead can have more than one bit assembly. There are single heads from 0.5 to 2.5 m, double heads from 0.5 to 0.9 m, and triple heads from 0.5 to 0.6 m, available (Hartman, 1987). The multiple heads can cut overlapping patterns, leaving less coal left at the pillar. When two cutterheads are used, each turns in opposite directions to make the coal move towards the centre of the flight. This counter-rotation stabilizes the mounting and helps drill straighter holes almost without out-of-seam deviation. The prime mover is either skid or pad mounted. It is equipped with jack-legs to level the carriage. The engine, drive head, and controls are mounted on this carriage. The carriage is able to drive towards the face and move back far enough to put another flight to deepen the hole. Individual flights are stored in racks, and are handled by hoist fixed to the frame. When coal comes to the surface, it is elevated by conveyor or front-end loader and placed in trucks. The auger has been developed to the thin seam miner (TSM), a type of surface continuous miner. This equipment has a cutterdrum that cuts a rectangular hole. Hydraulic hoses power the cutterdrum. The whole assembly is forced into the coal seam by push beams and rigid box sections, which contain a screw conveyor to transfer the coal to the surface. The cutterdrum has also sensors, which enable one to detect the bottom and top of the seam (McCarter, 1992).

The mining cycle for the TSM is automated and involves a sump cut at the base of the seam. The cutterdrum pivots upward, shearing the coal until it comes up to the top of the seam and then advances forward for another cycle. Ten cycles are made before another push beam is added. Ten push beams can be used to achieve a total depth of 73 m. The mining cycle lasts about three hours (Chironis, 1984). Auger mining is used in the hilly areas where the overburden increases rapidly and thin seams are common (McCarter, 1992).

Development of an augering operation is relatively simple. Continuity, uniform seam thickness, and near-horizontal orientation (less than 10°) are important for this technique. When during an initial drilling, the continuity is established, the face is made ready by cleaning the highwall. The pit floor is then levelled and cleaned to enable alignment of the prime mover and auger flights. A roadway parallel to the highwall is necessary for access, beltway, and/or truck transportation of the materials. Efficient operation requires only a clear area 15 to 25 m wide (Hartman, 1987). Operations are low cost and highly productive. Coal recovery, however, may be as low as 40 to 60%. Recovery can be increased by the shape of the highwall. If the holes are spaced far apart, coal will be lost between the holes. If they are close together, the ribs may be too thin to support the overburden. Maximum recovery is achieved with straight highwalls. Careful mine planning can optimise the shape of the final highwall in advance of augering. Furthermore, to achieve maximum coal recovery,

appropriate patterns for the auger holes are needed. Fig.6.23 shows the effect of highwall curvature on the spacing of holes and strategy for various hole patterns (Treuhaft, 1984).
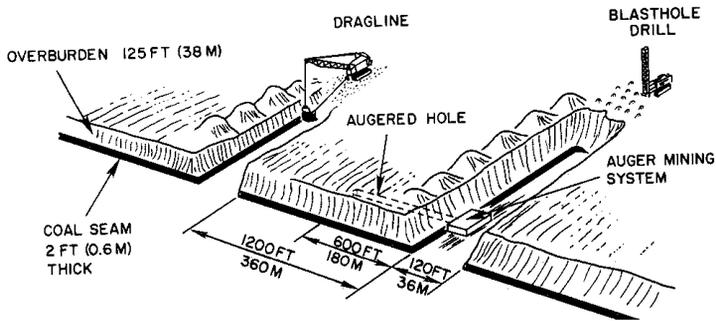


Fig.6.23. Auger hole patterns and effect of highwall curvature (after Treuhaft, 1984)

A method that may be used for relatively flat topography and deep seams is called trench-strip auger mining (Treuhaft, 1984). This method must have a series of parallel trenches to be excavated as Fig.6.24 shows. The coal between the trenches can then be removed with augers operating at right angles to the trench. The straight highwall should maximize recovery and minimize the surface area to be reclaimed. Auger mining can be introduced with low capital investments. It is safer than underground mining and more productive than both surface and underground methods. Productivity decreases with increasing depth of auger hole. However, it is highly specialised skill and requires a lot of experience to be successfully employed. Reclamation regulations influence negatively the application of auger mining. The requirement to immediately cover exposed highwalls encourages abandonment of reserves beyond the economic limit of conventional mining if auger or TSM are not immediately available. The requirement to restore highwalls to the approximate original topography has also made contour mining less attractive, which results in fewer mines available for augering (McCarter, 1992).
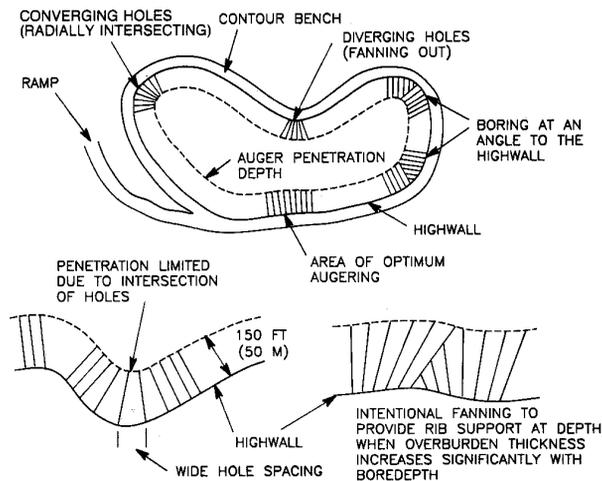


Fig.6.24. Trench-strip mining with the extended depth highwall mining system (after, Treuhaft, 1984)

# 7 Surface Mining: Comparison of Methods

Mining operations differ, either significantly or slightly, from one another in some respects such as deposit size, geometry, hydrology, geological structure, grade, mineralogy, location, or geotechnical properties; ownership type; and legal or environmental regulations. However, each mining operation sets itself the same goal of maximizing the return on the investment. The mining law says that a mineral resource can be referred to as a minable reserve if it is possible to extract the mineral(s) economically and legally. The deposit type, the mining rate, and the physical features of the deposit decide the selection of the most economically feasible method. In order to compare surface methods, one has to consider two mine planning goals of: (1) developing the most economic plan that will return the maximum profits (2) maximizing the recovery of the resource (Bohnet, 1992).

Surface mines can be subdivided into classes and subclasses as follows (Bohnet, 1992): Class: Mechanical; Subclass: -; Method: Open Pit Mining, Quarrying, Strip Mining, Auger Mining; Class: Aqueous; Subclass: Placer; Method: Dredging, Hydraulic Mining; Class: Aqueous; Subclass: Solution; Method: Surface Techniques, In Situ Leaching.

## 7.1 Factors and Conditions

Open pit mining and strip mining are the two most common types of surface mining methods in the world. They contribute to almost 90% of the surface mineral production. Strip (open cast) mining is utilized in large, tabular, flat-lying ore bodies or mineral seams, such as coal, that are close to the surface. In these mines, advancing mine operations should involve backfilling, which is not costly. Open pit mining is used in disseminated ore bodies or steeply dipping veins or seams where the mining progresses toward increasing depths. Backfilling is postponed until the pit is finished. The cost of filling these pits is high since huge quantities of waste rock have to be removed when mine life ends (Bohnet, 1992).

Planning methods and terminology used for seam deposit mining, like coal, differ from those applied in open pit mining for disseminated deposits. However, in all applications of surface mining methods mining operations are performed up to the economic limit. The two types of deposits use different procedures for evaluating the economic limit (Bohnet, 1992).

There are other factors influencing the choice of the type of mining method and the type and size of equipment such as: (1) Topography, (2) Remote location of mine, (3) Support infrastructure, (4) Availability and cost of skilled work force, (5) Climate and altitude, (6) Environmental constraints, (7) Groundwater conditions, (8) Slope stability and ground conditions, (9) Investment risk, (10) Blasting requirements, (11) Length of mine life, (12) Waste rock disposal and haulage distances (Bohnet, 1992).

Quarrying is an open pit mining used to produce dimension stone such as marble, granite, limestone, sandstone, flagstone, and slate. When rock joint fractures are frequent the bench faces may not be vertical. The lack of fractures allows near vertical highwalls as high as 300 m. Quarrying is a high-cost selective method (Bohnet, 1992).

Auger mining is mainly used to remove coal from under an ultimate highwall. The final highwall position depends on economic stripping limits, boundaries of mine, or physical constraints such water presence. If physical constraints occur, auger mining may be the only method to increase the coal production (Bohnet, 1992).

Aqueous surface mining methods, which use water for extraction, can be applied in special conditions. Dredging and hydraulic mining can be used to extract placer-type deposits that contain heavy metals in unconsolidated overburden if: (1) Water is available in adequate quantities. (2) The mining operation can obey environmental regulations. Hydraulicking, called hydraulic mining, is used in areas where a natural gradient away from the deposit exists

so that the ore or waste slurry can be hydraulically transported to the process or disposal site. Dredging can be used if the placer deposit underlies a natural or man-made water container. Dredging machines can be mechanical, bucket-line, bucket-wheel suction, and dipper; or hydraulic, suction, and cutterhead. The selection of dredging equipment depends on the type of material to be excavated (Bohnet, 1992).

Solution mining employing surface or in situ techniques is used for deposits of minerals that can be excavated by dissolution as well as by melting, leaching, or slurrying. The two methods are similar. Surface leaching employs heap or dump leaching of mineral values; copper, gold, and uranium are the examples. In situ mining uses water to dissolve, melt, or slurry the minerals. Barren solution is introduced down one set of wells and the loaded solution returns to the surface through concentric or another set of wells. This mining method is mainly used with sulphur, evaporite, or water-soluble minerals. In situ leaching utilizes chemical or bacteriological reagents, usually mixed with water to selectively dissolve the valuable minerals. Drillholes are used to inject and recover the solution (Bohnet, 1992).

The mining method is closely associated with the selection of the appropriate size and type of equipment. Both strip mines and open pit operations employ loading and hauling fleets. The type of equipment is chosen on the base of the size of the operation over time. Loading and hauling fleets are dependent fleets, which means that their effectiveness and availability are interrelated.. Therefore, to have an effective operation, more than one unit per equipment type is necessary. A mining operation should have one drill per shovel and at least one back-up shovel/loader to load trucks when a scheduled shovel breaks down (Bohnet, 1992).

Large mining operations need a lot of time, since they have to develop working area big enough for the equipment to be effectively used. Therefore, the equipment chosen for full production may not be suitable for the development or preproduction stage. Moreover, the development period may need different mining methods because material to be mined, is less moved and of different types. During the development stage, haul roads have to be built, and if land is rugged, benches may be much narrower than those utilized during production period. Therefore, smaller equipment may be more convenient during this stage. Sufficient time is necessary to drill, blast, and remove the material from the mine. During preproduction and the stripping stage of each mining phase, higher benches may be used to allow the use of larger equipment and cost reduction. Larger open pit and strip mines are often subdivided into smaller phases to have project economics improved (Bohnet, 1992).

## 7.2 Summary

The five most important criteria in choosing the best mining method and equipment are (1) Type of deposit, seam or disseminated. (2) Geometry of the deposit and terrain. (3) Total material movement per time period. (4) Projected project life. (5) Geographic location. Other less vital factors are (1) Climatic conditions and altitude. (2) Availability and cost of skilled work force. (3) Environmental restrictions. (4) Slope stability and groundwater. (5) Investment risk and available capital funding. (6) Blasting restrictions. (7) Method of waste rock disposal, external or backfilling. (8) Availability of support services and infrastructure. (9) Availability of electric power vs. local cost of diesel fuel. Table 7.1 presents a comparison of ore body conditions favourable to the different surface mining methods. The advantages and disadvantages of each surface mining method are shown in Table 7.2. To sum up, many factors need to be taken into consideration in selecting the best method associated with the most economic fleet of equipment. When there is more than one alternative, it is best to select one that makes the mining operation the most flexible. Since mining is not static, the operation that can fit in with changing conditions is the most suitable one (Bohnet, 1992).

Table 7.1. Comparison of Deposit Conditions Favourable to Surface Methods (after Hartman, 1987)

| Characteristic | Mechanical Extraction | | | | Aqueous Extraction | | | |
|---|---|---|---|---|---|---|---|---|
| | Open Pit | Quarrying | Strip Mining | Augering | Hydraulic Mining | Dredging | In Situ Mining | Surface Leaching |
| 1. Ore Strength | Any | Any (sound structure) | Any | Any | Unconsolidated, few boulders | Unconsolidated some boulders | Consolidated | Rubblized or cavable |
| 2. Rock Strength | Any | Any | Any | Any | Unconsolidated | Unconsolidated | Competent, impervious | Competent, impervious |
| 3. Deposit Shape | Any (preferably tabular) | Thick-bedded or massive | Tabular, bedded | Tabular, bedded | Tabular | Tabular | Any | Massive or thick tabular |
| 4. Deposit Dip | Any (preferably low dip) | Any, if thick | Any (preferably low dip) | Low dip | Low dip | Low dip | Any (preferably low dip) | Steep |
| 5. Deposit Size | Large, thick | Large, thick | Large, moderate thickness | Limited extent, thin | Limited extent, thin | Moderate extent and thickness | Moderate to large | Any (preferably large) |
| 6. Ore Grade | Low | High (assay not critical) | Low | Low | Very low | Very low | Intermediate | Very low |
| 7. Ore Uniformity | Uniform (or sort or blend) | Uniform | Fairly uniform | Uniform | Fairly uniform | Fairly uniform | Variable | Variable |
| 8. Depth | Shallow to moderate | Shallow to moderate | Shallow | Shallow | Very shallow | Very shallow | Moderate to deep | Shallow to moderate |

Table 7.2. Comparison of Advantages and Disadvantages of Surface Methods (after Hartman, 1987)

| Characteristic | Mechanical Extraction | | | | Aqueous Extraction | | | |
|---|---|---|---|---|---|---|---|---|
| | Open Pit | Quarrying | Strip Mining | Augering | Hydraulic Mining | Dredging | In Situ Mining | Surface Leaching |
| 1.Mining Cost (relative) | 10% | 100% (highest) | 10% | 5% | 5% | <5% | 5% | 5% |
| 2.Production Rate | Large-scale | Small-scale | Large-scale | Moderate | Moderate | Large-scale | Moderate | Moderate |
| 3.Productivity | High | Very low | High | Very high | High | Highest | Very high | Very high |
| 4.Capital Investment | Large | Small | Large | Small | Small | Large | Large | Moderate |
| 5.Development Rate | Rapid | Moderate | Rapid | Rapid | Rapid | Moderate | Moderate | Moderate |
| 6.Depth Capacity | Limited | Limited | Limited | Limited | Limited | Limited | Unlimited | Unlimited |
| 7.Selectivity | Low | High | Low | Low | Moderate | Low | Low | Low |
| 8.Recovery | High | High | High | Moderate | Moderate | High | Low | Very low |
| 9. Dilution | Moderate | Low | Low | Low | High | High | High | Very high |
| 10.Flexibility | Moderate | Low | Moderate | Very low | Moderate | Low | Low | Low |
| 11.Stability of Openings | High | Highest | High | High | Moderate | Moderate | High | Moderate |
| 12.Environmental Risk | High | Moderate | Very high | Low | Severe | Severe | Moderate | Moderate |
| 13.Waste Disposal | Extensive | Moderate | Minor | None | Moderate | Extensive | Minor | Minor |
| 14.Health and Safety | Good | Good | Good | Good | Fair | Good | Good | Good |
| 15.Other | Low breakage cost; rainfall and weather problems; large-scale best. | Waste-intensive; labour-intensive; high breakage cost. | No waste haulage; low breakage cost; large-scale best. | Restrictive; used for remnant coal. | Unconsolidated deposit; water required; no breakage cost; beneficiates. | Unconsolidated deposit; water required; no breakage cost. | Unconsolidated deposit; water required; no breakage cost; beneficiates. | Unconsolidated deposit; water required; no breakage cost. |

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Part III Underground Mining

# 8 Underground Mining: Case Study

The exemplary underground mining technology is used by the copper mines. This chapter presents technical and practical operational information related to the technology of exploitation in three Polish underground copper mines i.e., the Rudna mine, Lubin mine and Polkowice-Sieroszowice mine, which belong to KGHM Polska Miedź S.A. company. All the information and data presented in this chapter were taken from a book titled Monografia KGHM Polska Miedź S.A. written by A. Piestrzyński in 1996.

