OPEN PIT FUNDAMENTALS - Terminology

BENCH: Ledge that forms a single level of operation above which mineral or waste materials are mined from the bench face.

BENCH HEIGHT: Vertical distance between the highest point on the bench (crest) and the lowest point of the bench (toe). It is influenced by size of the equipment, mining selectivity, government regulations and safety.

BENCH SLOPE OR BANK ANGLE : Horizontal angle of the line connecting bench toe to the bench crest.

BERM: Horizontal shelf or ledge within the ultimate pit wall slope left to enhance the stability of a slope within the pit and improve the safety. Berm interval, berm width and berm slope angle are determined by the geotechnical investigation.

OVERALL PIT SLOPE ANGLE: The angle measured from the bottom bench toe to the top bench crest. It is the angle at which the wall of an open pit stands and it is determined by: rock strength, geologic structures and water conditions. The overall pit slope angle is affected by the width and grade of the haul road.

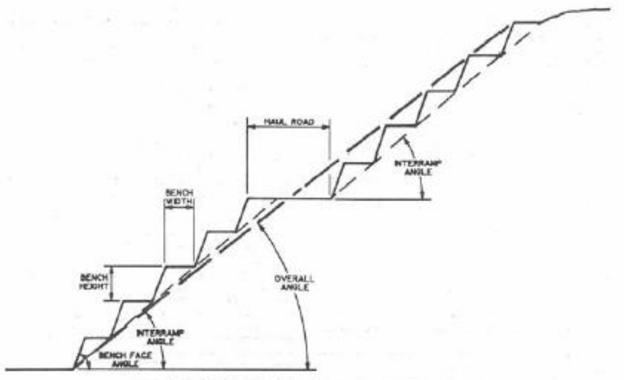
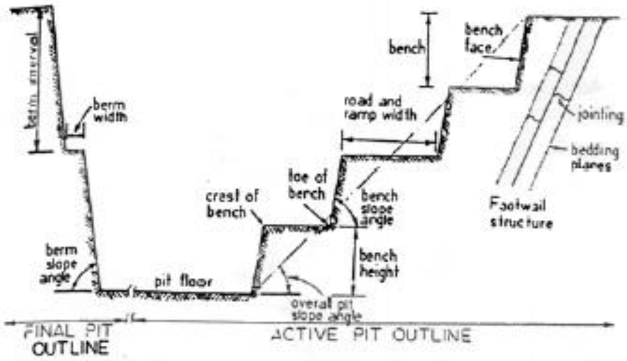


Fig. 5. Definition of bench face, interramp, and overall angles.

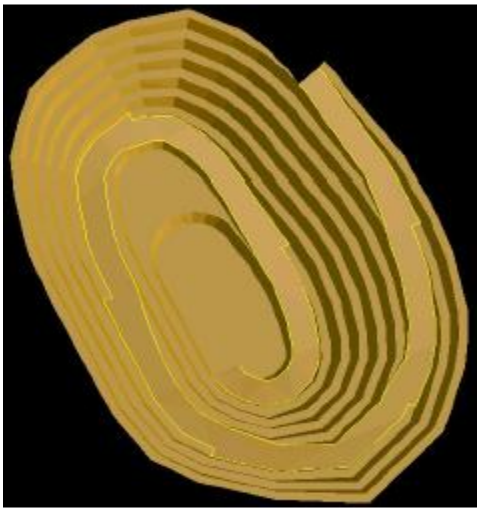


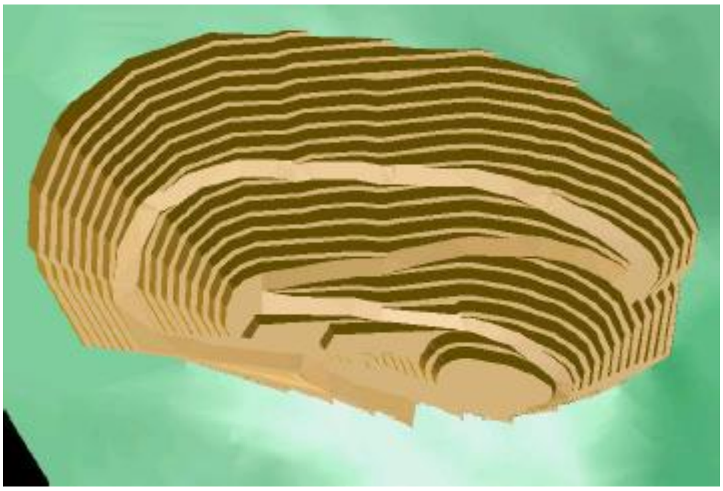
HAUL ROADS: During the life of the pit a haul road must be maintained for access.

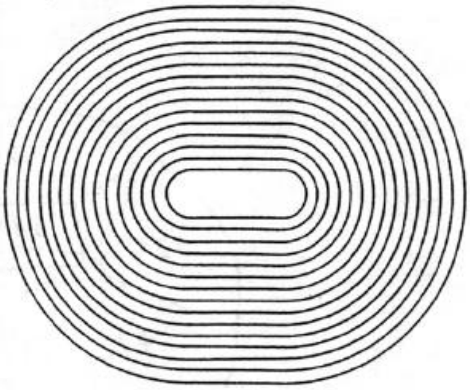
HAUL ROAD - SPIRAL SYSTEM: Haul road is arranged spirally along the perimeter walls of the pit.

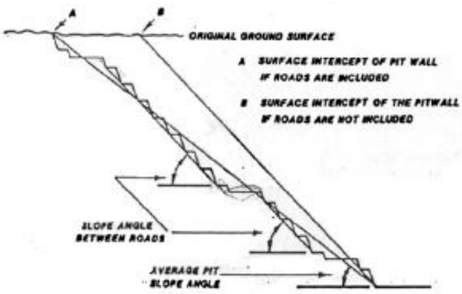
HAUL ROAD – SWITCH BACK SYSTEM: Zigzag pattern on one side of the pit.

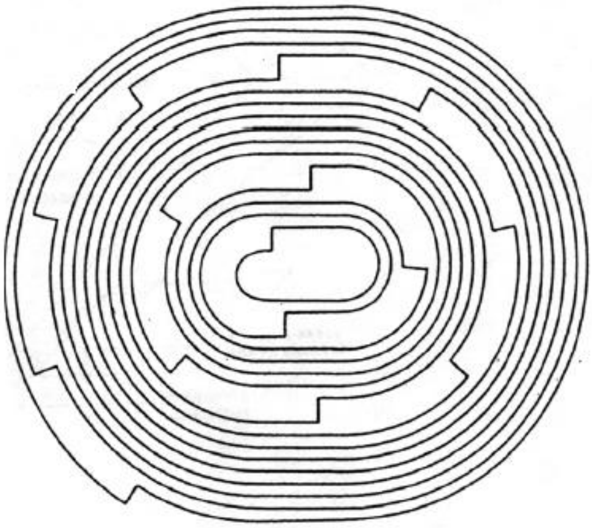
HAUL ROAD WIDTH: Function of capacity of the road and the size of the equipment. Haul road width must be considered in the overall pit design.











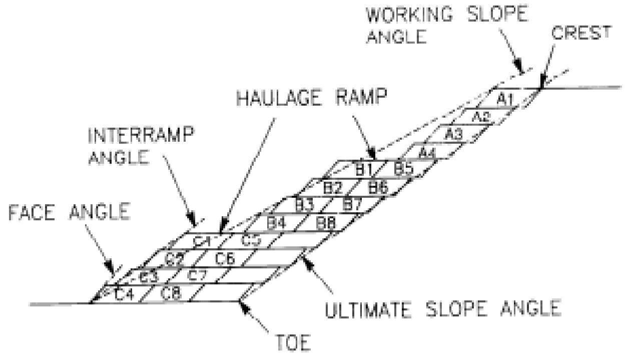
ANGLE OF REPOSE: Maximum slope of the broken material.

SUBCROP OR ORE DEPTH: Depth of waste removed to reach initial ore.

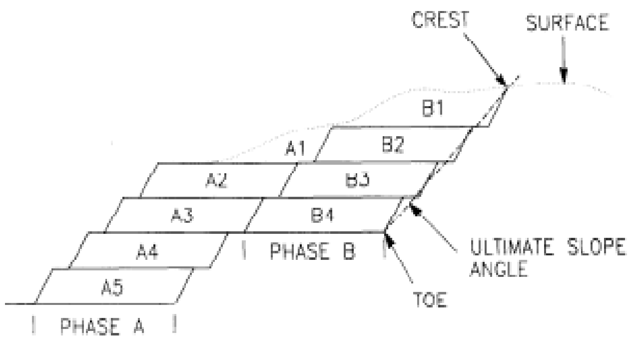
PRE-PRODUCTION STRIPPING: Stripping done to reach initial ore.

ULTIMATE PIT LIMITS: Vertical and lateral extend of the economically mineable pit boundary. Determined on the basis of cost of removing overburden or waste material vs. the mineable value of the ore.

PIT SCHEDULING: Material may be mined from the pit either in 1) sequential pushbacks 2) conventional pushbacks.



Cut sequence for sequential pushbacks.



Cut sequence for conventional pushbacks.