## 8.1 Exploitation Systems

Geological characteristics of a deposit, mechanical properties of deposit minerals and surrounding rocks, as well as in-situ conditions, strongly influence the selection and development of exploitation systems. The copper ore deposit is situated in the southwest part of Poland in the Lubin-Głogów district (Legnica-Głogów Copper District, abbr. LGOM). As a geological structure, the deposit belongs to the Fore-Sudetic monocline. Complex geological conditions and varying mechanical properties are characteristic of the deposit. It is gently dipping, tabular, and a nearly horizontal deposit (Kłapciński and Peryt, 1996). Since its discovery, economical and safe exploitation of the copper deposit has been largely dependent on the choice of a mining method. Due to the overall geological complexity of the deposit, when selecting an optimal mining technique one has to take into account some critical characteristics including: (1) substantial depth of the deposit occurrence that varies from 600m to 1200m below the ground surface; (2) diverse lithology; (3) substantial aerial extent combined with a low (4° to 6°) dip angle of mineralised strata; (4) rock competency differences (the strength of main roof strata is 7 to 10 times greater than that of the floor); (5) the ability of ore and surrounding rock mass to accumulate and violently release elastic strain energy; (6) complex tectonic conditions; (7) freely distributed mineral and the absence of macroscopically identifiable mineralisation markers; (8) deposit thickness varying from 0.4 to 26.0 m.

The above characteristics and technical-economic conditions were the main factors influencing changes and progress in the development of exploitation systems. They made it necessary to develop such mining systems that would allow for safe excavation and rational management of the deposit. Since 1974, when the first high rock bump (rockburst) hazard was noticed, technical modifications have had to result in bump hazard limitation. Tremors and bumps have been induced in the following way: very stiff pillars behaving as a quasi-elastic material have been suddenly destructed in the course of cutting them into smaller ones because of transformation of the solid rock state from the elastic to post-failure one (pillars have been crushed by the exploitation pressure of the rock mass). The methods for bumps and tremors control have been largely used and improved since then. Efficient use of the deposit resources is related to the excavation within protective (technological) pillars where the significant part of the ore deposit is located. Room-and-pillar mining methods have met these requirements. However, with the growing depth of exploitation and higher variability of rock mass and roof characteristics, the difficulties in using those methods caused by increasing pressure had occurred. Problems with proper adjustment of system parameters (pillar dimensions, rate of mining face progress etc.) to the given geology and rock mass conditions were encountered. The most important issue was selection of a technological pillar size providing the roof strata stability (proper control of stiff roof behaviour) (Piestrzyński, 1996).

When the room-pillar mining method is used at a great depth, a seam is cut by means of rooms with increased length (advance). Technological between-rooms pillars crack under the influence of the mining pressure and transfer into the post critical state (post-failure condition) (Fig.8.1). In the LGOM's mines, those pillars have a columnar wedge-shaped structure. Such the pillars behave like a yielding standing support, and they are not liable to accumulate the elastic strain energy. They secure the roof stability while being deformed under the influence of the main roof subsidence. These pillars of high deformability operate in the mining panel revealing post-failure supportability (resistance). The suitable (proper) resistance (load-bearing capacity) of the pillars is obtained by means of their planar dimensions and slenderness selected for the specific mining conditions and mechanical properties of the rock mass (Kunysz et al, 1996).
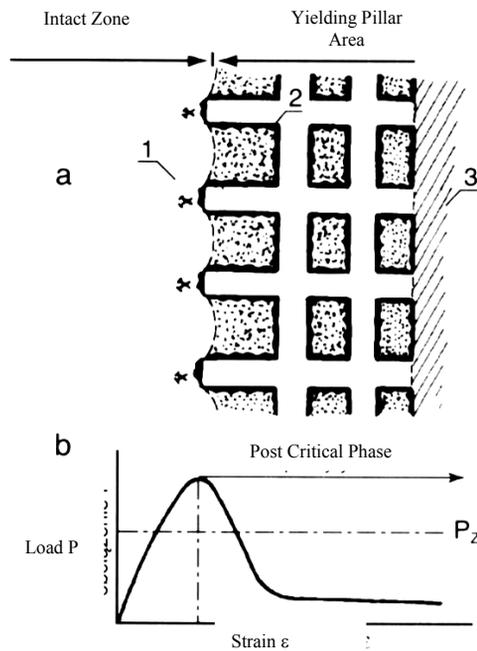


Fig.8.1.Schematic of one phase room-and-pillar mining system (a) simplified characteristics of the technological pillar behaviour (operation) (b) (after Kunysz et al, 1996)
1. intact zone ; 2. mined area with pillars in post critical (post failure) state (phase); 3. mined-out (gob) area

Such an excessive yielding load is imposed on pillars lying between the development (mine face) and gob area line. The width of such panels varies from several metres to above one hundred metres. When an area of great size has been worked out and a gob zone has reached a suitable width, yielding pillars get loaded with the total weight of roof rocks (Fig.8.2). It is accepted that in shallow mines the full load of pillars occurs when the panel's width is greater than 0.7 to 1.1 its depth. The post failure supportability (resistance, load-bearing capacity) of technological pillars mainly depends on the mechanical properties of pillars' rocks, the dimensions and shape of pillars, as well as on the rate of loading. The load-

bearing capacity of mining technological pillars may be calculated using the compression strength of pillars' rocks. The roof stability of the complete mining field is characterised by the ratio of the pillar load-bearing capacity (resistance) to their load. The same formula is used in copper mines operating at great depths (Kunysz et al, 1996).
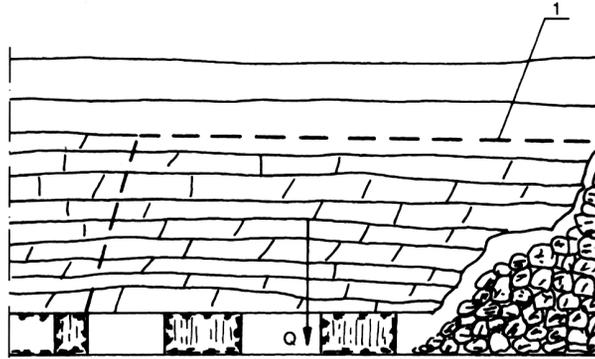


Fig.8.2. Schematic diagram of loading imposed on technological pillars by rocks loosened in the roof of the panel being mined at great depths with the use of a room-and-pillar method with roof caving. 1 – zone of loosened rocks (after Kunysz et al, 1996)

In Polish copper mines, room-and-pillar mining methods with continuous gob area liquidation are used. The ore is extracted by one mine face consisting of numerous room faces with a width of 5 to 7 m. The rooms are driven parallel (strips) or perpendicularly to the mining face line. In this way, the panel is cut into pillars with dimensions related to local geological-mining conditions. The pillars are left between the rooms. Several rows of strips and pillars are kept behind the mine face. Following the advancing face, the mined out space is liquidated behind the mining face line through inducing the roof caving, backfilling, partial roof deflection and caving, or partial backfilling and roof caving. Before the gob area is liquidated, pillars in the last row are ripped (cut) to get smaller ones to minimise mining waste. About forty types of room-and-pillar mining methods were given an approval for use in LGOM's mines (Janowski et al, 1996).

Fig.8.3 shows typical developments of mining panels with the use of room-and-pillar mining methods. The length of a panel varies from 200 to 600 m, and the panel's width reaches 1000 m. Generally, the exploitation is conducted to the rise. Mine development consists in making level roads and inclines as double or triple entries due to ventilation and transport conditions. Multi-road entries allow transporting large amount of air to panels, which is necessary because of highly intense mining, machines used, and high original temperature of rocks (Janowski et al, 1996).

One phase and two-phase room-and-pillar methods are used to mine the ore deposit. In two-phase methods, the mining panel is cut into large pillars with dimensions 25mX35m by rooms driven in the form of strips (phase I). Next, the large pillars are cut by means of rooms into smaller ones (phase II) (Fig.8.4). In one-phase systems, the ore is mined by one face consisting of numerous room faces connected by workings with technological pillars left between the rooms to secure the roof stability within the working space (Fig.8.5). The deposit of a medium thickness (3-5m) was extracted with the use of one-phase method with induced roof caving (Fig.8.5). The roof-caving increment clasped one row of pillars and one row of rooms. The increment was about 30 m long. The roof caving was induced through blasting in long blastholes drilled into the roof. The roof hole blasting was associated with blasting the

remnant (between-rooms) pillars (Fig.8.6). The length of the roof holes was 12 m and was too small to obtain the volume of rock waste suitable for supporting the roof, which sometimes broke away in an uncontrolled way. Therefore, methods for the roof control were improved, and consequently mining methods with limited roof caving and with partial roof deflection (retarded caving or bending) without induced roof caving as well as those with hydraulic backfilling were introduced. Fig.8.7 shows a stowing station with surface backfilling and water containers (Janowski et al, 1996).
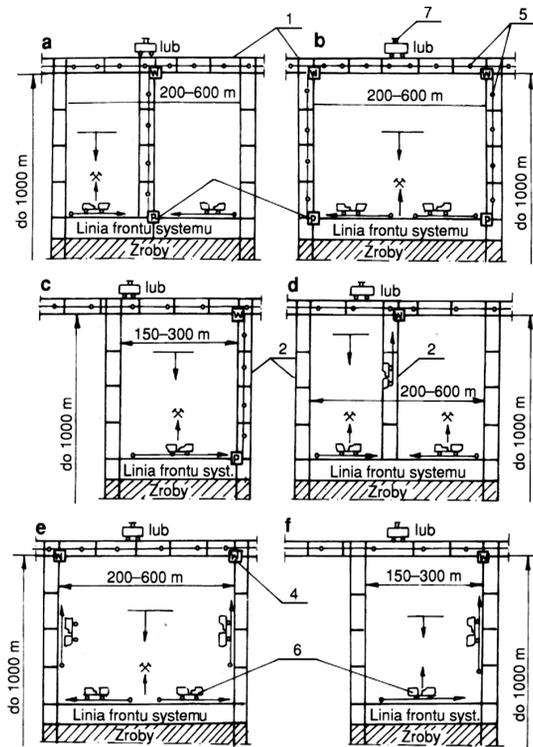


Fig.8.3. Types of mining panel development (after Janowski et al, 1996)
1. level roads; 2. inclines; 3. transfer point for the mined ore from underground haulers onto a belt conveyor; 4. transfer point for the mined ore from a belt conveyor onto the rail transport or main conveyor; 5. a conveyor flight; 6. haulage with underground haulers; 7. rail haulage

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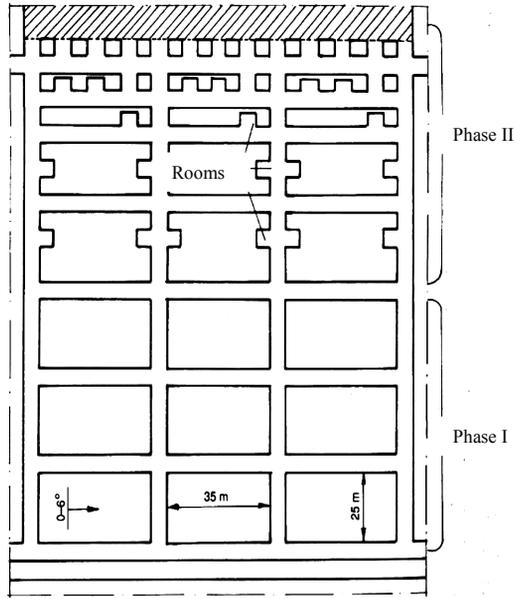
Fig.8.4. Two-phase room-and-pillar mining method (after Janowski et al, 1996)
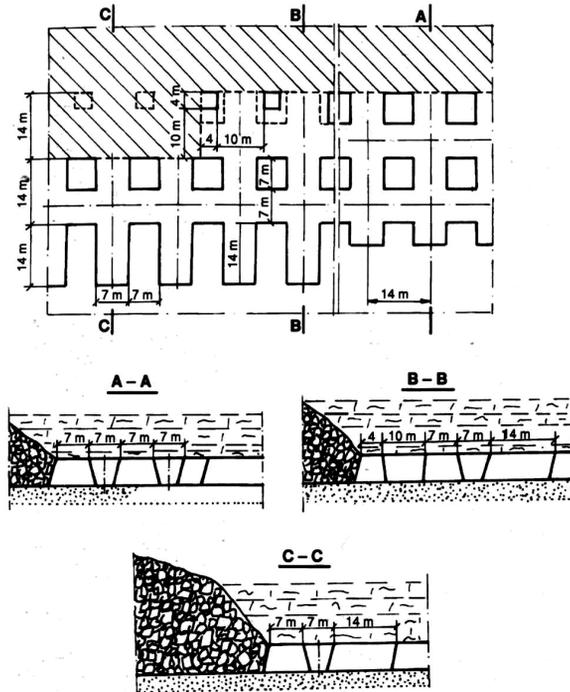


Fig.8.5. One-phase room-and-pillar mining method with roof caving (after Janowski et al, 1996)