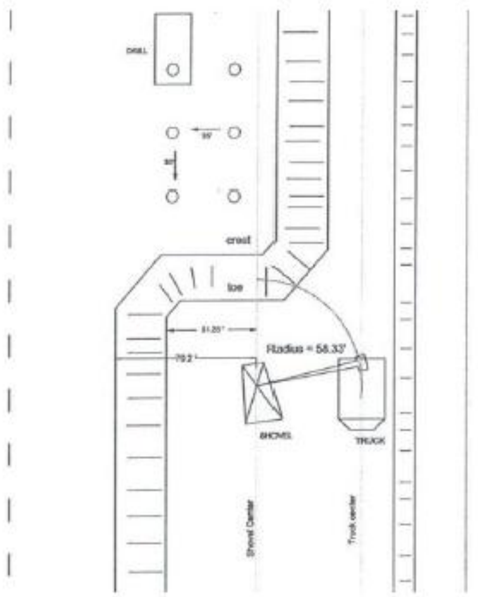
STRIPPING RATIO: Expressed in tons of waste to tons of ore in hard rock open pit operations. Critical and important parameter in pit design and scheduling.

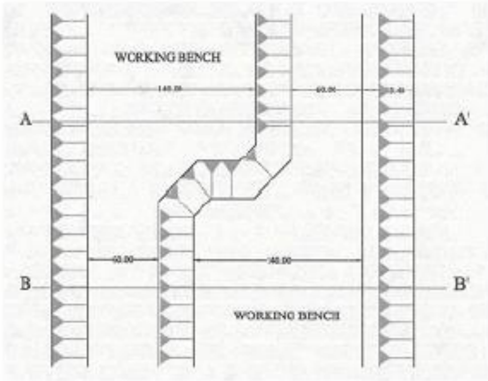
Stripping ratio refers to the ratio of the volume of overburden (or waste material) required to be removed to the volume of ore recovered. For example, a 3:1 stripping ratio means that mining one cubic meter of ore will require mining three cubic meters of waste rock.

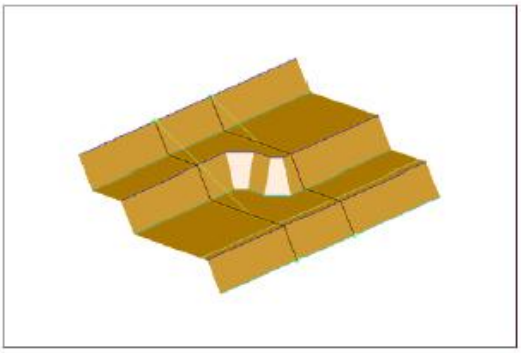
AVERAGE STRIP RATIO: Total waste divided by total ore within the ultimate pit.

CUTOFF STRIPPING RATIO: Costs of mining a ton of ore and associated waste equals to net revenue from the ton of ore.

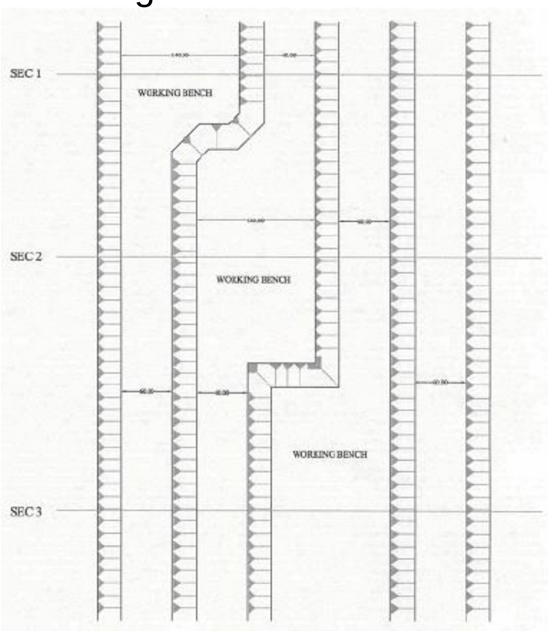
Single Working Bench

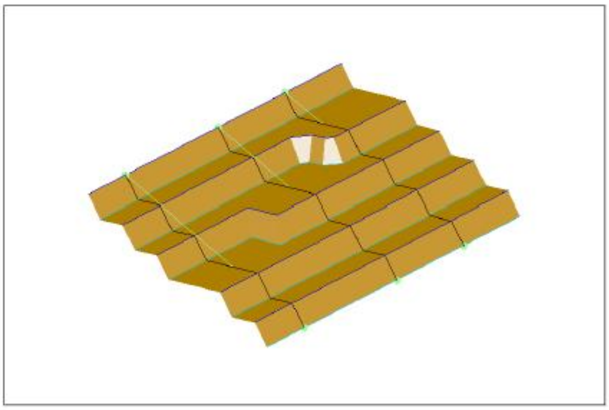






Two Working Benches





Shovel in Working Bench

Assumptions:

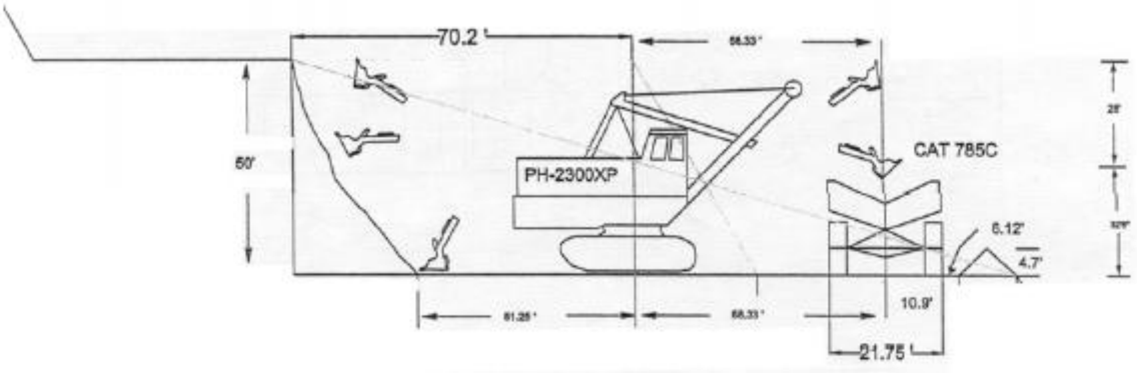
Shovel: PH-2300XP & CAT 785C

Tire type:33.00x51 (From Cat Handbook)

Axle height = (51 + 2x0.95x33)/2 /12 = 4.70' (formula)

D = 51.25 (Radius at level floor)

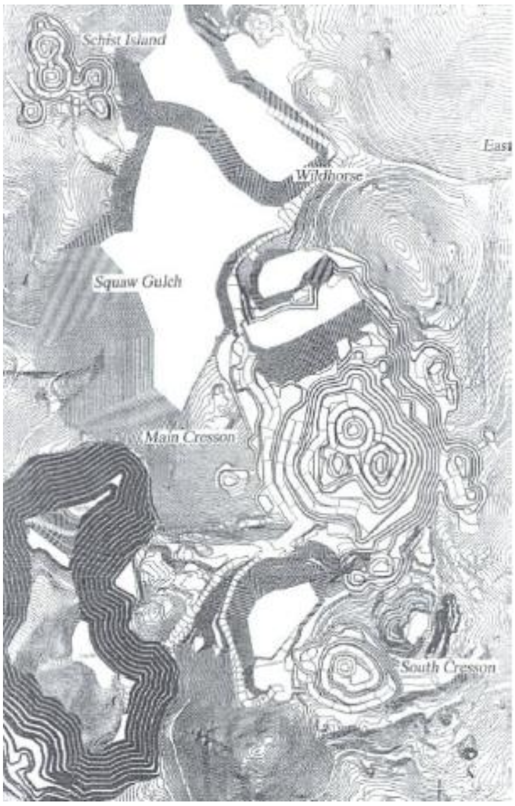
Width of cut = 0.90x2xD (Aprox. 90')



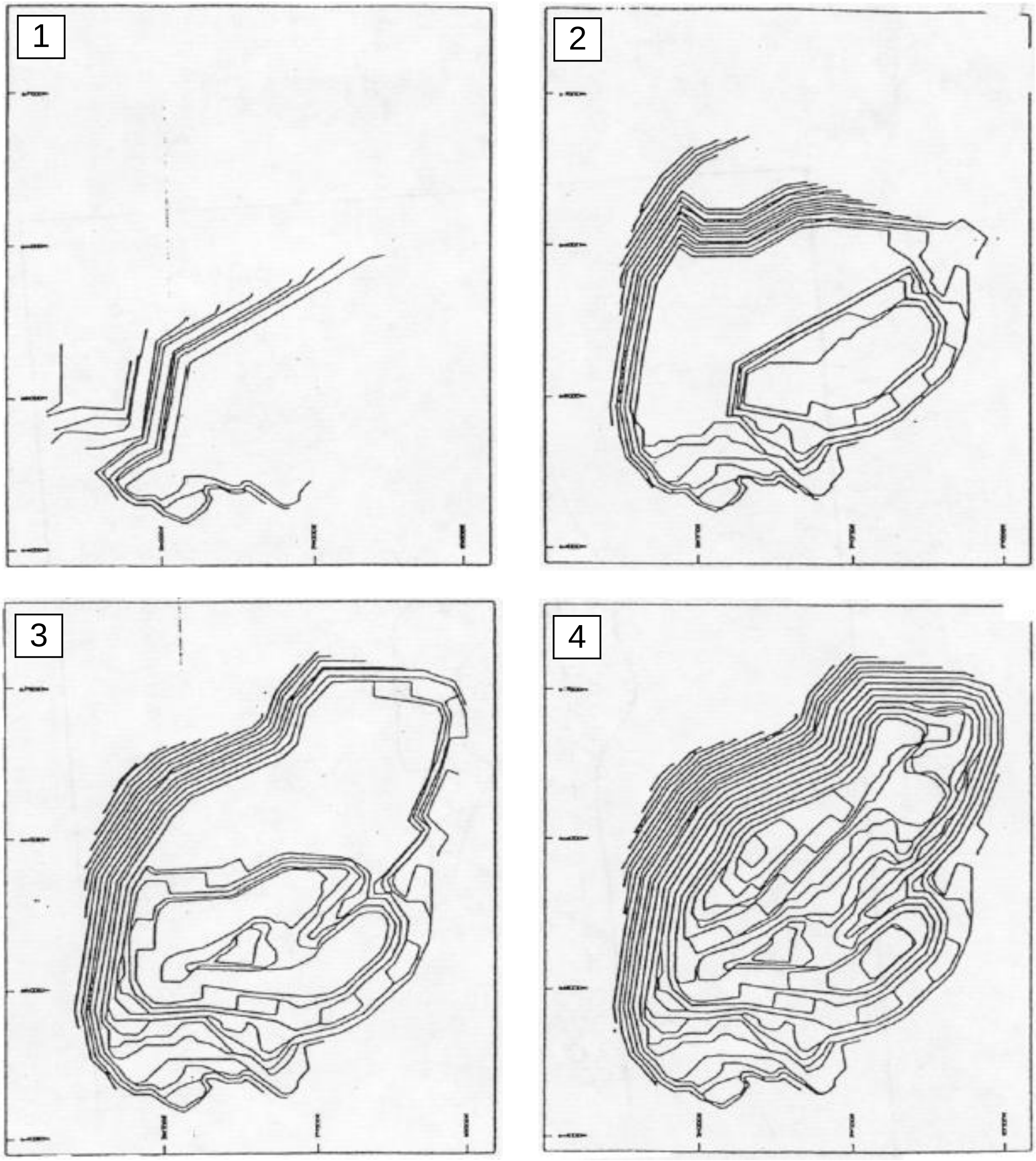
FinalPit Limit

Cresson Mine – Year 2001

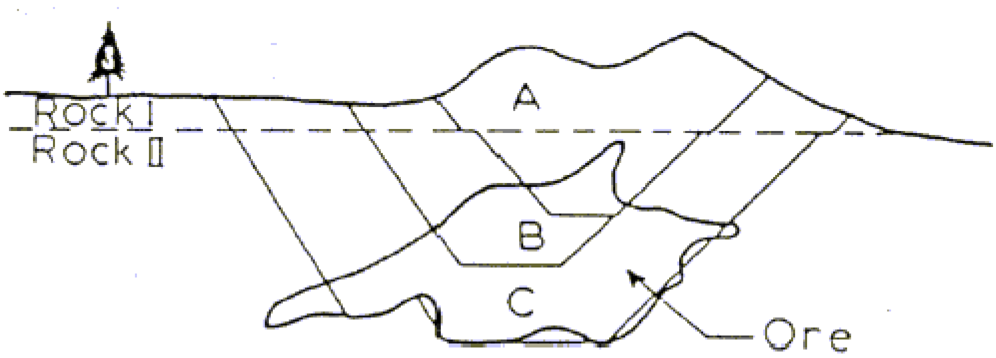
Cresson Mine – Year 2007



Cresson Mine – Year 2011



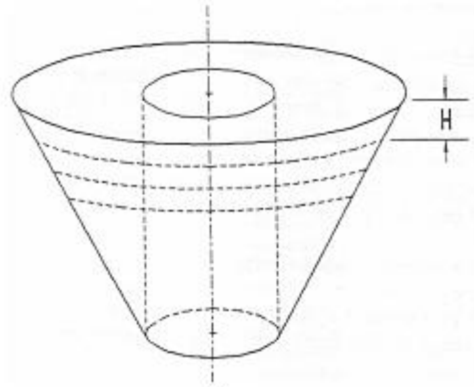
Pit sequences



Section of Pit Sequence

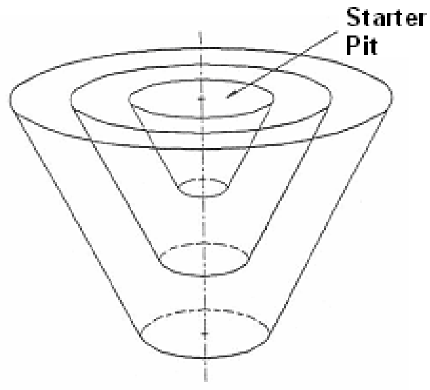
Geometric sequencing

There are several ways in which the volume of a deposit can be mined. The first step in the process is to divide the volume into a series of benches (see Fig.a). If a sinle bench is mined per year then the ore production would remain constant while both the total production and the stripping ratio would decrease. This would lead to a particular cash flow and net present value. For most mining projects, a large amount of waste mining in the early years of a project is not of interest.



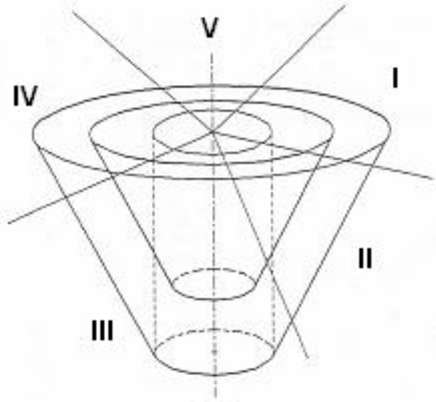
(a)

An alternative mining geometry is shown in Fig.b in which a number of levels are mined at the same time. The overall geometry looks much like that of an onion. An initial "starter pit" is first mined. In this particular example the push-backs (The 'bites' in surface mining terms are called *push-backs* or phases) result in the pit being extracted in a series of concentric shells. Hence for a consant production rate there might be x years of ore production in sheel 1, y years of production in 2, etc.



(b)

Sequencing within a pit shell and between shells becomes imporant. To this point simple concentric shells have been considered. The next level of complication is to split the overal pit into a number of sectors such as shown in Fig.c. Each sector (I→V) can be considered as a separate production or planning unit. A natural basis for dividing the pit this way is due to slope stability/design considerations.

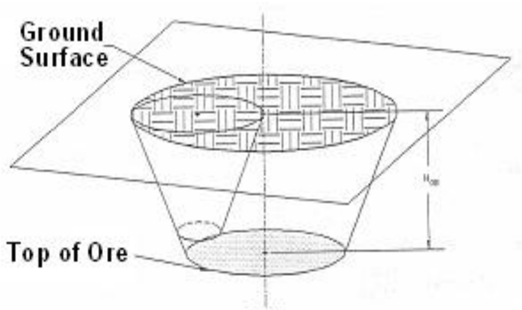


(c)

It has been assumed that the orebody outcrops (is exposed) at the surface. If this is not the case, such as is shown in Fig.d, then a preproduction or stripping phase must be first considered.

Due to cash flow considerations a variety of aspects enter:

- desire to reach the ore as quickly as possible,
- requirement to expose enough to maintain the desired plant production,
- combination of higher grade ore at greater depth versus lower grade at shallower depth.

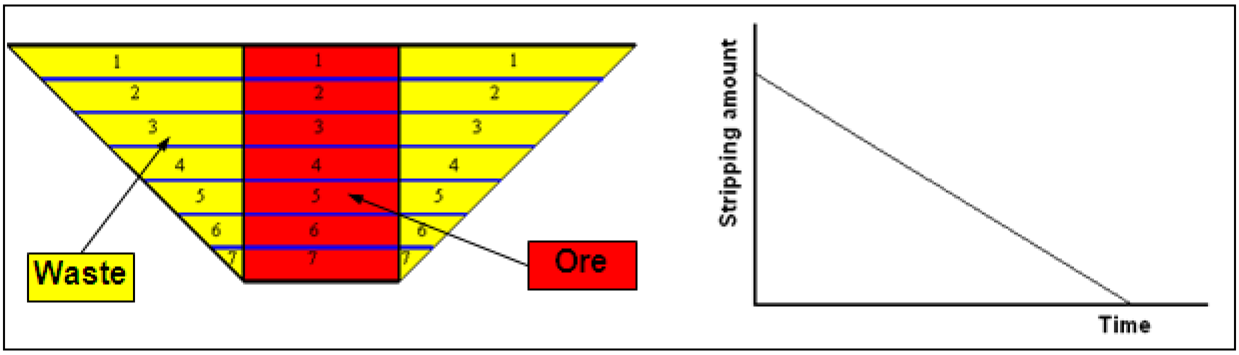


(d)

The geometry-sequencing decisions then become even more complex.