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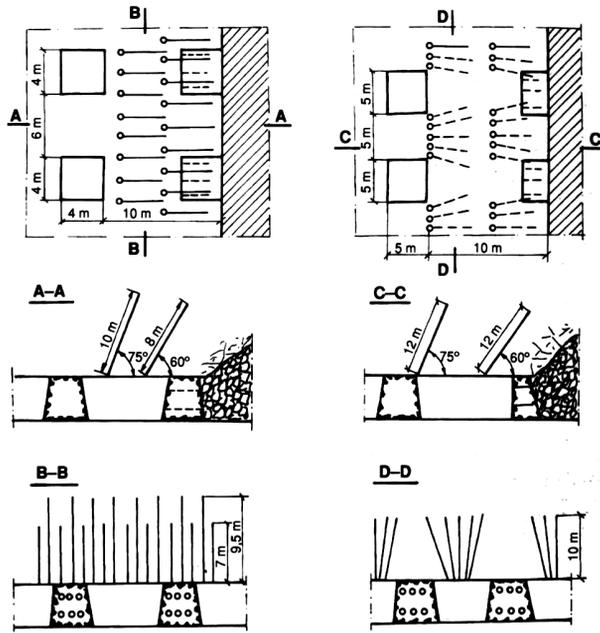
Fig.8.6. Arrangement of roof blastholes to induce the roof caving (after Janowski et al, 1996)
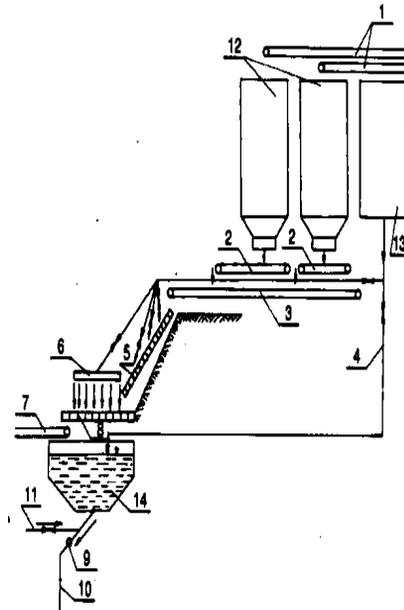


Fig.8.7. Stowing (backfilling) installation (after Piechota, 1988)
1. belt conveyors delivering backfilling material; 2. metal link-belt conveyors; 3. belt conveyor; 4. water pipeline; 5. stationary screen; 6. spraying pipe; 7. conveyor for mesh fraction haulage; 8. vibrating screen; 9. isotopic densimeter; 10. stowing pipeline; 11. emergency water pipeline; 12. backfilling material tank; 13. water reservoir; 14. stowing (backfilling) hopper

Hydraulic Backfilling. Sand is transported from a recharging point to containers located above the ground surface (Fig.8.7). Next to the sand container, there is a tank of water used to wash gravitationally the sand away. Due to surface location of the tanks, the water and sand may be batched automatically. The sand is batched through the outlet closed with the culvert hatch that is controlled by the operator. Under the tanks' outlets, there are metal link-belt conveyors that load the sand onto the belt conveyor equipped with a device for weighing the sand. This belt conveyor loads the sand into the stationary screen, which uses water to wash the sand from the gravitational water tank. Washing the pipeline, preparation of the mixture and feeding the stowing hopper are controlled automatically by the operator. After establishing the mixture parameters and feeding amount, the quantity of water and sand are automatically regulated (Piechota, 1988; Janowski et al, 1996).

Since 1994, the one-phase room-and-pillar method with retarded caving (roof deflection) has been predominantly used in Polish copper mines. The idea of mining with the retarded caving (roof deflection) consists in cutting the ore by means of the system of rooms, and creating this way technological (between-rooms) pillars with such the dimensions that allow the pillars to yield to an excessive load (make the pillars break and work in the post-critical state). The yielding pillars working within the decreasing branch of the stress-deformation characteristics (Fig.8.1) allow the retarded roof caving (deflection) in the same way as timber piles or dry backfill strips do in the classical mining methods. Partial roof caving takes place far from the mining face front after partial extraction of the technological pillars. The bolted roof (resin-bedded and expansive bolts) has to have the tendency to deflect (bend) without causing the caving on the mining front. Planar dimensions of pillars and their slenderness are being selected depending on the characteristics of pillar rocks and local geological conditions. This method for the roof control has been used to mine very thin parts of the deposit as well. Ripped barren rocks are placed into the gob area in the thin conditions. The number of the strip rows behind the mining face has been increased. Introduction of the methods with retarded caving (roof deflection) has improved the stability of mine workings (Janowski et al, 1996).

In the deposit with the thickness smaller than 5 m, the two-phase method with hydraulic backfill was used. In the first stage, the panel was cut by means of headings into pillars with dimensions 25m x 25m (Fig.8.8). During the second stage, the pillars were mined using six-metre-wide rooms leaving a four-metre-wide pillar (barrier) at the side of the goaf area. This pillar was then ripped to get a one-metre-wide pillar that served as a fence separating the rooms from the gob area. After mining the panel that consists of three rooms, screens were built and the fenced up space was backfilled (Fig.8.9). The two-layer, two-phase room-and-pillar mining method with two layers of hydraulic backfilling has been worked out to mine the thick deposit. In the first stage of mining, the upper layer where development workings for both layers are located, is excavated. The lower layer is mined from the workings of the upper layer. Gravity inclines are built from the upper layer workings to haul the mined ore from the lower layer (Janowski et al, 1996).
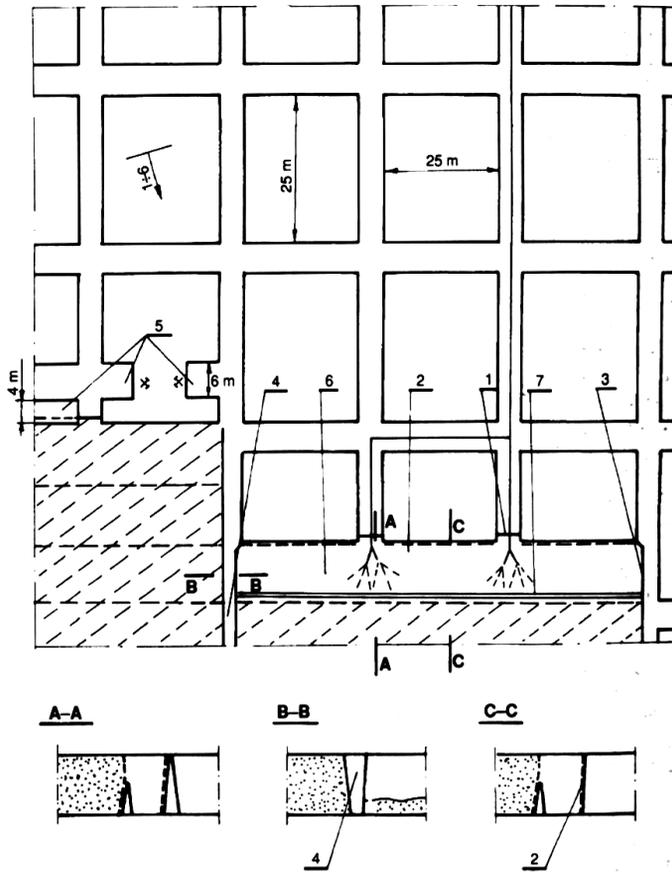
Fig.8.8. Two-phase room-and-pillar mining method with hydraulic backfill (after Janowski et al, 1996)
1. head screen (stopping); 2. elastic screen (stopping); 3. side screen (stopping); 4. road fenced in backfill; 5. mining; 6. backfilling; 7. ore and brattice
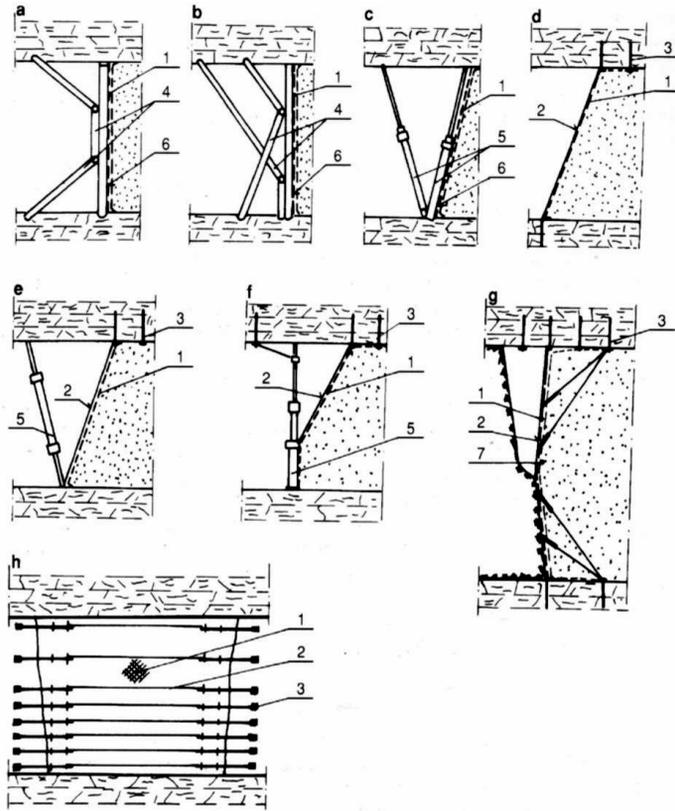
Fig.8.9. Schematic diagram of the construction of screens (backfilling stoppings) used in Polish copper mines (after Janowski et al, 1996)

a, b-timber screens; c-metal screen; d, e, f, g, h-rope screens; 1. screen sheet; 2. head ropes; 3. bolts; 4. timber anchor prop; 5. metal prop; 6. boards; 7. metal or plastic mesh (screen)

## 8.2 Room-and-Pillar Mining Methods for Underground Cooper Mining

### 8.2.1 Room-and-Pillar Methods for Deposits with Thickness from 3 to 5 m

#### 8.2.1.1 One-Phase Room-and-Pillar Method with Roof Caving (Fig.8.10)

The panel with a width of 200 to 600 m is separated by means of double entries (Fig.8.10). The deposit is mined with the use of six-metre-wide rooms leaving technological between-rooms rectangular pillars with a width of 5 to 7 m and length of 16 m. Three rows of rooms and three rows of pillars are kept on the mining face. Next, the pillars located next to the gob area are developed by means of seven-metre-wide insets (roads) driven towards the goaf area. The resistant (supporting) pillars with dimensions 7 x 4.5 m are separated in this way. Before the roof caving is induced, the supporting pillars are ripped to smaller ones (remnant pillars) and their final area, measured at the roof reaches about 12 square metres. The roof caving is induced through blasting the remnant pillars associated with blasting in the

holes drilled into the roof. The deposit is mined with drilling-and-blasting technique. The blastholes are 2.0 to 3.5 m long. Distressing shooting is also employed when necessary. The mined ore is loaded with loaders (scoop-trams) onto haulers or/and hauled to recharging points onto belt conveyors or to district bunkers at loading points. To protect the roof, the walls of rooms, strips and insets at the mining face, roof bolting is employed. The working face is ventilated with fresh airflow. Separate (local) ventilation (booster fans) is also used. The mined-out space is liquidated through the roof caving, which is mainly induced by means of the roof blasting. At the same time, the last row of pillars is also liquidated with blasting technique (Cieślik et al, 1991).

Mobile machines perform all the mining operations at the mining face: (1) Blasting holes are drilled with drill carriages; (2) Dry blasting agents are charged with pneumatic charging devices; (3) The mined ore is loaded and hauled with mobile loading-hauling-dump (LHD) machines; (4) Mobile bolting cars are used to bolt roofs and walls; (5) Blastholes are drilled in the roof with mobile drill carriages; (6) Mobile rippers are used to rip the roof and walls.
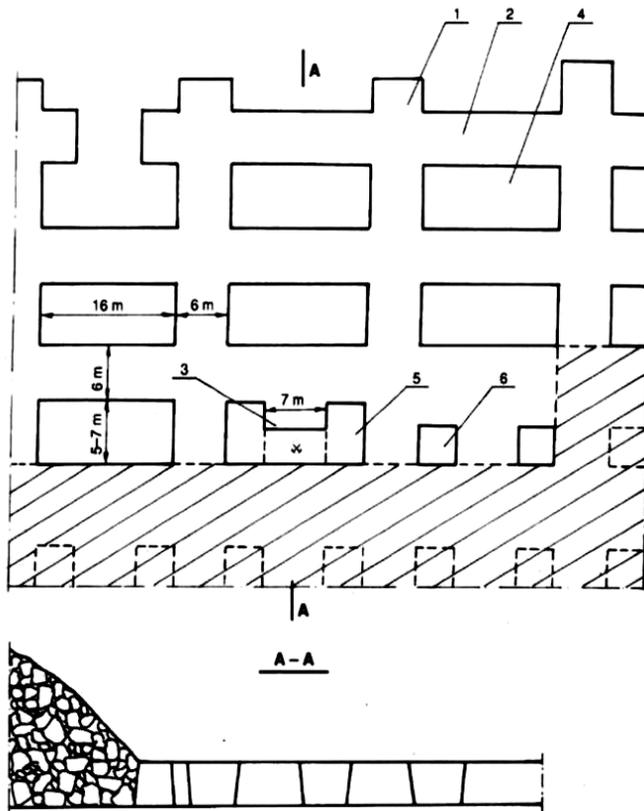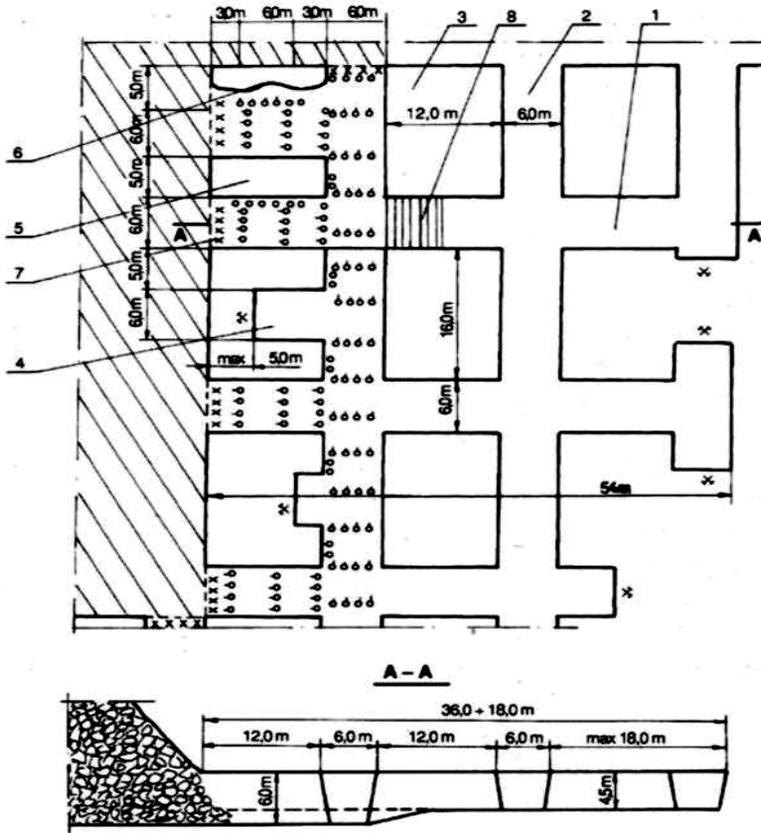


Fig.8.10. One-phase room-and-pillar method with roof caving (after Cieślik et al, 1991)
1. room; 2. strip; 3. cut into gob area; 4. pillar; 5. supporting (resistant) block (pillar); 6. ripped supporting pillar

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8.2.1.2 One-Phase Room-and-Pillar Method with Retarded Caving (Roof Deflection) (Fig.8.11)

In this method, the panel is developed by means of double entries (Fig.8.11). In deposits with thickness greater than 4.5 m, the entries are driven within the near-roof layer. The length of the mining face may reach 600 m. The ore is mined by means of seven-metre-wide rooms leaving yielding technological between-rooms rectangular pillars with dimensions 7 to 10m x 8 to 38 m. The longer axis of each pillar is perpendicular or parallel to the mining face line. The mining face is divided into three or two segments. In one segment, the rooms are mined leaving between-rooms pillars standing. While, within the second segment, the between-rooms pillars are ripped to get smaller (remnant) ones with residual dimensions. The ripping encompasses making seven-metre-wide cuts (insets) in the middle of the pillar towards the gob. Two smaller pillars with dimensions 7 to 10m x 2.5 to 4.5m are separated in this way. Next, the smaller pillars are ripped leaving the remaining ones with the near-roof area of 12 to 20 square metres. The deposit is mined with the use of blasting technique applied to short blastholes. The length of blastholes is from 2.5 to 3.0 m. The remnant pillars do not often need ripping because they break and chip under the pressure of deflecting (subsiding, gradually caving) roof. The self-mining of the pillars is supported by water spraying. Blastholes are driven with mobile drill carriages. Mobile blasting cars with a pneumatic charging device charge dry explosives. The mined ore is loaded with mobile loaders onto haulers, or loaders haul it to the discharging points onto conveyors or to the surge bunkers. The roof exposed during the extraction of the supporting (technological) pillars is protected by means of roof bolting. The adjacent pillar can be mined if the previous one has been completely extracted and the working area has been separated from the gob that has been just created. The liquidation increment is from 12 to 22 m. long. The mined out space is automatically liquidated due to the roof subsidence onto the residual pillars. Liquidation operations i.e., the technological pillars mining should be performed continuously from their beginning to end. The working face is ventilated with fresh airflow. Separate (local) ventilation (booster fans) is also used (Cieślik et al, 1991).
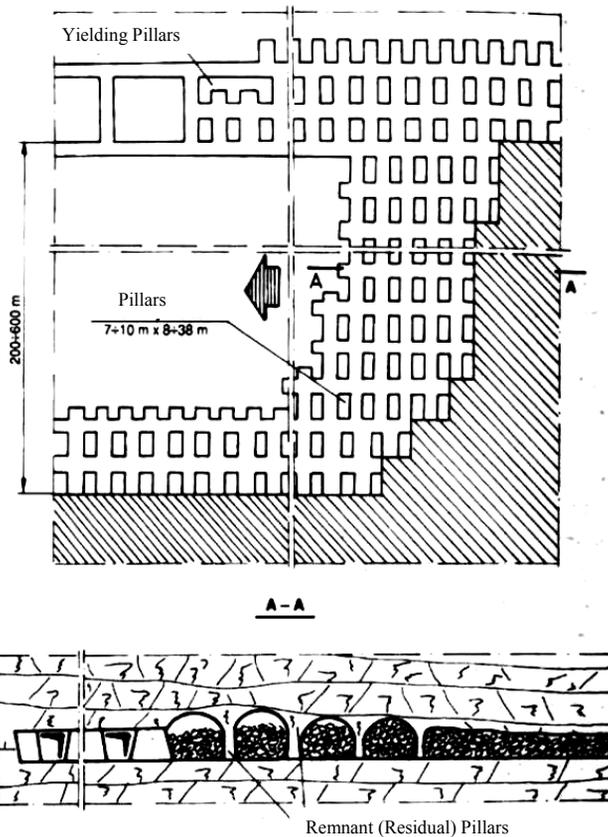
Fig.8.11. One-phase room-and-pillar method with retarded caving (roof deflection) (after Cieślik et al, 1991)

## 8.2.2 Room-and-Pillar Mining Methods with Roof Caving for Deposits with Thickness Greater than 5m

### 8.2.2.1 One-Phase Room-and-Pillar Method with Roof Caving (Fig.8.12)

In the panel separated (delineated) by means of roadways, strips and rooms with a width of 6m (measured at the roof) and the height of 4.5 to 5 m are mined creating three rows of pillars with dimensions 12 x 16 m. In the row of pillars located next to the gob, the seven-metre-wide cuts (insets) are made towards the gob area separating the smaller pillars with dimensions 12 x 16 m. Next, caving blastholes are drilled in the roof. After that, the bottom and pillars located in the row near the gob are ripped. The retaining partition is made in that way. The roof-caving increment is 18 m long (Fig.8.12) (Cieślik et al, 1991).

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Fig.8.12. One-phase room-and-pillar method with roof caving (after Cieślik et al, 1991)
1. room; 2. strip; 3. pillar; 4. room for caving; 5. supporting block; 6. insets (cuts); 7. retaining partition; 8. inclined drift


## 8.2.2.2 Two-Layer Room-and-Pillar Method with Roof Caving (Fig.8.13)

Advancing pillars with dimensions 31 x 57 m are separated in the near roof layer of the deposit. Within the main working front, these pillars are divided by means of the room system into pillars with dimensions 9x12 m and then into smaller ones with dimensions 6x12 m. In strips and rooms located near the caving area, the eighteen-metre-long caving roof holes are drilled. Having drilled the caving holes, one rips the lower ore layer and partially the technological (between-rooms) pillars. The mined ore coming from the lower layer is hauled along the descending inclines to the workings of the upper layer and then to recharging points. When the lower layer has been worked out, the roof caving is induced in the last row of rooms and pillars (Fig.8.13) (Cieślik et al, 1991).

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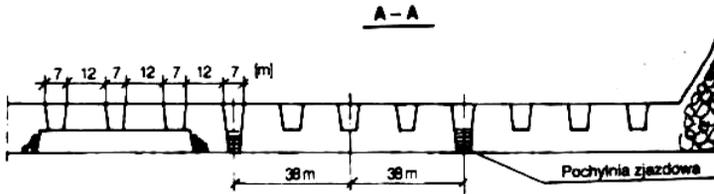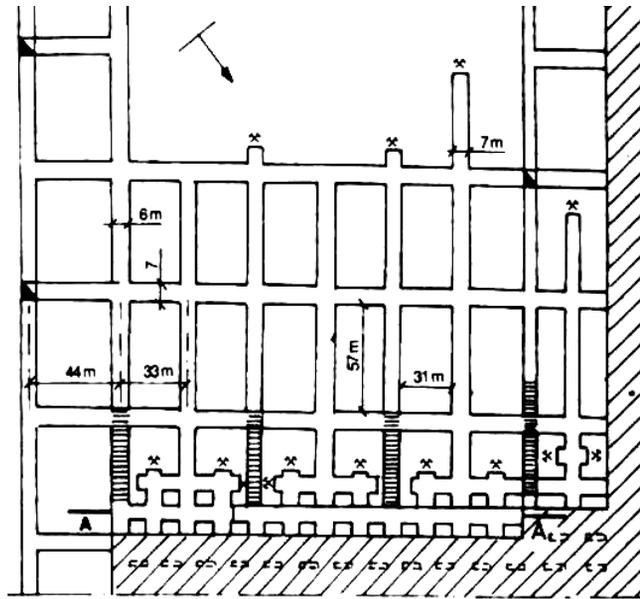
Fig.8.13. Two-layer room-and-pillar method with roof caving (after Cieślik et al, 1991)

### 8.2.3 Room-and-Pillar Mining Methods with Hydraulic Backfilling for Deposits with Thickness from 5 to 10m

#### 8.2.3.1 Two-Phase, Two-Layer Room-and-Pillar Method with Hydraulic Backfilling (Fig.8.14)

The mining of the thick deposit is carried out using the two-phase room-and-pillar method with two layers of hydraulic backfilling. Due to easy roof control in the dolomite-limestone rocks, the upper layer galleries are usually excavated in the first stage of mining. During this phase, the stiff large dimension pillars are cut. During the second phase, the pillars have been split before the lower layers can be mined. The mining face is divided into three blocks where the following operations are carried out in sequence: mining the upper layer, mining the lower layer, and backfilling the mined out space. Development works constitute driving the inclined drifts to divide the mining face into three eighty-metre-long segments (blocks) (Fig.8.14). In

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the first segment, the upper layer is mined, in the second one, the lower layer is excavated, and in the third, backfilling is carried out. Within one segment, firstly, the upper layer of the deposit is cut into pillars with dimensions 32 x 24m. Next, the pillars are cut into blocks with dimensions 12 x 8 m. In the next stage, the lower layer is mined. Then, the blocks are liquidated completely or partially. Deposit residues in the form of a cylinder with a roof diameter of six metres are left. When the roof conditions are difficult, the stowing increment is shortened from 32 to 16 m. In this case, the pillars with dimensions 32 x 24 m are cut into pillars with dimensions 32 x 8 m before they are ripped. Only the row of pillars located near stowing is ripped. When mining is completed, the worked out space is isolated with screens built in the near-roof layer and with one screen built approximately the inclined drift leading to the room. Then, the blocked space is backfilled. The pipe channels placed in the backfill carry away water to precipitation tanks (Cieślik et al, 1991).
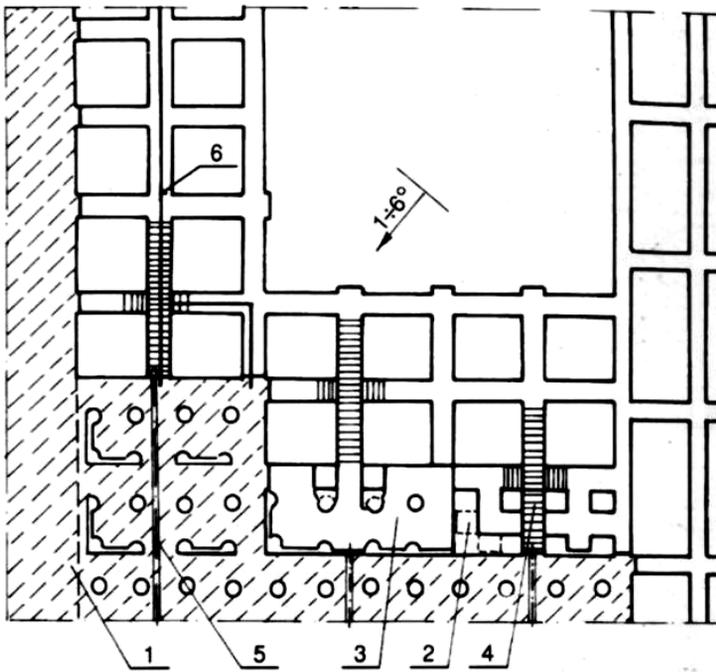


Fig.8.14. Two-phase, two-layer room-and-pillar method with hydraulic backfill (after Cieślik et al, 1991)
1. backfill; 2.upper layer mining; 3. lower layer mining ; 4. inclined drift; 5. dewatering collectors; 6. stowing pipeline

8.2.4 Room-and-Pillar Method with Hydraulic Backfilling for Deposits with Thickness Greater than 10m

8.2.4.1 Two-Layer Room-and-Pillar Method with Hydraulic Backfilling (Fig.8.15.)

The method is used to mine a deposit with a thickness of about 15 m. During development works in the near-roof (upper) layer, the mining panels with a width of 150 to 200 m and a

length of 800 m are separated by means of double entries and inclines driven (Fig.8.15). Mining is conducted to the rise within at least two panels. At first, the near-roof layer with a thickness of 4.5 m is excavated. In this layer, the pillars with dimensions 28 x 10.5 m are cut by means of room workings. The longer axis of the pillar is parallel to the goaf line. The width of the rooms, measured at the roof, is about 7 m. The walls of the rooms are inclined at 15° to the axis of the floor. Three rows of pillars are cut in advance. During development works in the lower layer, the inclined drifts (from the upper to lower layer) are made by means of ripping the incline floor within the upper layer. These inclined drifts can also begin in the room workings. The floor ripping is carried out continuously with the workings advance. The excavation of both layers is conducted at the same time with suitable advancing. In the upper layer, the last row of pillars is mined and the maximum width of the exposed roof is 17.5 m. The pillars are excavated in two directions, from the inclined drift to the middle of the panel. The excavation of the pillars is followed by the lower layer extraction. The distance between the two faces of the layers varies from 12 to 36 m. The lower layer is mined by means of long blastholes drilled from the upper layer. When both layers are worked out, the room has a height of 15 m, a length of 150 to 200 m, and a width, measured at the roof, of 17.5 m. Blasthole drilling, excavated ore hauling and loading are carried out with the use of mobile transport equipment (carriages and/or cars) (Cieślik et al, 1991).
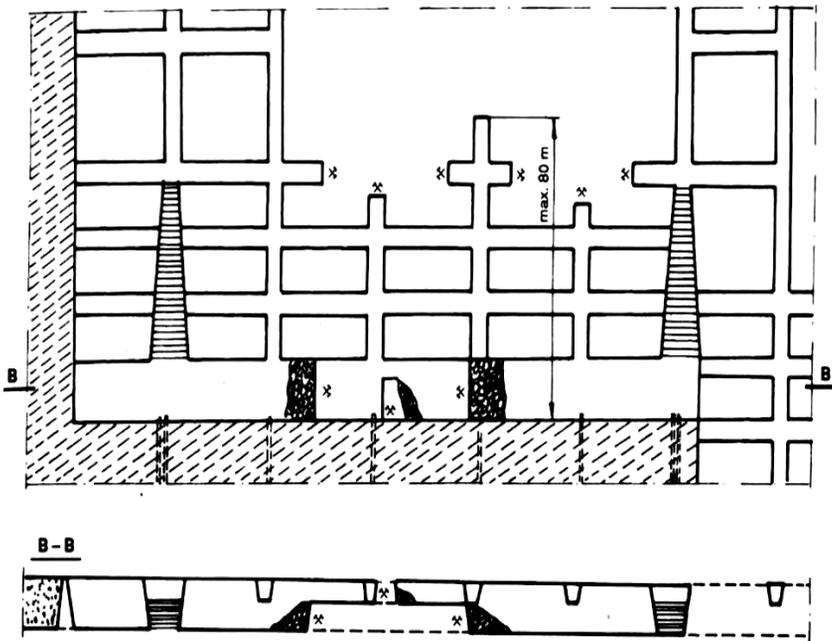