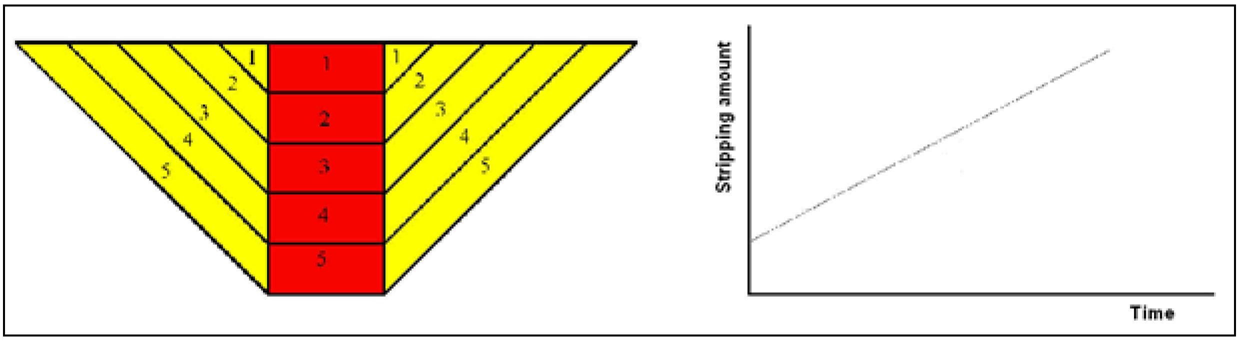
STRIPPING RATIO VS. TIME

Declining Stripping Ratio Method- this requires that each bench of ore be mined in sequence, and all the waste on the particular bench is removed to the pit limit.



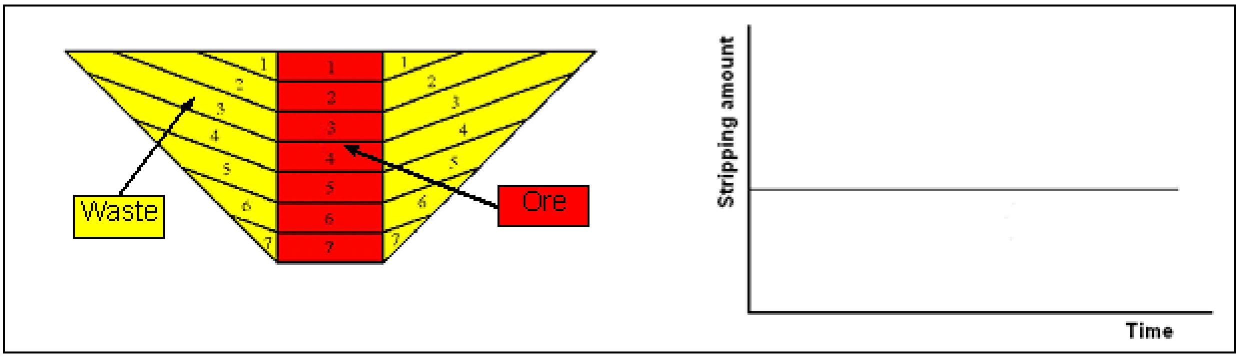
Advantages	Disadvantages
1. Operating space available 2. Accessibility of the ore on subsequent bench 3. All equipment working on the same level 4. No contamination form waste blasting above the ore 5. Equipment requirements are minimum towards the end of the mine's life	1. Operating costs are a maximum during initial years when profits are required to handle interest and repayment of capital.

Increasing Stripping Ratio Method- stripping is performed as needed to uncover the ore. The working slopes of the waste faces are essentially maintained parallel to the overall pit slope angle.



Advantages	Disadvantages
1. Maximum profit in the initial years 2. Greatly reduces investment risks in waste removal for ore to be mined at future date 3. Method is popular where the mining economics or cutoff ratio is likely to change	1. Impracticality of operating a large number of stacked, narrow benches simultaneously to meet production needs.

Constant Stripping Ratio- This method attempts to remove the waste at a rate approximated by the overall stripping ratio. The working slope of the waste face starts very shallow, but increases as mining depth increases until the working slope equals the overall pit slope.



Advantages	Disadvantages
1. Equipment fleet size and labor requirements throughout the project are relatively constant. 2. Good profit initially to increase cash flow. 3. Distinct mining and stripping areas can be operated simultaneously, allowing for flexibility in planning.	1. Disadvantage and advantages is the compromise that removes the extreme conditions of other two stripping methods.

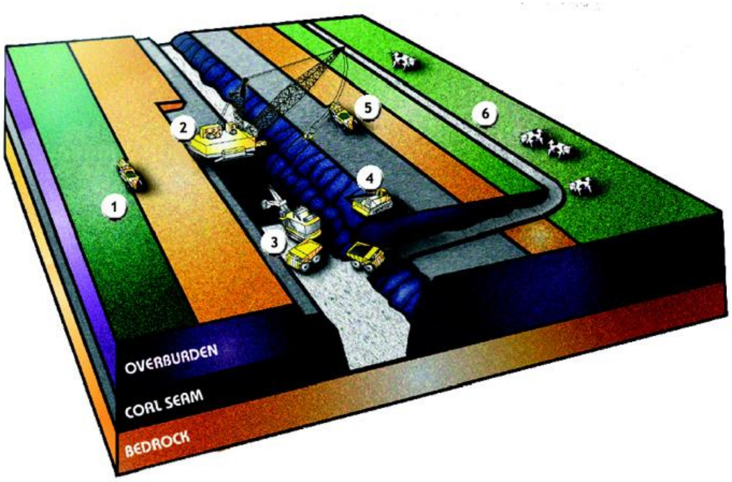
STATUS OF SURFACE MINING

• Metalliferous ores ⟹ OPEN PIT mining

Mirny Diamond Mine , Siberia

This hole is an absolute beast and holds the title of largest open diamond mine in the world. At 525 metres deep with a top diameter of 1,200 metres, there is a no-fly zone above the hole due to a few helicopters having being sucked in. The red arrow in the photo above points to a huge truck.

• Coal ⟹ STRIP mining
OPEN CAST mining



• Non-metallic materials ⟹ QUARRYING

Surface mining of ore bodies dates back to ancient times, but
- Insignificant before 20 th century
- After World War II ➡ rapid tehcnological improvements

• Introduction of fertilizer grade AN-FO explosives

• rotary drilling

• truck haulage

• Increasing capacity of trucks and excavating equipment

• Introduction of auxilary, labour-saving devices

• Innovations in MINERAL BENEFICIATION

⇨ IMPROVEMENT IN PRODUCTIVITY OF SURFACE MINING...

At present, about 2/3 of the estimated world production of crude mined from surface

1/2 metallic ores
1/3 coal
all Non-metallic ores
(clays, stone,sand)
} mined from surface

Estimated world and US production of Crude Metallic and Non-metallic ores and coal by Surface Mining (*10^6 short-tons)

	WORLD			USA			Total US % of World
	Total	Surface	%	Total	Surface	%	
Metallic Ores	1800	900	50	458	376	82	25
Non-Metallic Ores	1000	850	85	148	114	77	15
Clay,Stone Sand&Gravel	3000	3000	100	1657	1621	98	55
Coal	3000	1000	33	504	176	35	17
TOTALS and %	8800	5750	65	2767	2287	83	31

Exceptions:

	USA	WORLD
Copper Ore	74 %	40 %
Iron Ore	90 %	50 %

In USSR:
50 % of all minerals ⇨ At present
80 % of iron ore ⇨ In the future
65 % of non-ferrous ore ⇨ In the future
50 % of coal ⇨ In the future

Proportional share of Surface Mining in the total production of minerals (%)

	1950	1960	1965	1970	1980
Coal	11	20	24	28	52
Iron ore	44	56	63	70	80
Ores of other metals	46	53	61	64	70
Manganese ore	-	29	55	75	80
Mined chemical raw materials	-	50	72	78	80
Building Materials	100	100	100	100	100

ADVANTAGES OF SURFACE MINING (OPEN PIT)

- Greater possibilities of MECHANIZATION the use of large, powerful and highly efficient mining equipment
  - 270 tons capacity trucks (500 ton capacity)
  - 170 m^3 bucket capacity draglines
  - 140 m^3 bucket capacity shovels
  - 8400 m^3/hr bucket capacity bucket wheel excavators
- Higher labour efficiency ➡ Output per man 5-6 times higher
  Lower mining cost ➡ 25-30 % of U/G mine (per ton of ore)
- The possibility of quicker increase in production of minerals, the higher rate of their recovery
- Freedom of movement ➡ Improved hygienic and working conditions
  Sufficient light & air
  ⇩ higher efficiency of workers
- No need for support, filling
- Higher percentage of recovery (mineral loss is less)
- Less danger of accidents.
  However:
  - – presence of numerous machines
  - – heavy transport network
  - – blasting operations
  - – fall of blocks

  } Sources of potential accidents ⇩ observe safety rules
- Effective control

DISADVANTAGES OF SURFACE MINING

- Considerable original capital outlays for the purchase of equipment and stripping operations
- Rainfall
  Snow
  Severe cold
  } makes work in open pits difficult ⇒ labour and machine efficiency drop
- Destroys the natural beauty of the land by <u>scars</u> left from open pits and accumulation of <u>rejected waste</u>.
  ⇒ strong pressure from public.
- Restricted to relatively shallower depths

Surface mining is more advantageous than U/G mining in
- RECOVERY
- GRADE CONTROL
- ECONOMY
- FLEXIBILITY OF OPERATION
- SAFETY
- WORKING ENVIRONMENT

• However, when the deposit is
  - too small
  - irregular
  - deeply buried

  Can not be extracted economically by S/M methods.