Fig.8.15. Two-layer room-and-pillar method with hydraulic backfilling (after Cieślik et al, 1991)

To support the roof of workings, one uses resin-bedded bolts with a length of 2.6 m arranged in a square net 1x1m. The first row of bolts is located 0.3 m from the wall. In the upper layer, the workings are of trapezoidal shape while the inclined drifts and rooms are oval (Fig.8.16 and Fig.8.17). The room in the lower layer has to be narrower because rocks scale and are loosened under great pressures. Therefore, the walls are given the shape of natural

talus of scaling rocks. Four rows of bolts with a length of 2.6 m are placed at the near roof layer level and are the additional protection of the roof. A wire mesh is under slung on the roof bolts. The mesh is also lowered onto the screen in the lower layer. The working front is ventilated with independent airflows. The room workings in the upper layer are ventilated with the separate ventilation until the crosscutting takes place (Cieślik et al, 1991).
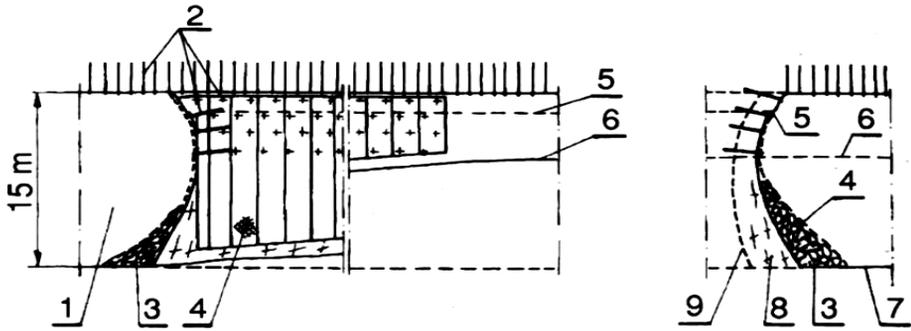


Fig.8.16. Supporting for roof and walls in high inclined drifts (after Cieślik et al, 1991)
1. inclined drift; 2. resin-bedded bolts; 3. loosened rocks; 4. wire mesh; 5. floor of hard rocks; 6. floor of the upper layer; 7. floor of the lower layer; 8. wall scaling zone; 9. boundary of wall scaling zone
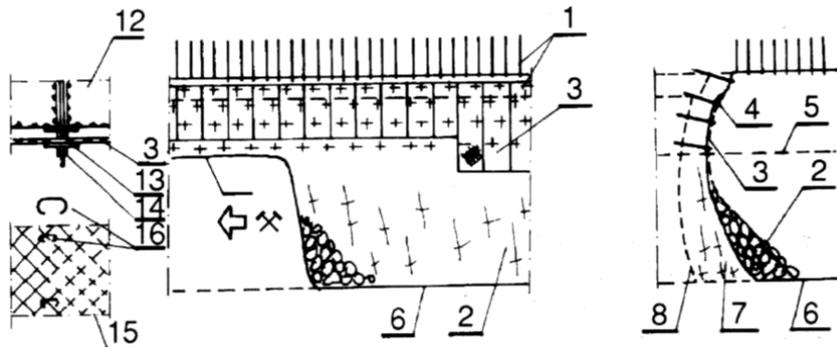


Fig.8.17. Supporting of roof and walls in high mining rooms (after Cieślik et al, 1991)
1. resin-bedded bolts; 2. screen; 3. wire mesh; 4. floor of hard rocks; 5. floor of upper layer; 6. floor of lower layer; 7. wall scaling zone; 8. wall cracking zone boundary; 12. mesh mounting; 13. pad; 14. nut; 15. mesh joining; 16. fastener for mesh elements

When the strip with a width of 17.5 m has been worked out, the mined out space (room) is liquidated by means of hydraulic backfilling (Fig.8.18). In the upper layer rooms, the screens are built. In the inclined drifts, the timber stoppings are built on the sand after filling the room to the level of the upper layer floor. Stowing water is carried away to precipitation tanks through pipe channels (with diameter of 500 mm) that are placed in the backfill at the level of the upper layer floor and the pipe channels placed onto the floor of the lower layer. The pipe

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channels are placed into each room of the upper layer. The hydraulic stowage pipeline is mounted in the room workings of the upper layer. The cycle of operations is repeated in the panel when the mined out space has been backfilled (Cieślik et al, 1991).
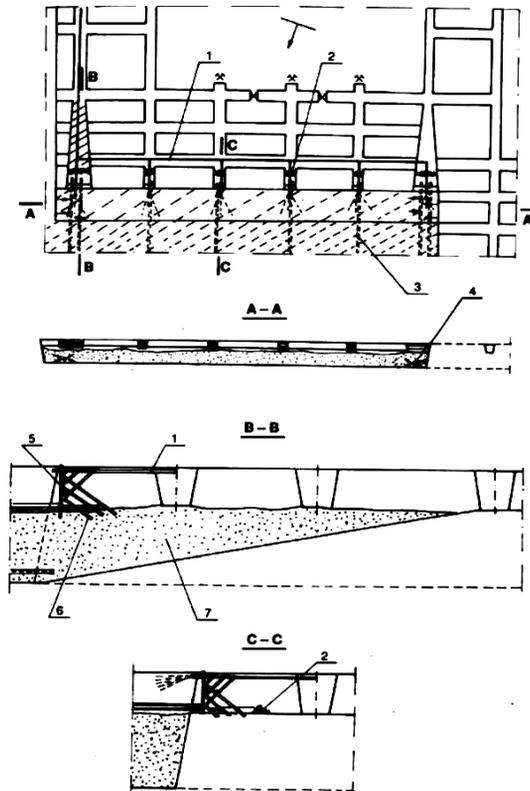


Fig.8.18. Mined-out area (goaf) backfilling (after Cieślik et al, 1991)
1. hydraulic stowage pipeline; 2.stoppings; 3. pipe channels; 4. filtration pipeline; 5. screen; 6. stretchers in sand floor on the floor bars; 7. backfilled inclined drift

## 8.3 Methods of Rock Breakage

In LGOM's mines, the copper ore is broken (excavated) with drilling-and-blasting technique. This mining method is used in development, opening and in exploitation works. The selection of mining technology depends on properties of rocks. Group blasting of working faces is commonly used (several or more faces blasted concurrently) (Janowski et al, 1996) to control the rockburst hazard.

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### 8.3.1 Explosives

Due to significant differences between mechanical properties of mineralized layer rocks and those of rocks surrounding the deposit, characteristics of explosive materials should be in harmony with workability of the hardest rocks in the area. Thus, the blasting agents ought to be of high density and energy concentration. Dynamites meet these requirements. Sensitivity of dynamites to external stimuli (possible to explode at the moment of being hit) and emission of harmful fumes in the course of charging the holes, make mechanical charging with the pneumatic charging device difficult to be used. This device needs less sensitive loose or liquid materials such as charged ammonium nitrate dry mixes, saletrol, or blasting emulsions (liquid or semi-liquid, or charged). Generally, blastholes are charged manually or with the pneumatic device (Janowski et al, 1996).

### 8.3.2 Initiation Systems

Explosive charges are fired in short blastholes with the 0.5-second-delay electric detonator. In long blastholes (longer than 6 m), the PETN (pentaerythritol tetranitrate-brisant explosive) detonating fuse is also used. The time-delay intervals can be from 0 to 10s or from 0 to 15s. The detonating fuse is placed into the blasthole along with an electric detonator (Janowski et al, 1996).

Non-electric delay systems are also employed. The shock tube system, a nonelectric delay system commonly called Nonel, is proving very successful. These units consist of plastic tubes containing a very light coating of explosive adhering to the inside. The detonation is of such low energy that the tube is not broken during firing. Because of the low activation energy, delays manufactured using Nonel are potentially much more accurate than cordline systems Janowski et al, 1996).

### 8.3.3 Electric Initiation

Rocks in the LGOM copper deposit are of high electric conductivity that makes it necessary to use low-resistance electrical firing circuits. The circuits consist of copper cables with big section diameter and have special connections of detonators within a working face. There are aerial connections (parallel and parallel-series) in which electric detonators (exploders) are connected to aerials made of a bare copper wire, and connections without aerials in which parallel groups of detonators' conductors are connected in series. (Fig.8.19). Electrical circuits and their elements have to be controlled before blasting. Continuity and resistance of them are checked. In copper mines, the average voltage of detonators (exploders) varies from 500 to 1000 V (Janowski et al, 1996).

### 8.3.4 Drilling and Blasting Operations

Blasting Operations. Blast carriages (Fig.8.20) are used to perform the following tasks: (1) Haulage of firemen team along with explosives from explosive storage to districts; (2) Armouring of high working faces when charged explosives are used; (3) Armouring of long roof holes and holes in faces charged with saletrol by means of the pneumatic charging device; (4) Haulage of the auxiliary equipment such as detonators, cables, testing device, and short-circuiting switches.
Fig.8.20 shows operational platform of a blast carriage connected with a haulage tractor (Janowski et al, 1996).
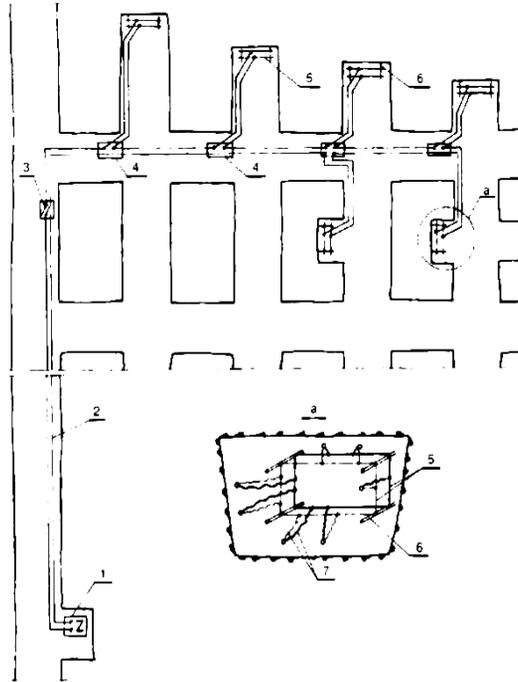
Fig.8.19. Electrical circuits for group firing of faces with parallel-series connection of electric detonators (after Janowski et al, 1996)

a – parallel aerial connection within a face; 1. electric detonator (exploder); 2. main firing line; 3; short-circuiting switch; 4. connection box; 5. aerials made of bare copper wire; 6. wooden pin to fix an aerial; 7. conduits of detonator
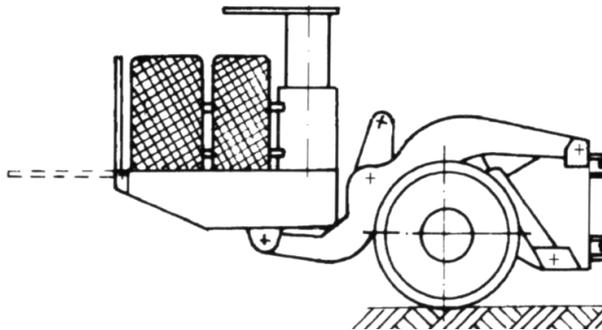


Fig.8.20. Blast carriage operational platform (after Janowski et al, 1996)

Drilling Operations. Drilling operations are carried out with the use of drill carriages equipped with electric-hydraulic drive for drills (Fig.8.21). The drills have one boom because of the size of mining faces (cross-section is of 20m$^2$) and pulls (3 m). Hole diameters 41, 45, or 64 mm is used depending on the diameters of explosive cartridges. The drill carriage consists of a haulage tractor and drilling platform. It is equipped with a diesel engine. The hydraulic rotary-percussion drill is mounted on the frame of the telescopic arm (Janowski et al, 1996).
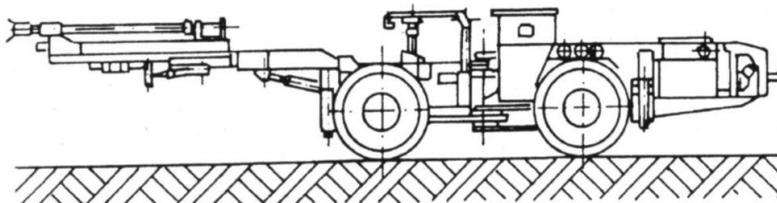


Fig.8.21. Drill carriage (after Karty Katalogowe Boart Lena, 1995)

Blasthole Arrangement. The most important part of a drill is a vertical wedge cut (Fig.8.22). Other types of the cut are: a fan cut, wedge near floor cut, and wedge near roof cut. The wedge cut constrains a pull to 3.0 m. A longer pull, even 6 m long, can be obtained using parallel cuts if the drill can elongate its stem. The blasting pattern for elongated pulls is presented in Fig.8.23 (Janowski et al, 1996).