• Deeper mineralization ⇒ Overburden ↗ ⇒ impose economic limits

• Changing public opinion ⇒ reduction or elimination of S/M
  ⇒ deciding factor in determining the future trend in surface vs. U/G mining

Complaints:
- destroying the natural beauty of land
- accumulation of rejected waste
- pollution of streams by acids and solids.
- pollution of atmosphere by dust

Developments in:
  electronics
  automation
  rock-boring machines
Research in:
  rapid excavation
  principles
  techniques
  equipment

Significant contribution to u/g mining

National defense ➡ locating strategic industry in u/g
Programs ➡ storing strategic supplies in u/g

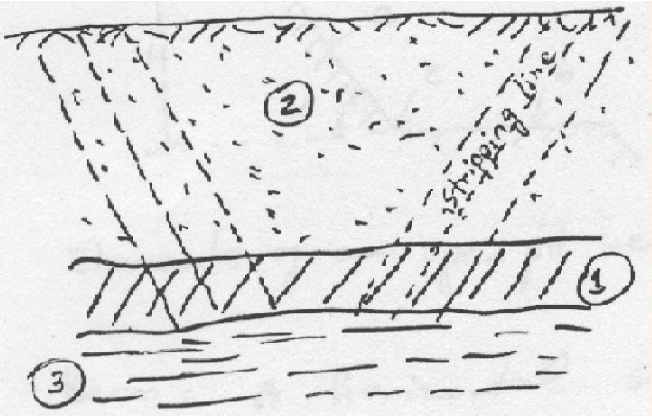
IT CAN BE CONCLUDED THAT:

• <u>Wherever</u>, surface mining operators can put large quantities of mineral on the market at costs that can not be matched by underground operators
  ⇒ SURFACE MINING METHODS will retain and expand.

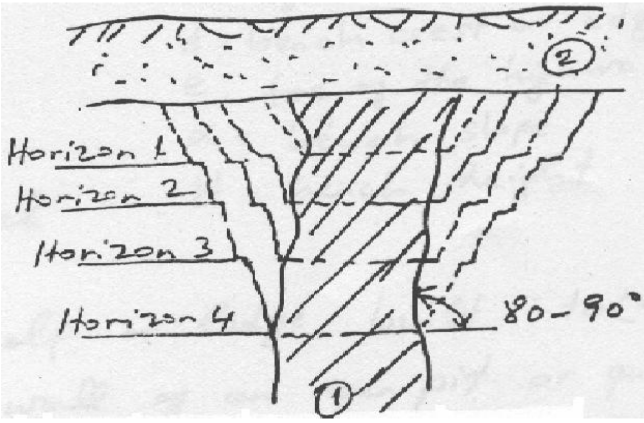
• <u>However</u>, the factors opposing surface mining and favoring U/G mining are increasing in densely populated, industrialized areas of the world, and
  ⇒ the trend toward SURFACE MINING IN THESE AREAS CAN BE EXPECTED TO REVERSE IN THE FUTURE.

TYPICAL EXAMPLES OF OPEN PIT MINING

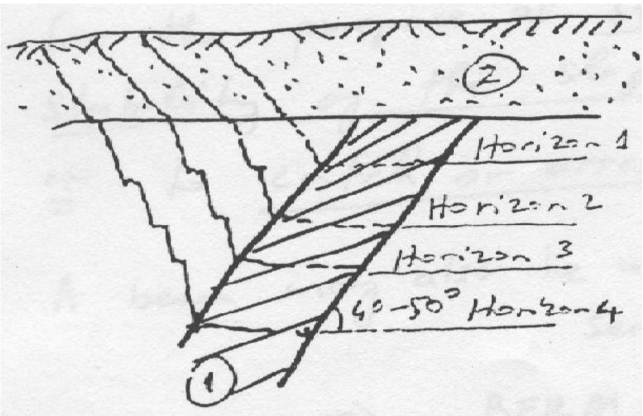
When deposit lie immediately near surface or at a relatively small depth } open pit mining justifiable (technologically, economically)



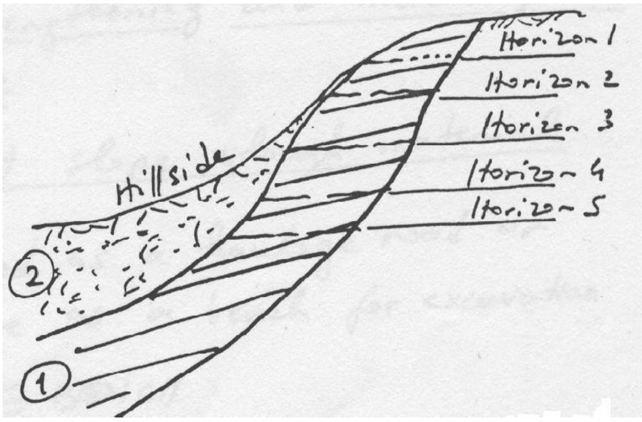
HORIZONTAL DEPOSIT



STEEPLY DIPPING DEPOSIT



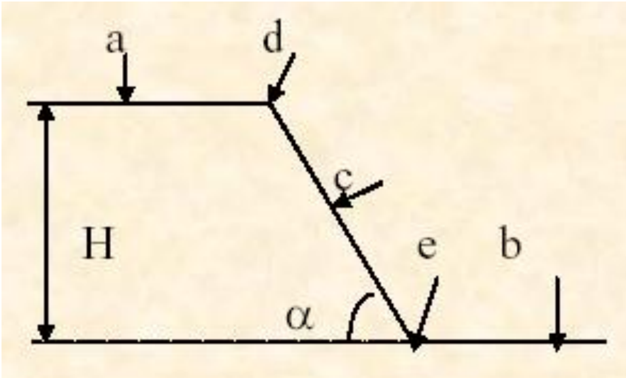
INCLINED DEPOSIT



DEPOSIT OCCURING ON THE HILLSIDE

BASIC CONCEPTS AND TERMINOLOGY

- Benching Bank or BANK or BENCH: Single level of operation above which mineral or waste materials are excavated from a continous bank.

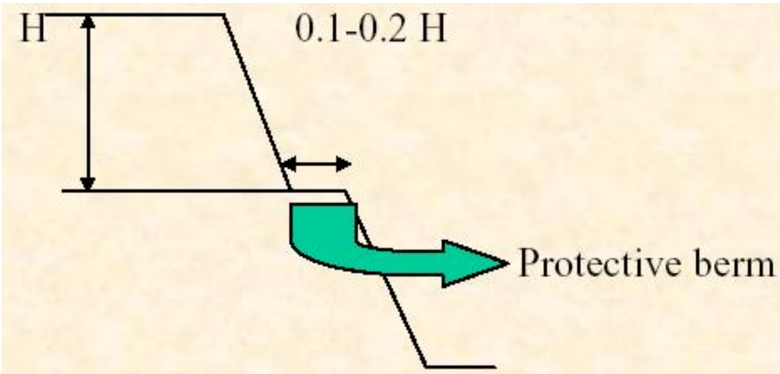


a. top (roof)
b. bottom (bench floor)
c. highwall (bench face)
d. bench crest or edge
e. toe of the highwall
α. bench slope
H. bench height

Elements of an open pit bench

- Berm: A horizontal shelf or ledge built into (an embankment) or sloping wall of an open pit or quarry to break the continuity of an otherwise long slope for the purpose of <u>strengthening and increasing the stability of the slope or to catch or arrest slope slough material</u>.
- A berm may also be used as a haulage road or serve as a bench for excavation

$\Rightarrow$ BERM $\approx$ BENCH



DIMENSIONS OF A BENCH

DIMENSION → its height, angle of slope } depends on → Equipment used → Nature of ground → Conditions of work

Correct relationship between the height of bench and the working dimensions of the excavator ensures the most

- efficient
- safe
- economical

} operation of the equipment

<u>In non-solid rocks:</u>

BENCH HEIGHT ≈ Digging height of power shovel.

<u>In hard rocks.</u>

HEIGHT OF BROKEN ROCK PILE ≤ 1.5*Digging height of power shovel.

Selective loading of different grades of ore
or
Loading of ore separately from waste
} BENCH HEIGHT ≤ Digging height of excavator.

<u>In Practice:</u> (Open Cast Work)

BENCH HEIGHT = 3 → 20 m
- waste → 10-15 m
- ore → 8-12 m

} with power shovel

SLOPE ANGLE (α)

- Friable and soft Rocks → $\alpha \leq$ Angle of Repose
- Very hard Ign. & met. Rocks → $\alpha = 70^o$-80^o
- Sedimentary Rocks → $\alpha = 50^o$-60^o
- Semi-ledge & dry sand grounds → $\alpha = 40^o$-50^o
- Argilleous rocks → $\alpha = 35^o$-45^o

STABILITY OF SLOPES depends on
→ Petrographic composition
→ Structural Defects
→ Groundwater conditions

MINE PLANNING

1. INTRODUCTION

1.1 The meaning of ore

Ore : A natural aggregation of one or more solid minerals that can be mined, processed and sold at a profit.

The key concept is "*extraction leading to a profit*". For engineers, profits can be expressed in simple equation form as

Profits = Revenues – Costs

Revenues = Materials sold (units) * Price / unit

Costs = Materials sold (units) * Cost / unit

Combining the equations yields

Profits = Materials sold (units) * (Price / unit - Cost / unit)

The price received is more and more being set by world wide supply and demand. Thus, the price component in the equation is largely determined by others. Where the mining engineer can and does enter is in doing something about the unit costs. To remain profitable over the long term, the mining engineer must continually examine and assess smarter and better site specific ways for reducing costs at the operations. This is done through a better understanding of the deposits itself and the tools/techniques employed or employable in the extraction process. A failure to keep up is reflected quite simply by the profit equation as

Profits < 0

For the mining engineer (student / practicing), the personal meaning of ore is

Ore ≡ Profits ≡ Jobs

Hence one important meaning of "ore" to us in the mineral business is jobs. Probably this simple practical definition is more easily remembered.