Parallel cuts (breaking-in holes) can be produced in two ways: (1) As a system of parallel holes located at very small spacing between adjacent holes, with the same diameter, alternately empty and charged with explosive materials; (2) As a system of holes with a large-diameter ($\phi$ 90-130mm or more) central hole not charged with explosive materials and cut holes with normal-size diameters, concentrated at very short gradually increasing distances from the central hole (Janowski et al, 1996).
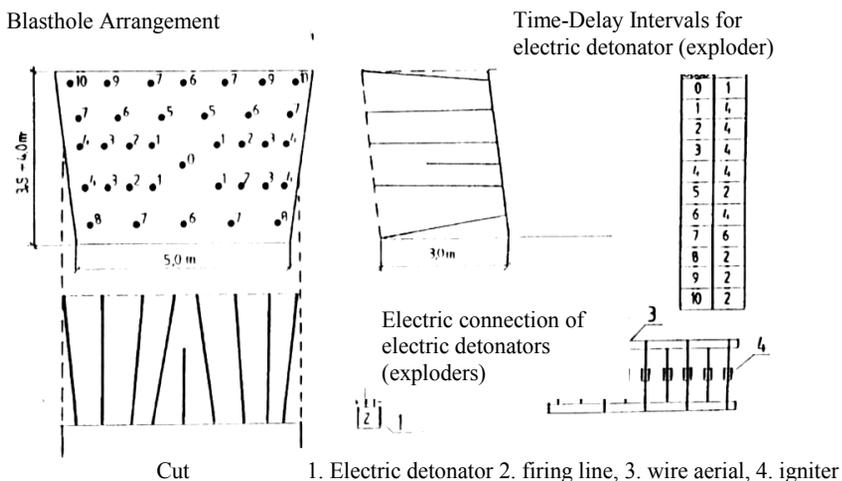


Fig.8.22. Blasting pattern for blastholes and cut holes (breaking in hole) (a) time-delay intervals (b) connections for electric detonator (exploder) (c) (after Janowski et al, 1996)
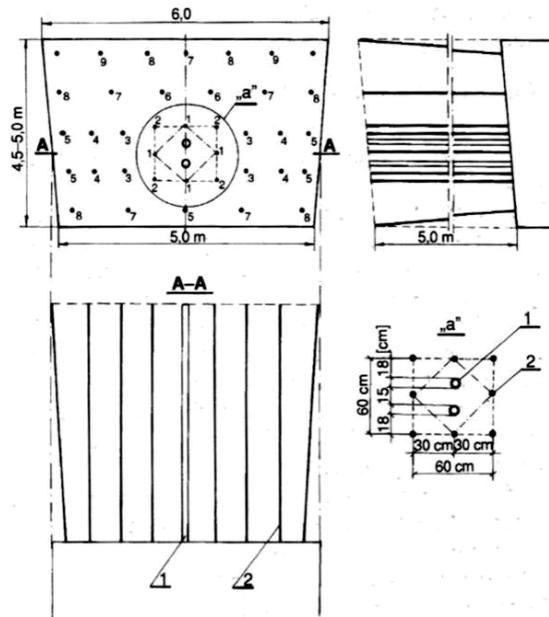
Fig.8.23. Drilling pattern of a face and distribution of igniters with the use of a cut with parallel holes (to obtain enlarged pulls) (after Janowski et al, 1996)
a – part of a cut with parallel holes; 1. empty cut holes, $\phi$ 76-90 mm; 2. cut holes charged with explosive materials, $\phi$ 64 mm

## 8.4 Materials Handling: Loading, Haulage, Hoisting (Horizontal and Vertical Transport)

Loading and haulage of excavated copper ore are next to mining and supporting the prime mining operations. Mining methods used in LGOM's mines enabled one to introduce mechanized systems based on mobile mining machines equipped with fuel engines which facilitated mechanization of mining operations (Gładysiewicz and Pelc, 1996).

The transport system encompasses machines, devices, and equipment for hauling excavated material from mining faces to processing plants. Bunkers, auxiliary equipment, devices for ore crushing (breaking up), as well as equipment for reloading materials from one means of transport to the other one also belong to the transport system. The horizontal transport uses means of the cyclic transport (mobile machines, underground railway) as well as means of the highly effective continuous transport belt conveyor). The load-haul-dump (LHD) machines are also employed (Gładysiewicz and Pelc, 1996).

The mine transport operation is carried out by means of overhead loaders (Fig.8.24), which on short haulage roads can work as load-haul-dump machines (LHD). The overhead loader consists of the loading platform, a haul tractor, and a driving axle. The driving system embraces the diesel fuel engine, hydraulic gearbox and two driving axles. The loader loads the excavated material from the face onto the hauler (Fig.8.25). The hauler consists of the haul tractor and open load-carrying box. The diesel fuel engine, hydraulic gearbox, and driving axle make the driving system of the machine (Gładysiewicz and Pelc, 1996).
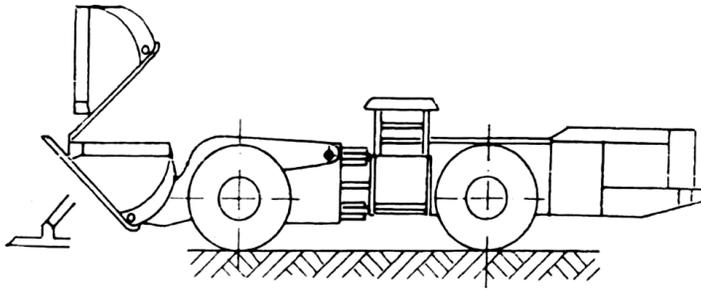
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Fig.8.24. Schematic diagram of a mining loader (LHD) (after Karty Katalogowe ZM Legmet, 1995)
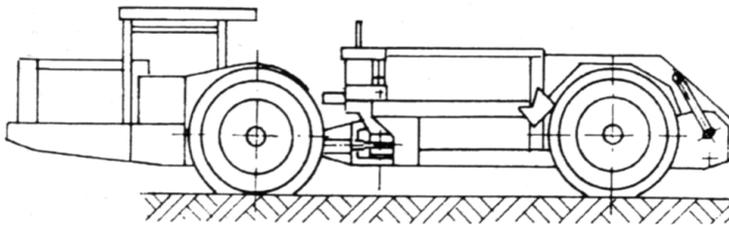


Fig.8.25. Schematic diagram of a hauler (after Karty Katalogowe ZM Legmet, 1995)

The haulers haul the excavated material from the face to the mine discharging (reloading) points (Fig.8.26), where the ore is directly discharged onto the grate serving as a screen classifier. Oversized rocks left on the grate are crushed (broken up) mechanically with hydraulic hammers. The grate is directly situated above the bunker from which the mined ore is fed onto the belt conveyor by means of the chute. Several discharge points with a grate may be installed on the single belt conveyor route. Streams of excavated ore flowing out of the mining districts are directed to the district bunkers made in the form of a pit-hole (Fig.8.27). Each mine district may have a single bunker or two, or several districts may have one common bunker. The main underground transport from districts to the pit-bottom bunkers (drawing shaft) may be performed by means of the underground rail transport. In this case, it is necessary to use loading points for mine cars at the contact of the belt conveyor and rail transport, and discharging points in the form of a mine-car tippler in the pit-bottom area (Fig.8.28). It is also possible to employ the combination of belt conveyor transport and rail transport. The pit-bottom surge bunkers play an essential role in the transport systems as they allow the districts and shaft devices to work independently of the main underground transport. Fig.8.29 shows the schematic diagram of main horizontal underground transport (Gładysiewicz and Pelc, 1996).

The vertical (hoisting, shaft) transport uses the skip winding plants. The winders are built on the head frames of steel construction. The head frame has four hoisting ropes and two motors (engines). The motors are direct current powered with automatic control. The skips of the winding plants are loaded from the vertical bunkers mounted on the shaft. Each skip has a co-operating bunker loaded by means of the belt conveyor during the skip riding. The ore is delivered to the skip winding plant that delivers the ore to the drawing shaft. The landing of the drawing shaft is equipped with devices and machines for discharging and collecting the ore from the skips (Gładysiewicz and Pelc, 1996).
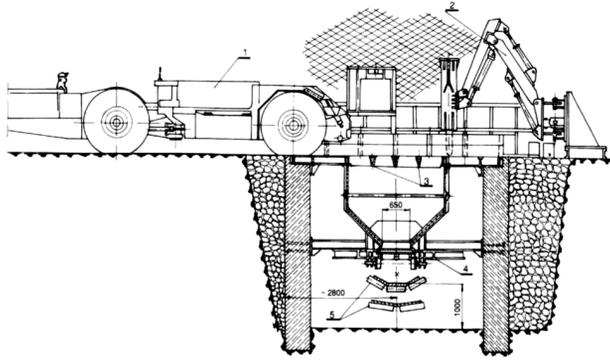
125

Fig.8.26. Mine discharging point (after Gładysiewicz and Pelc, 1996)
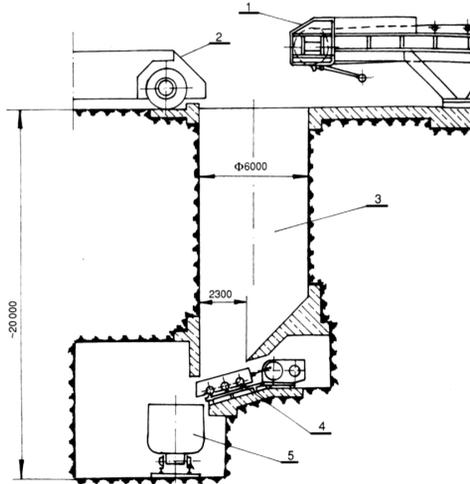1. hauler; 2. hydraulic hammer; 3. discharge grate; 4. chute; 5. belt conveyor



Fig.8.27. Surge bunker at loading point for mine rail cars (after Gładysiewicz and Pelc, 1996)
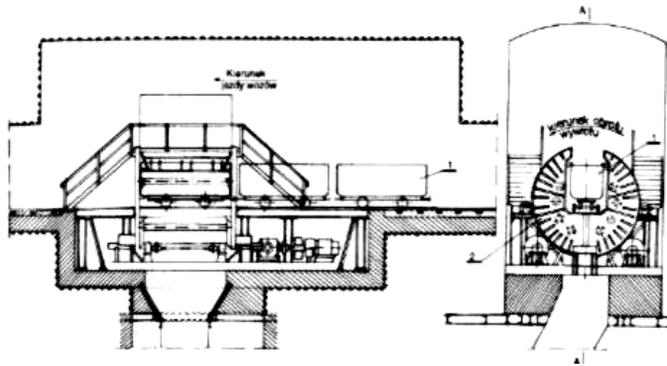1. belt conveyor; 2. hauler; 3. pit-hole as a bunker; 4. mine car feeder; 5. mine car

Fig.8.28. Discharging point with mine-car tippler (after Gładysiewicz and Pelc, 1996)
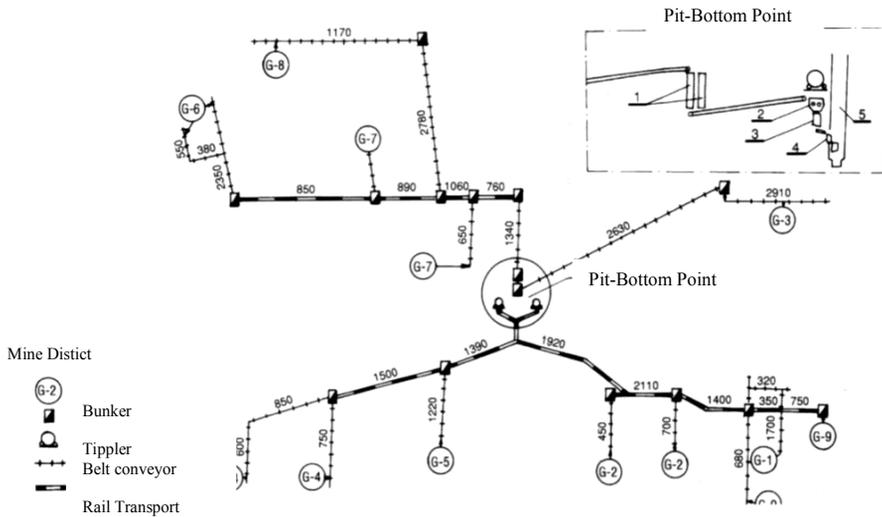1. mine car; 2. mine-car tippler



Fig.8.29. Transport of excavated ore with a combination of belt conveyor and rail transport (after Gładysiewicz and Pelc, 1996)

8.5 Ventilation of Workings

Ventilation of the Polish copper mines faces very serious technical and logistic problems. They are caused by a number of site-specific factors. Mining takes place at great depths, geothermal conditions of the deposit are unfavourable, mining methods need driving and maintaining a large number of mine workings and diesel powered equipment is widely used. All this creates a serious challenge to the task of assuring proper ventilation and provision of acceptable climatic conditions in many mining areas. In copper mines there are hard climatic conditions, toxic gases and dust as well as fire hazard (Wacławik et al, 1996).

8.5.1 Ventilation Network Structure

The structure of mine openings, access roads, and development workings is typical of mines excavating seam deposits. One seam nature of the deposit, its great depth and relatively small dip of the seam have influenced the structure of the workings. Moreover, the cost considerations had limited the number of shafts sunk to access the deposit. The number of shafts is now too small compared to the size of the mining area. One shaft services 4 to 10 km$^2$ of the deposit area. In Polish copper mines, there is a complex network of development and access roadways, as well as workings of room-and-pillar mining method. Each working acts as an airflow channel. Such a system of workings strongly influences the structure of ventilation network in each mine. The climate and toxic gas hazards, which are present at the mines, strongly influence the amount and distribution of air. The toxic gas hazard is related to 2000 mobile machines using diesel fuel engines and operating in roadways and panels. Air that flows through the mine and through each panel remains in contact with large surfaces of exposed in-situ rock for the extensive period. Thus, there are conditions for conductive heating that results in the significant increase of temperature (Wacławik et al, 1996).

The great number of workings makes their ventilation difficult. Although the airflow resistance of the workings is small, the airflow is distributed into a great number of workings in mining panels as well as on the fresh airways. The advancing mining causes significant air leakage and re-circulation through the gobs (Wacławik et al, 1996).

The Lubin mine is accessed with seven shafts: 4 downcast shafts and 3 upcast shafts. The shaft's diameter is from 7.5 to 6 m. The total airflow through the mine is 1300m$^3$/s of fresh air. At each upcast shaft there are three central ventilation fans installed.

The Polkowice mine is accessed with seven shafts: 4 downcast shafts and 3 upcast shafts. At each upcast shaft, three central ventilation fans are installed. The total airflow through the mine is 1200m$^3$/s of fresh air.

The Rudna mine is accessed with ten shafts: 7 downcast shafts and 3 upcast shafts. The total air- flow of fresh air is 3000 m$^3$/s. At each upcast shaft, there are three central ventilation fans.

The Sieroszowice mine has two shafts: one downcast shaft and one upcast shaft. The total airflow of fresh air is 850 m$^3$/s. At the upcast shaft, there are six central ventilation fans. Fig.8.30 shows the simplified scheme of ventilation of the KGHM's mines.

The central ventilation fans are connected in parallel to assure adequate volume of air and to facilitate delivery of air at small airflow resistance in the side drifts of the ventilation network. The central fans work normally in a sucking mode. At every fan station, there are fan drifts and reverse trap doors allowing for reversing a part of air in the shaft to reverse the direction of air flow within the whole mine (Wacławik et al, 1996). Fig.8.31 shows the fan station of main ventilation in the Sieroszowice mine.