1.2 Some important definitions

The exploration, development, and production stages of a mineral deposit are defined as:

Exploration : The search for a mineral deposit (prospecting) and the subsequent investigation of any deposit found until an orebody, if such exists, has been established.

Development : Work done on a mineral deposit, after exploration has disclosed ore in sufficient quantity and quality to justify extraction, in order to make the ore available for mining.

Production : The mining of ores, and as required, the subsequent processing into products ready for marketing.

The definitions presented are tied closely to the sequential relationship between exploration information, resources and reserves shown in Figure 1.

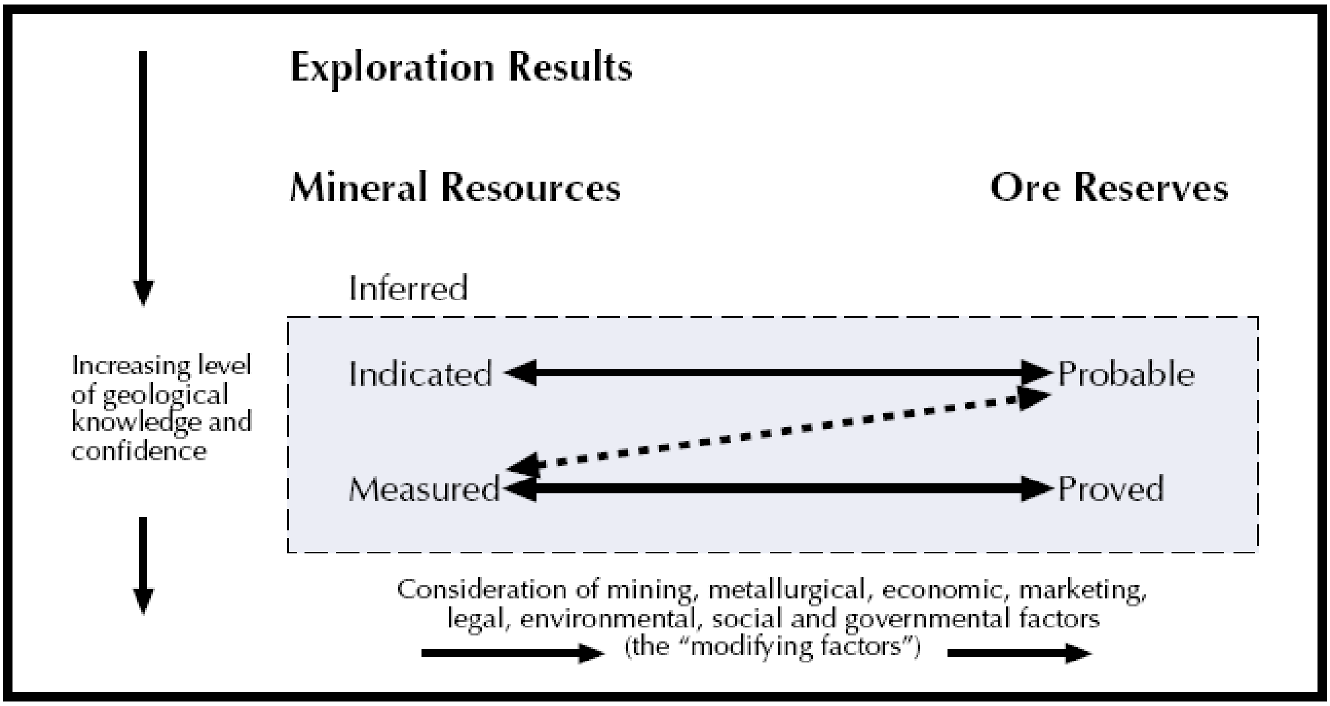


Figure 1. The relationship between exploration information, resources and reserves (The JORC Code, 2004)

With an increase in geological knowledge, the exploration information may become sufficient to calculate a resource. When economic information increase it may be possible to convert a portion of the resource to a reserve. The double arrows between reserves and resources indicate that changes due to any number of factors may cause material to move from one category to another.

Exploration information : Information that results from activities designed to locate economic deposits and to establish the size, composition, shape and grade of these deposits. The methods include geological, geomechanical, and geophysical surveys, drill holes, trial pits and surface underground openings.

Resource : A concentration of naturally occurring solid, liquid or gaseous material in or on the Earth's crust in such form and amount that economic extraction of a commodity from the concentration is currently or potentially feasible. Location, grade, quality and quantity are known or estimated from specific geological evidence. To reflect varying degrees of geological certainity, resources can be subdivided into measured, indicated, and inferred.

-Measured (Görünür) : Quantity is computed from dimensions revealed in outcrops, trenches, workings or drill holes; grade and/or quality are computed from the results of detailed sampling. The sites for inspection, sampling and measurement are spaced so closely and the geological character is so well defined that size, shape, depth and mineral content of the resource are well established.

-Indicated (Muhtemel) : Quantity and grade and/or quality are computed from information similar to that used for measured resources, but the sites for inspection, sampling, and measurements are further apart or are otherwise less adequately spaced. The degree of assurance is high enough to assume geological continuity between points of observation.

-Inferred (Mümkün) : Estimates are based on geological evidence and assumed continuity in which there is *less confidence* than for measured and/or indicated resources.

Reserve : A reserve is that part of the resource that meets minimum physical and chemical criteria related to the specified mining and production practices, including those for grade, quality, thickness and depth; and can be reasonably assumed to be economically and legally extracted or produced at the time of determination. Reserves relate to resources as follows:

-Proven reserve : That part of a measured resource that satisfies the conditions to be classified as a reserve.

-Probable reserve : That part of an indicated resources that satisfies the conditions to be classified as a reserve.

The terms "measured reserve" and "indicated reserve", generally equivalent to "proven reserve" and "probable reserve" respectively. The terms "measured", "indicated" and "inferred" qualify resources and reflect only differences in geological confidence. The terms "proven" and "probable" qualify reserves and reflect a high level of economic confidence as well as differences in geological confidence.

It is recommended that proven and probable reserves be reported separately. Where the terms reserve is used without the modifiers proven or probable, it is considered to be the total of proven and probable reserves.

2. MINE DEVELOPMENT PHASES

The mineral supply process is shown diagrammatically in Figure 2. As can be seen a positive change in the market place creates a new or increased demand for a mineral product.

In response to the demand, financial resources are applied in an exploration phase resulting in the discovery and delineation of deposits. Through increases in price and/or advances in technology, previously located deposits may become interesting. These deposits must then be thoroughly evaluated regarding their economic attractiveness. This evaluation process will be termed the "*planning phase*" of a project. The conclusion of this phase will be the preparation of a feasibility report. Based upon this, the decision will be made as to whether or not to proceed. If the decision is "go", then the development of the mine and concentrating facilities is undertaken. This is called the *implementation, investment,* or *design and construction phase*. Finally there is the *production* or *operational phase* during which the mineral is mined and processed. The result is a product to be sold in the market place. The entrance of the mining engineer into this process begins at the planning phase and continues through the production phase.

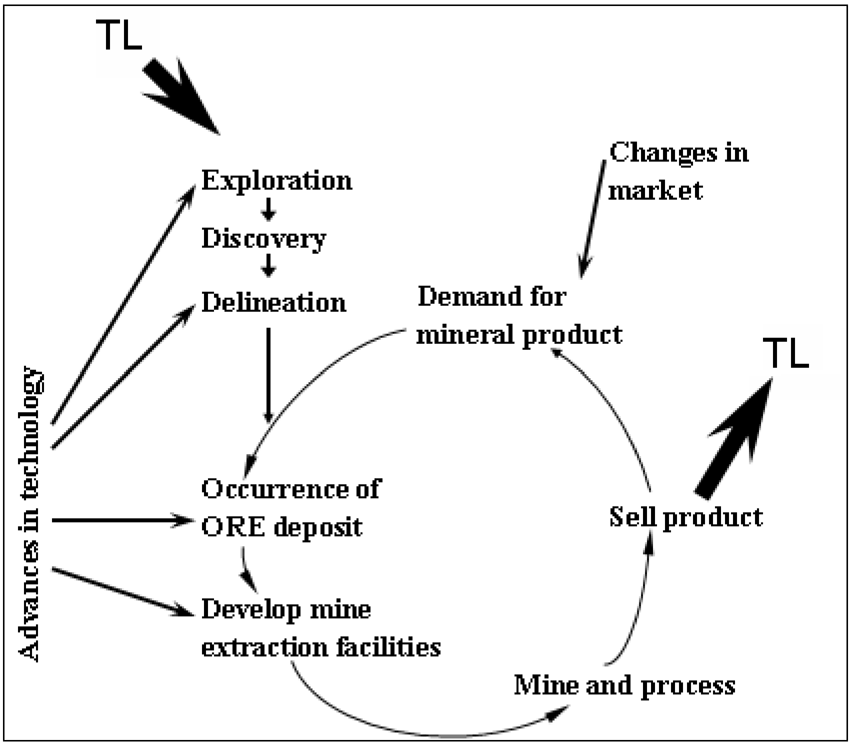


Figure 2. Diagrammatic representation of the mineral supply process (McKenzie, 1980)

1.5 Ore Reserves

General

Historically, mineral properties were simply classified into three reserve categories: *Proven, Probable and Possible*. In 1980, the USBM and USGS jointly introduced an expanded system that separated the geological analysis (resource determination) from the subsequent mineable analysis (reserve determination). In 1989, the AIMM of Australia expanded definitions and introduced the concept that a reserve analysis should be carried out under the direction of and certified by an "expert," who is a qualified professional. Since that time, more work has been accomplished to further refine the classification system. Most recently, the Canadian government published *"Standards for Disclosure" (National Instrument 43-101)* that provides further interpretation. Unfortunately, there is not yet an international standard that defines the terms used, let alone a common code for the determination of the separate categories of certainty. The good news is that an effort is underway to set international standards that will satisfy the securities commissions of the stock exchanges that provide the great majority of equity financing for the world's mining industry (Johannesburg, London, New York, Sydney, and Toronto). These new standards are more restrictive (more conservative) than the traditional standards.

In the mean time, major mining companies (who often finance new projects internally without resorting to the equity markets) developed their own systems. These are generally similar to the mainstream standards. One significant exception is that normally a qualified *team* of professionals is responsible for determining and certifying ore reserves rather than an individual "Expert," "Accredited Senior Appraiser," or "Qualified Person." The professional team concept appears wiser than the individual concept promoted by the published guidelines and standards, since few, if any, individuals are expert in four disciplines (geological interpretation, statistical analysis, mining methods, and mineral economics).

Definitions

The following are unofficial definitions intended to meet the expectations of a handbook. They are generally representative of current industry standards.

A discovered mineralization zone will undergo three phases of identification before it is ready for mining.

DEPOSIT → RESOURCE → RESERVE

A *Deposit* is a mineralized zone that is examined to the extent that the mineral constituents are identified. The examination may reveal some geological continuity and approximate grade (tenor), but there is insufficient information to class it as a *Resource*.

A *Resource* is a mineralized zone that is sampled and studied to the extent that a credible estimate of tonnage and grade can be made. This estimate is sufficiently definitive to divide it into three levels of certainty: Measured, Indicated, and Inferred.

A *Measured* Resource is the highest order and implies that the measurements are made in three dimensions at intervals close enough to be used as a basis for detailed mine planning, as would be the case for stope development in an operating mine. In the instance of a new project being contemplated, it is typical that there will be little or no measured resources.

An *Indicated* Resource is one that the estimate of tonnage and grade is trustworthy to the extent that it may serve as a basis for major expenditures.

An *Inferred* Resource is a potential resource that is assumed to exist, by reason of logical extrapolation or interpolation.

A *Reserve* is the portion of a mineral resource that may be extracted at a profit. While a resource is based upon geological interpretation, a reserve has been analyzed further to take actual mining into account. A more precise definition is, *"That part of a mineral resource that has been analytically demonstrated to justify mining, taking into account, at the time of determination, mining, metallurgical, marketing, legal, environmental, social, economic and other applicable conditions."* (Extracted from the Johannesburg Stock Exchange listing requirements: Mineral Companies, Chapter 12.)

A *Reserve* may be divided into two categories of certainty: Proven and Probable.

Table 1-2 Categories of Certainty

Category	Mineral Resource	Mineral Reserve	
1	Measured	Proven	2P
2	Indicated	Probable	(Proven + Probable)
3	Inferred		

A *Proven Reserve* is derived only from a measured resource. This category is generally limited to a mining zone being mined or one that is already prepared for mining, to include preproduction stope development. Typically, for most new mining projects, there will be little or no proven reserves.

A *Probable Reserve* is that part of an Indicated Resource subjected to the scrutiny required of a Detailed Feasibility Study (refer to Chapter 6) and thereby may be found to justify the expenditure required for mining. In a formal economic analysis, only probable and proven reserves are employed. The sum of these two categories may be referred to as "2P."

A *Possible Reserve* is an obsolete category since by current standards it may not be used in any formal economic analysis. When reporting on a mineral property, a potential or possible zone of mineralization best remains identified as an *Inferred Resource*. For purposes of a preliminary study, a "possible reserve" may be calculated to perform an analysis only when desired to justify additional exploration work.

3. AN INITIAL DATA COLLECTION CHECKLIST

In the initial planning stages for any new project there are a great number of factors of rather diverse types requiring consideration. Some of these factors can be easily addressed, whereas others will require in-depth study. To prevent forgetting factors, checklists are often of great value.

1. Topography (maps, surveys, control stations, contour)
2. Climatic conditions (altitude, temperatures, precipitation, wind, humidity, dust, fog and cloud conditions)
3. Water (sources, availability, quantities, quality, sewage disposal method)
4. Geological structure (within mine and surrounding areas, earthquakes)
5. Mine water as determined by prospect holes (depth, quantity, drainage)
6. Surface (vegetation, unusual conditions)
7. Rock type-overburden and ore (drillability, fragmentation)
8. Locations for concentrator (mine location, site preparation, process water, maintenance)
9. Tailings pond area (location and elevation, enclosing features, pond overflow, tailings dust)
10. Roads (roads, access roads)
11. Power (availability, lines, substation location)
12. Smelting (availability, method, rates, railroads and dock facilities)
13. Land ownership (owners, usage, price, leases and royalties)
14. Government (political climate, mining laws, local restrictions)
15. Economic climate (principal industries, labor, wages, tax structure, availability of goods and services, material costs and/or availability)
16. Waste dump location (haul distance, profile)
17. Accessibility of principal town to outside (transportation, communication)
18. Methods of obtaining information
    - Past records (i.e. government sources)
    - Maintain measuring and recording devices
    - Collect samples
    - Field observations and measurements
    - Field surveys
    - Make preliminary plant layouts
    - Check courthouse records for land information
    - Check local laws and ordinances for applicable legislation
    - Personal inquiries and observation on economic and political climates
    - Maps
    - Make cost inquiries
    - Make material availability inquiries
    - Make utility availability inquiries

4. THE PLANNING PHASE

The planning phase commonly involves three stages of study.

Stage 1: Conceptual study

A conceptual (or preliminary valuation) study represents the transformation of a project idea into abroad investment proposition, by using comparative methods of scope definition and cost estimating techniques to identify a potential investment opportunity. Capital and operating costs are usually approximate ratio estimates using historical data. It is intended primarily to highlight the principal investment aspects of a possible mining proposition. The preparation of such a study is normally the work of one or two engineers. The findings are reported as a preliminary valuation.

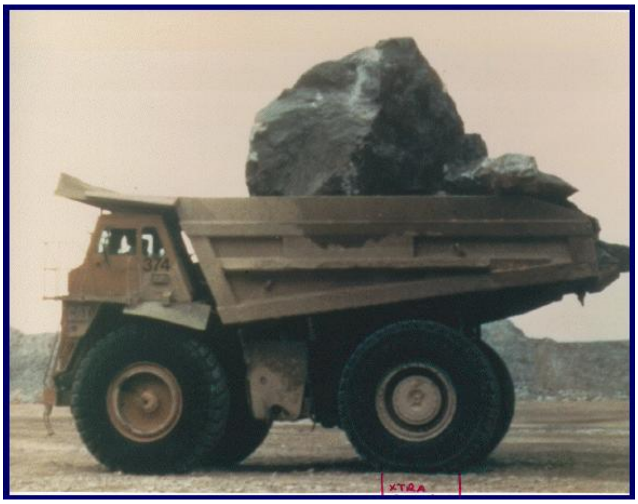
Stage 2: Preliminary or pre-feasibility study

A preliminary study is an intermediate-level exercise, normally not suitable for an investment decision. It has the objectives of determining whether the project concept justifies a detailed analysis by a feasibility study, and whether any aspects of the project are critical to its viability and necessitate in-depth investigation functional or support studies.
A preliminary study should be viewed as an intermediate stage between a relatively inexpensive conceptual study and a relatively expensive feasibility study. Some are done by two or three man team who have access to consultants in various fields others may be multi-group efforts.

Stage 3: Feasibility study

The feasibility study provides a definitive technical, environmental and commercial base for an investment decision. It uses iterative processes to optimize all critical elements of the project. It identifies the production capacity, technology, investment and production costs, sales revenues and return on investment. Normally it defines the scope of work and serves as a base-line document of the project through subsequent phases.

Mine Plan Frequency- Mine plans vary in frequency from short to long range. The plans can be daily, weekly, monthly, yearly or a life-of-the mine plan. The shorter the plan the more concise it will need to be. The longer plans will have to establish financial forecasts for replacement of equipment, variations in operating costs due to haul distances, and dewatering requirements. Changes in the ore type may dictate plan changes also.