To improve climatic conditions and to intensify ventilation in panels (locally), one can use the separate ventilation such as ventilating ducts (air ducts), which are the underground

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electric fans. They usually operate at inlets in to the vented area as well as in places where excavation takes place. They are also installed in the fresh or used air flows in order to increase the amount of the flowing air in the specific air split of the network (Wacławik et al, 1996).
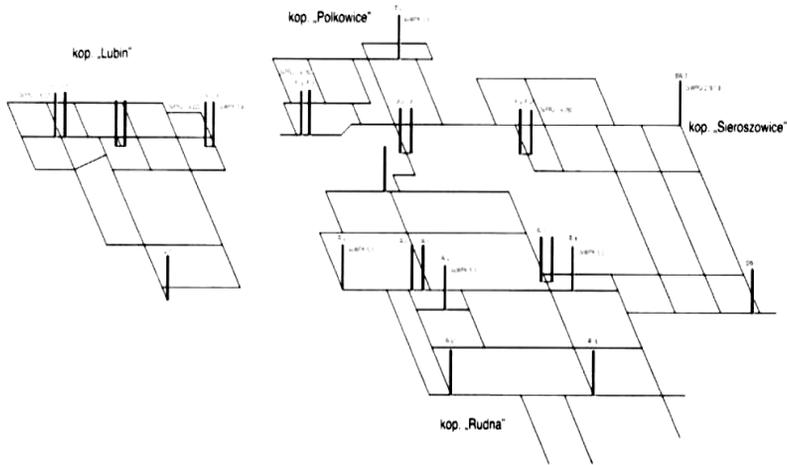


Fig.8.30. Spatial lay-out of the main ventilation system (after Wacławik et al, 1996)



Fig.8.31. Fan station of central ventilation, Sieroszowice mine (after Wacławik et al, 1996)

8.5.2 Roadway Ventilation

Development (access, opening) operations give at least 30% of excavated ore. During these operations, double parallel entries are driven; fresh air flows through one entry, while

used air flows through the rest ones. Mobile machines with diesel engines move in the airflow of used air. Break-throughs that join drifts with fresh air, and those with used air are stopped. The last break-through closest to the goaf is not isolated. The stoppings built within the roadways do not have to be completely tight because they are not under the high pressure. Blind heads (faces) are ventilated with ventilating ducts or with an air duct (electric underground fans), which should blow the air towards the road head to get the airflow moving at 0.3 m/s speed in the working where diesel engines operate. The ventilating ducts are effective in blind working shorter than 30m. One should use one duct at every head (face) (Fig.8.32) (Wacławik et al, 1996).

The copper mines have very complex and substantial ventilation network system. One-level-regulation of air distribution is not effective. To get the desired proportion of airflow division, doors in fan drift should be used or some operation should be done using central fans. In the next stage of airflow regulation, fans are used in airflows of fresh air. The next stages of air regulation are: application of fans to the dams, regulation stopping, and fans. The purpose of the regulation is to lessen air leakage, to get desired airflow parameters, and to direct the to the places where people work. Mining face is always ventilated by fresh airflow. Sometimes ducts are used to intensify ventilating locally (Wacławik et al, 1996).
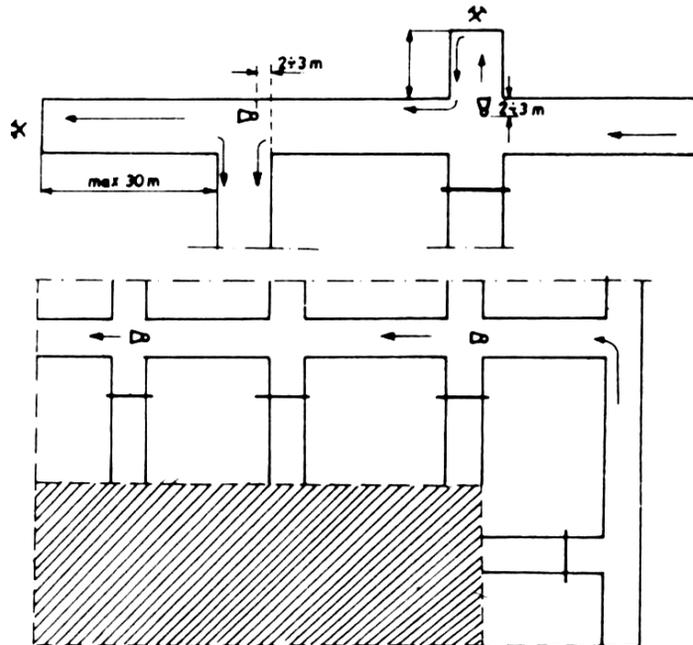


Fig.8.32. Ventilation of roadways by means of ventilating ducts (air ducts) (after Wacławik et al, 1996)

8.5.3 Air Stoppings (Air Dams)

In LGOM's mines air stoppings are of regulating, separating, and insulating type. Airflows are regulated by means of different types of air stoppings (Wacławik et al, 1996).

Regulating air stoppings. They throttle the airflow to lessen amount of air in the workings where they are built. They are made of bricks, concrete with a door (Fig.8.33).

Separating air stoppings. They should separate airflow of fresh air from the airflow of used air. They should be tight enough. They can be built in form of wall.

Insulating air stoppings. They should insulate old workings from the rest. They are usually equipped with a device (a tube) to control the air behind.

Workings with belt conveyor transport may also be blocked. To lessen air leakage, some kind of stopping can be placed near the point of ore recharge. Fig.8.34 shows the idea of blocking the air at the recharging point.
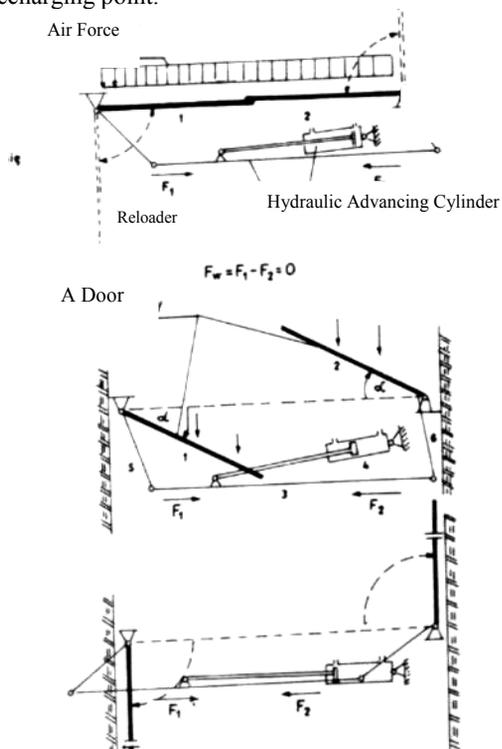
Air Force

Hydraulic Advancing Cylinder

Reloader

$F_w = F_1 - F_2 = 0$

A Door

Fig.8.33. Reloading air dam (stopping) (after Wacławik et al, 1996)
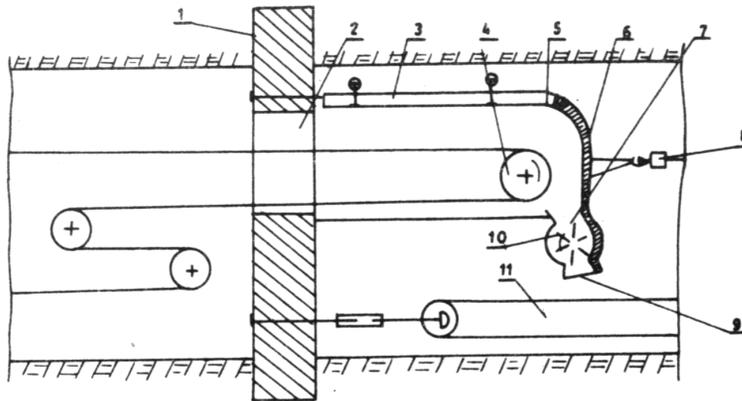
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Fig.8.34. An air stopping over the belt conveyor (Wacławik et al, 1996)
1. stopping; 2. window; 3. hermetic shield; 4. conveyor; 5. joints; 6. front wall; 7. trough; 8. counterweight; 9. packing; 10. closure; 11. conveyor

8.6 Supporting of Workings with Special Attention to Roof Bolting

At the beginning of exploitation, development, exploration, opening, and excavation workings were protected by means of standing supports in the form of late bearing arches and props. Introduction of heavy machines and room-and-pillars mining methods made it necessary to use roof bolting method to protect and control the roof and walls. The main purpose of bolting is to strengthen the rocks surrounding the mining working by means of artificial creation of compressive stress in the rock mass and taking tensile stress by bolts. The increase of the rock strength obtained in this way, should be big enough for the rock to keep up its structure and to take over stresses existing around the working (Janowski et al, 1996).

8.6.1 Types of Supports in Mining Openings

8.6.1.1 Roof Bolting

There are four types of roof bolts being used: (1) Wedge-and-sleeve; (2) Resin-bedded; (3) Rope bolts with cement charges; and (4) Abrasive (Janowski et al, 1996).

1. Wedge-and-Sleeve Bolts. They are used in hard rocks with medium and high mechanical properties. The rod of the bolt is fixed in the hole by means of the bolt's head expanding sleeves against the hole walls with the wedge pulling in. The rods are the resistant (strengthening) elements. The rod's diameter varies from 16 to 20 mm.

There are two kinds of wedge-and-sleeve bolts: (1) Bolts with load-bearing capacity of 100 kN with elements (Fig.8.35): a rod with a length of 1,6 m i 1,8 m, expanding wedge head and resistant washer; (2) Bolts with load-bearing capacity of 150 kN with elements (Fig.8.36): a rod with a length of 1,8-2,6 m, expanding 2-wedge head and resistant washer
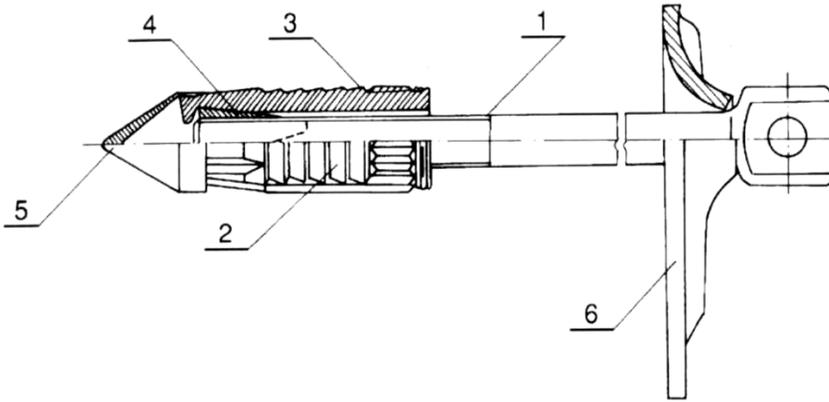
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Fig.8.35. Wedge-and-sleeve bolt with load-bearing capacity of 100 kN (after Janowski et al, 1996)
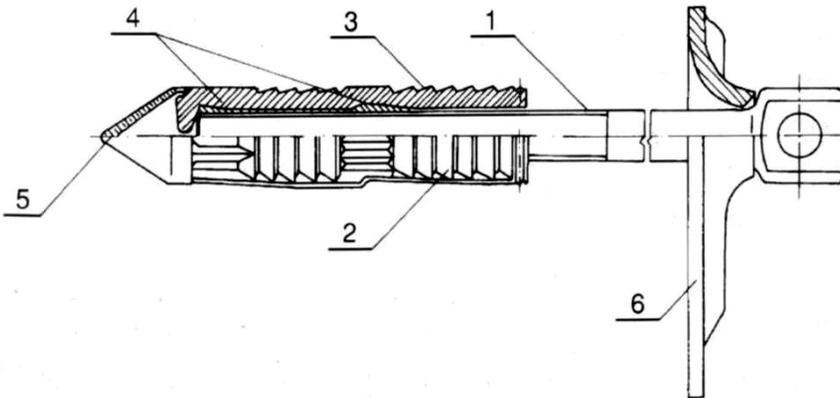1.rod; 2. head; 3. jaw; 4. expanding wedge; 5. cone cap; 6. resistant washer



Fig.8.36. Wedge-and-sleeve bolt with load-bearing capacity of 150 kN (after Janowski et al, 1996)
1. rod; 2. head; 3. jaw; 4. expanding wedge; 5. cone cap; 6. resistant washer

2. Resin-Bedded Roof Bolt of RM Type. It is used in weak rocks, fissured, fractured, in faulted zones, or in long-lived workings mined in hard rock. Resistant element: steel rods with a diameter of 18 mm and a length from 1.6 to 2.6 m. Elements of this roof bolt are presented in Fig.8.37.
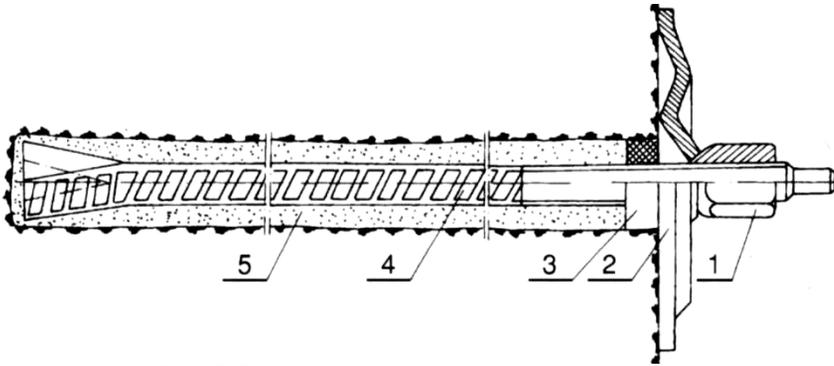
Fig.8.37. Resin-bedded roof bolt RM *(after Janowski et al, 1996)
1. nut M20; 2. washer; 3. rubber packing; 4rod RM; 5. resin

3. Rope Bolts with Cement Charges. They are used in week rocks, fissured, and fractured faulted zones in long-lived workings. Elements similar to those of resin-bedded. Cement is placed into the hole in form of a charge of semi-liquid. The length of a rod is from 1.6 to 2.6. (Fig.8.38)
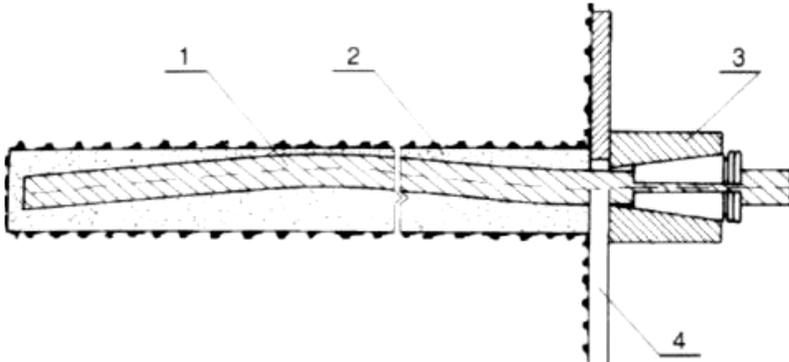


Fig.8.38. Rope bolt with cement (after Janowski et al, 1996)
1.rope rod; 2. cement; 3. clamp; 4. washer

4. Abrasive Bolts. Tube-abrasive bolts are used to support weak rocks or faulted mining roofs (Fig.8.39). The elements of this bolt are: rod with a length of 1.8 to 2.0 m in the form of tube and a resistant washer.
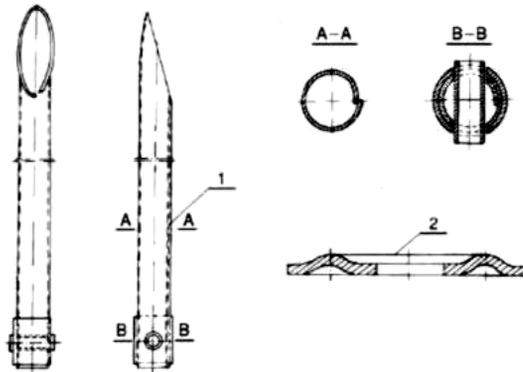
Fig.8.39. Tube-abrasive bolt (after Janowski et al, 1996)

## 8.6.1.2 Standing Support

In the panel of the mine where there is the possibility of roof caving and rock falling, the standing support is used. This support is a kind of auxiliary device for roof bolting. This support in form of a single hydraulic prop or (timber) breaking prop is a yielding support (Janowski et al, 1996).

## 8.6.1.3 Technique of Roof Bolting

Roof bolting is mechanized. Mobile bolters (carriages) are used to support workings. These cars are employed in workings with height of 2.3 to 6.0m. The holes diameters are 38 mm and they are drilled with percussion or rotary-percussion drilling rigs. The bolters are equipped with special pneumatic device to shoot resin charges (Fig.8.40). Mobile drill-bolt carriages are used to drill holes for bolts and for blasting (Fig.8.41). The holes are drilled in the roof, in walls and in the floor (Janowski et al, 1996).
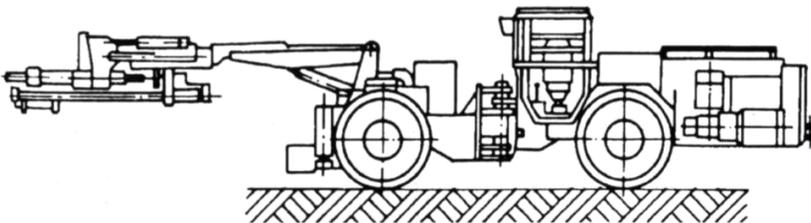


Fig.8.40. Mobile bolter (after Karty Katalogowe Boart Lena, 1995)
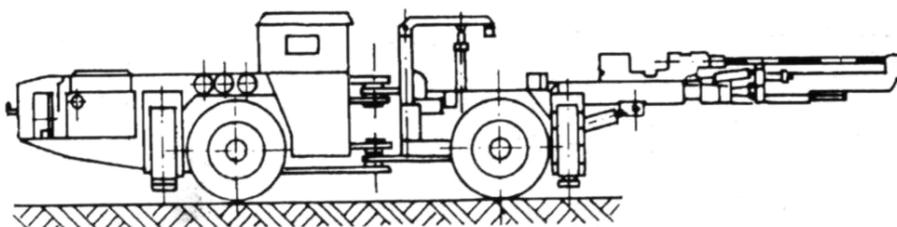
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Fig.8.41. Mobile drill-and-bolt carriage (after Karty Katalogowe Boart Lena, 1995)

### 8.6.2 Protection of Roof and Walls in Excavation Workings

The selection of the bolting type is based on mechanical properties of rocks, tectonic conditions (faults), the usage of the working, the size of the working, and the class of the roof. The class of the roof is established using a value of the roof stability coefficient, which shows the relation among the rock strength, the bank thickness, and the fissility degree (Janowski et al, 1996).

For any class of the roof, the length of the roof bolts in the workings with a height of 10 m and a width of 12 m should be at least 1.6 m, while in the workings with a height more than 10m and a width of 12 to 25 m, not less than 2.6m. When the roof is of class I and the compression strength of the layer being bolted is not less than 15 MPa, the holes for bolts should be arranged in a net pattern with the spacing dimensions 1.0mx1.0m. When the roof is of class II or class III and the compression strength of the roof rocks is less than 100Mpa, the bolts should be arranged with the spacing dimensions 1.5mx1.5m. When the roof is of class IV and the compression strength of the roof rocks is not less than 80 Mpa, the bolts should be arranged with the spacing dimensions 2.0mx2.0m. In high mining workings, the walls should be bolted. For workings with a width less than 10 m, the length of wall bolts is not less than 1.6m. While for workings with a width more than 10 m, the length of the wall bolts is not less than 2.6m (Janowski et al, 1996).

### 8.6.3 Supporting of Horizontal Opening and Development Workings

Roof bolting is the main method for supporting horizontal opening and development workings. The standing support is only used in faulted parts of the deposit or in panels and roadways where the rocks are under the increasing pressure of the rock mass. Resin-bedded roof bolts, wedge-and-sleeve bolts, cement bolts, or rope bolts are employed in roadways. The roofs of any classes are always supported with bolts with the length of 1.6m. When the roof is of class II or I, bolts are arranged in a net pattern with spacing dimensions 1.0mx1.0m. When the roof is of class III or IV, bolts need to be located with spacing dimensions 1.5x1.5m. The walls are bolted when the width of the roadway is greater than 3.5m. In the opening roadways and locally in development ones, lagging is used for roof and walls with a wire mesh or concrete spray is employed. Different types of support are also used in roadway. They are as follows: the brick lining, concrete block support, prop-and-bar system, arch support, prop-stall, props, and chock-type support (Janowski et al, 1996).

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8.7 Dewatering of Underground Workings

Dewatering of underground workings is described using the examples of three copper mines, which belong to KGHM Polska Miedź S.A. In underground workings, technological water and water from the rock mass is removed from the face with pumps when gravitational method is not possible to be applied.

8.7.1 Lubin Mine

Groundwater from the rock mass and water from the backfilling (about 30% inflow) mainly form the mine-water inflow. The inflows are led through the pipeline to the district or main pump chambers. The main mine drainage (main mine pumping) system embraces four water drainage installations located at shafts: L-I-L-II, L-III-L-VII, L-IV-L-V and at the shaft L-VI. In the shaft L-VI from the level 810, water is pumped with pipelines to the pump chamber at the shafts L-IV-L-V on the level 640. From here, waters are carried away through shaft pipelines to the level 670 and next, through pipelines to the pump chamber at the shafts L-I-L-II on the level 610. There is one pipeline in each shaft: L-IV and L-V. From the level 610 at the shafts, L-I-L-II the water is pumped to the surface through two pipelines. In the shaft L-I, there is a pipeline with the diameter of 400mm and in the shaft L-II one with the diameter of 500 mm. Waters from the rock mass and backfilling inflow to the pump chamber located on the level 610 near the shafts L-III-L-VII. From the area of the shafts, L-III-L-VII (level 610) the water is pumped to the surface through two pipelines with the diameter 400 mm located in the shaft L-VII, which will stand for one of the pipelines in the shaft L-III (Duda and Adamek, 1996).

8.7.2 Polkowice Mine

As much as 90% of the mine water inflow flows to the pump chamber at the main shaft and to the pump chamber at the eastern shafts only 10 % of the main inflow come. From the western region, waters flow off gravitationally to the pump chamber at the main shafts. The main mine drainage system embraces two two-level systems for dewatering: one at the main shafts and the second at the eastern ones. There is also the third system at the western shafts, but it is not taken into account in the main drainage system because of little inflow (Duda and Adamek, 1996).

The drainage system at the eastern shafts consists of three water drainage installations, one on level 810m, the second on the level 740 m and the third on the level 680 m. From the level 240m, the water is pumped by means of two pipelines with the diameter 400 mm, one pipeline in one shaft. From the level 740m, the water is pumped through the same shafts to the level 680m. From the level 680m, the water is pumped to the surface by means of two pipelines with the diameter 500mm (Duda and Adamek, 1996).

8.7.3 Rudna Mine

The mine-water inflow consists of ground waters, the water from backfilling and mine water of the Sieroszowice mine.

The main mine drainage system is based on the main pump chamber located at the level 1000m in the area of the Rudna shafts and on two regional pump chambers. One of them is located at the Rudna shafts (western field) and the second is located at level 1000m near the shaft R-VII (Fig.8.42 and Fig.8.43). District chambers and sumps or drainage levels supply the pump chambers (Duda and Adamek, 1996).
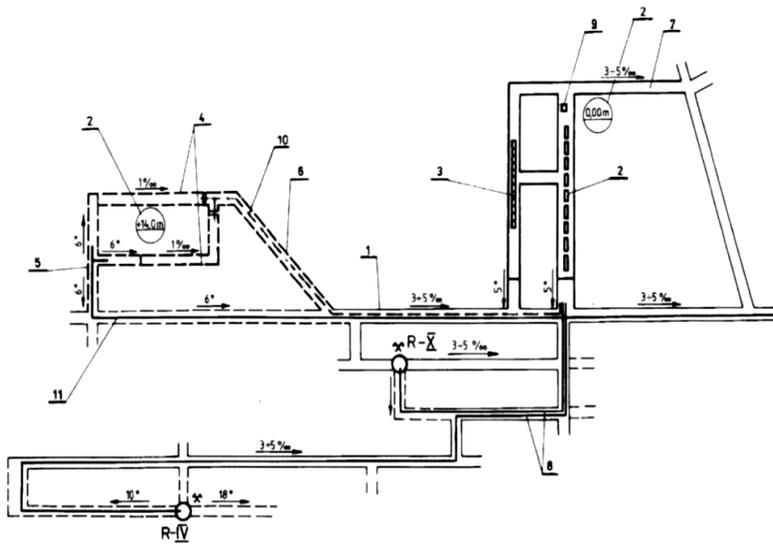
Fig.8.42. Main mine drainage system in Rudna mine (after Duda and Adamek, 1996)
1. roadway; 2. main drainage pump chamber, pumping the water directly onto the surface; pumping height 980 m; flow height 14,0m; 3switching station; 4. water tanks; 5. delivering water from deep parts of mine; 6. pipelines; 7. air roadway; 8. pumping pipelines to shafts; 9. cooling water; 10. pipeline; 11. pipeline
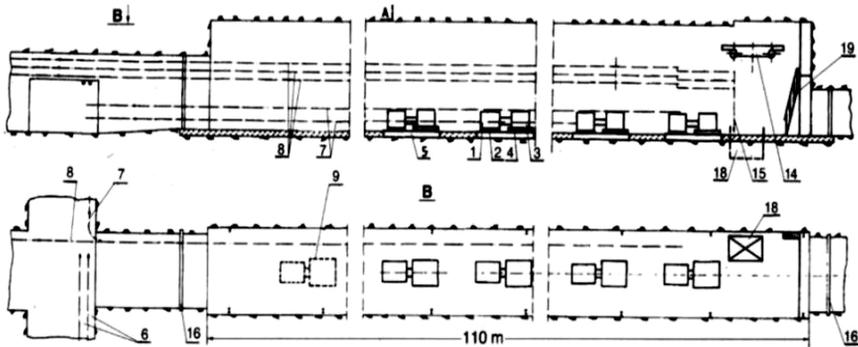


Fig.8.43. Main drainage chamber Rudna Główna – level 1000 (after Duda and Adamek, 1996)
1. pumping engine; 2. pump foundation; 3. engine; 4. engine foundation; 5. pumping engine foundation; 6. pipelines from lower levels; 7. pipelines; 8. pipelines; 9. spare position; 10. cable channel; 11. sewer; 12. pipeline support; 13. supports; 14. support 15. pipelines; 16. pump chamber locking; 17. roof bolting for pump chamber; 18. pumping installation 19. steps

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8.8 Power Supply and Communication

Power Supply. The equipment working in mining districts are electric energy powered. Moreover, mining machines use fuel engines. The panels are supplied with electric energy by the district switching stations (6 kV). The district switching stations are supplied with electric energy by means of transformers (500 V) from the regional and main underground stations. The power receivers at the mining faces are supplied with a voltage of 500V from the network switching stations through the transformer station (600/500). High and low, voltage cables are used to deliver the electrical energy. Thermal relays and fuses are used as an overload and over voltage protection. Protective grounding is employed as a part of fire protection (Piestrzyński, 1996).

Switchgear rooms. The room of the main underground switchgear station is located in the special working built at the pit-bottom. Each section of the station is located in the separate room working ventilated with the fresh airflow. Each room is accessible from two sides and is equipped with a switchgear 6V, transformer station, and box switchgears. The rooms of the regional and district switchgears are located in the break-throughs between roads ensuring necessary ventilation (Piestrzyński, 1996).

Communication and Signalling. There is a telecommunication network in the LGOM area. In each industrial plant, the automatic exchange is connected to the main cable network. There are also computer networks (intranet) for particular shafts. The underground and surface telephone communication is based on Definity G3 and V3 telecommunication-information system. The system is powered with a constant voltage of 58 V and has emergency power supply system in the form of batteries that can work for 12 hours. Alarm signalling is built on the base of AUD-80/160 NN alarm system. This system co-operates with alarm acoustic signalling devices and with emergency telephones ATA–80. In mine districts, there are signalling systems operating for ventilation system, belt conveyor transport, vertical transport of people and machines, fire hazard, and organization of transport systems (Piestrzyński, 1996).

8.9 Control Systems-Industrial Safety

Rockburst Hazard. It is the most serious natural hazard in Polish copper mines. The main cause of the occurrence of tremors and bumps is the geological structure of the deposit. Many methods of the hazard protection and fighting it have been worked out. There is a technological method, the active method and methods for evaluation and monitoring of the rock mass behaviour. Rock mass is observed using seismological measures such as seismic activity and tremor energy. Then some geotechnical tests are done such as measurements of compression strength, degree of roof subsidence, etc. Technological methods for fighting the hazard consist in the improvement of mining methods and roof control. The active methods uses a blasting technique to induce tremors, relieving strain energy accumulated in rock mass Kłeczek et al, 1996).

Radiation Hazard. To lessen concentration of radioactive elements i.e. radon in the air some things could be done: (1) increasing ventilation intensity; (2) insulating gob area; (3) using independent ventilation systems; (4) separating workings which are not necessary; (5) leading water directly to pipelines; (6) controlling the employees individual dose of radiation (Piestrzyński, 1996).

Silicosis Hazard. Authorized laboratories perform analyses. Preventive actions are performed such as (1) using water when drilling operations are carried out; (2) spraying excavated material with water during mining, loading, hauling; (3) spraying water on transport roads. Workers should wear filter masks (Piestrzyński, 1996).

Toxic Gases Hazard. To decrease concentration of CO and $NO_2$ in mine air one may do: (1) limited number of cars powered with oil; (2) shortening haulage roads; (3) testing cars exhausted gases; (4) repairing cars properly. Measurements of the gases concentration are carried out every day (Wacławik et al, 1996).

Fire Hazard. The factors and potential causes of fire are: fuel, machines, belt conveyors, blasting, oil, explosives, and electricity (Wacławik et al, 1996).

Climatic Hazard. Temperature where people work should be lower than 28°C. There are many places where temperature is higher. Some activities should be taken up (1) insulation of gob; (2) short ways for fresh air, (3) air conditioning device, (4) using underground portable fans (Wacławik et al, 1996).

Noise and Vibration Hazard. Noise is tested at work places. Vibrations are also tested at work places. Workers should use special gloves and muffs.

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