Factors Requiring Consideration in Mine Planning and Feasibility Studies

I. Information On Deposit

- A. Geology: Overburden
  - a. Stratigraphy
  - b. Geologic structure
  - c. Physical properties (highwall and spoil characteristics, degree of consolidation)
  - d. Thickness and variability
  - e. Overall depth
  - f. Topsoil parameters
- B. Geology: Coal
  - a. Quality (rank and analysis)
  - b. Thickness and variability
  - c. Variability of chemical characteristics
  - d. Structure (particularly at contacts)
  - e. Physical characteristics
- C. Hydrology (Overburden and Coal)
  - a. Permeability
  - b. Porosity
  - c. Transmissitivity
  - d. Extent of aquifer(s)
- D. Geometry
  - a. Size
  - b. Shape
  - c. Attitude
  - d. Continuity

- E. Geography
  - a. Location
  - b. Topography
  - c. Altitude
  - d. Climate
  - e. Surface conditions (vegetation, stream diversion)
  - f. Drainage patterns
  - g. Political boundaries
- F. Exploration
  - a. Historical (area, property)
  - b. Current program
  - c. Sampling (types, procedures)

II. General Project Information

- A. Market
  - a. Customers
  - b. Product specifications (tonnage, quality)
  - c. Locations
  - d. Contract agreements
  - e. Spot sale considerations
  - f. Preparation requirements
- B. Transportation
  - a. Property access
  - b. Coal transportation (method, distance, cost)
- C. Utilities
  - a. Availability
  - b. Location
  - c. Right-of-way (geçiş- kullanma hakkı)
  - d. Cost
- D. Land and Mineral Rights
  - a. Ownership (surface, mineral, acquisition)
  - b. Acreage requirements (onsite, offsite)
  - c. Location of oil and gas wells, cemeteries, etc.
- E. Water
  - a. Potable and preparation
  - b. Sources
  - c. Quantity
  - d. Quality
  - e. Costs
- F. Labor
  - a. Availability
  - b. Rates and trends
  - c. Degree of organization
  - d. Labor history
- G. Governmental Considerations
  - a. Taxation (local, state, federal)
  - b. Royalties
  - c. Reclamation and operating requirements
  - d. Zoning
  - e. Proposed and pending mining legislation

III. Development and Extraction

A. Compilation of Geologic and Geographic Data
- a. Surface and coal contours
- b. Isopach development (thickness of coal and overburden, stripping ratio, quality, costs)

B. Mine Size Determination
- a. Market constraints
- b. Optimum economics

C. Reserves
- a. Method(s) of determination
- b. Economic stripping ratio
- c. Mining and barrier losses
- d. Burned, oxidized areas

D. Mining Method Selection
- a. Topography
- b. Refer to previous geologic/geologic factors
- c. Production requirements
- d. Environmental considerations

E. Pit Layout
- a. Extent of available area
- b. Pit orientation
- c. Haulage, power, and drainage systems
- d. Pit dimensions and geometry

F. Equipment Selection
- a. Sizing, production, estimates
- b. Capital and operating cost estimates
- c. Repeat for each unit operation

G. Project Costs Estimation (Capital and Operations)
- a. Mine
- b. Mine support equipment
- c. Office, shop, and other facilities
- d. Auxiliary facilities
- e. Manpower requirements

H. Development Schedule
- a. Additional exploration
- b. Engineering and feasibility study
- c. Permitting
- d. Environmental approval
- e. Equipment purchase and delivery
- f. Site preparation and construction
- g. Start-up
- h. Production

IV. Economic Analysis

A. Sections III and IV repeated for various alternatives

5. PLANNING COSTS

The cost of these studies varies substantially, depending upon the size and nature of the project, the type of study being undertaken, the number of alternatives to be investigated, and numerous other factors. However, the order of magnitude cost of the technical portion of studies, excluding such owner's cost items as exploration drilling, special grinding or metallurgical tests, environmental and permitting studies, or other support studies, is commonly expressed as a percentage of the capital cost of the project:

Conceptual study	: 0.1 to 0.3 percent
Preliminary study	: 0.2 to 0.8 percent
Feasibility study	: 0.5 to 1.5 percent

Metaliferous deposits that occur near the surface are mined by open pit methods. Quarrying and strip mining methods are used for other deposits such as sand, gravel, iron ore and coal. The advantages of open-pit mining over underground mining are lower costs, greater safety, and mechanically easier operations. These open pit mines range in sizes from "dog holes" to one half cubic mile. Many low-grade deposits are mined this way because of the lower costs and higher productivity.

Soil and barren rock must be removed to expose the ore bodies. This process is known as pre-production stripping. This process of stripping this surface away may take a short time or as long as years. A series of benches that are arranged in spiral or connecting ramps are developed. The ramps or benches are usually 25ft to 100ft wide, with heights being 25 to 70 feet. Equipment and stability of the rock determine the height and width of the each bench. Bench height and width are also related to slope stability. The slope can vary between 20% to 70%, with the limit being determined by slope stability and economics.

The amount of barren rock that is to be mined is the major consideration in open-pit mining planning.

Open Pit Mining

The basic concept of an open pit mine is simple. Open pit mines, however, require a lot of planning to make sure that as much ore as possible can be extracted. Note: the problems and cost of removing overburden often dictate the limits of the pit.

Mining is done at large pits by track-mounted electric shovels. Diesel-powered front end loaders are generally used at smaller operations. Truck, railroad, or conveyors usually do haulage.

Open Pit Mine- an excavation or cut made at the surface of the ground for the purpose of extracting ore and which is open to the surface for the duration of the mine's life. To expose or mine the ore, it is generally necessary to move large quantities of waste rock. The purpose of the mine is to make a profit so careful planning and engineering must take place from the very beginning. The planning of an open pit mine is an exercise in economics, constrained by geologic, and mining engineering aspects.

Increased population, created greater demand, which in turn, initiated increased productivity. Production increased with improved engineering and technology and for the most part, greater productivity by workers. The shift was from underground mining to surface mining even though the grade and quality was declining.

It is generally conceded that surface mining is more advantageous than underground mining in terms of recovery, grade control, economy, and flexibility of operation, safety, and the working environment. There are many deposits, that are too small or irregular, and or deeply buried to be extracted economically by surface mining methods. When the minerals extend deep in the ground, the removal of the waste rock becomes too cumbersome and expensive and the mine must be converted to underground operations or abandoned.

Open pit design is conducted in several stages.

- Devising a scheme or set of alternatives
- Evaluation
- Selection of optimum scheme

The most economic final pit design is often out of the hands of the designer. The design depends of factors such as:

- Geometric outline of the ore body,
- The topography
- Maximum allowable slope angles

In the end, the economics depend on the engineers' choice of plans, equipment and the mining ratio to production rates.

Exploration Input for Open Pit Planning

The data from the core samples is crucial at this point. Besides the information collected thus far being used to determine the shape and size of the ore body, these core samples help determine the slope stability. The water pressure plays an important role in the stability of the slope also.

The detrimental effects of the presence of ground water on surface mining are:

- Water pressure reduces the stability of the slope and tends to induce sliding of materials in the slope
- Increased weight of the unit rock thus higher rates for transportation.
- Freezing water can block drainage paths
- Erosion can cause instability and silting up of drainage systems
- Increased operating costs from discharge of water for equipment and blasting costs

Bench Plan Preparations

Technology plays an important role in planning and the design of a modern mine operation. A model representation is created called a block model or ore body model. The models enable mining planners to effectively select the most promising means of extracting the ore body both physically and economically. With the assistance of modern computer facilities, sophisticated and complex bench plans can be prepared.

Stripping Ratio Considerations

The term **stripping ratio** is almost universally used and represents the amount of uneconomic materials that must be removed to uncover one unit of ore. The ratio of total volume of waste to ore volume is defined as the **overall stripping ratio**.

$$R = \frac{\text{Volume of waste removed to depth } d}{\text{Volume of ore recovered to depth } d}$$

While a volume relationship, calculated in cubic yards/cubic yards (cubic meters/cubic meter). It is more commonly expressed as tons/ton. Note that in mining certain mineral commodities, however, *stripping ratio* is expressed in units of cubic yards/ton.

Cutoff stripping ratio is the one that the costs of mining the ore and waste are matched by the revenue for that block of ore. Factors used to determine costs should include the added costs of mining as the mine deepens and the interest charges on the prestripping of waste.

In the most complete analysis, the entire ore body is mined on paper. The production from each time period is determined, the costs and revenues listed, and a cash flow generated. The profits are projected. The result is to be the value of the mine or production. Mining is continued until it no longer increases the value, and so a pit limit is determined. The ratio of the total volume of waste to total volume of ore is then the ***overall stripping ratio***.

Major Steps in Surface Mining Development

Jones (1977), SME Mining Eng. Handbook, has outlined 10 major steps involved in planning and developing a surface coal mine. These steps can take up to 10 years and require millions of dollars of expenditure exclusive of that for actual mine preparation and equipment purchase.

I. Assembly of the Mining Coal Package

1. Leasing 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 centers (development drilling)
8. Sampling, logging analysis
9. Surface Acquisition

II. Market Development
- 1. Market survey
- 2. Potential customer identification
- 3. Letter of intent to develop and supply
- 4. Contact negotiation

III. Environmental and Related Studies
- 1. Initial reconnaissance
- 2. Scope of work development
- 3. Implementation
- 4. Environmental impact report
- 5. 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. NEPA Process (National Environmental Policy Act of 1969)
- Identification of lead agency for Environmental Impact Statement (EIS)
- Draft EIS
- EIS review and comments
- EIS hearing and record
- Federal EIA review
- Council on Environmental Quality filing
- Mining and /or reclamation plan approval

VI. Permits
- State water well rights appropriation permits
- State mining permit
- State industrial siting permit
- Federal NPDES permit
- US Forest Service special land use permit

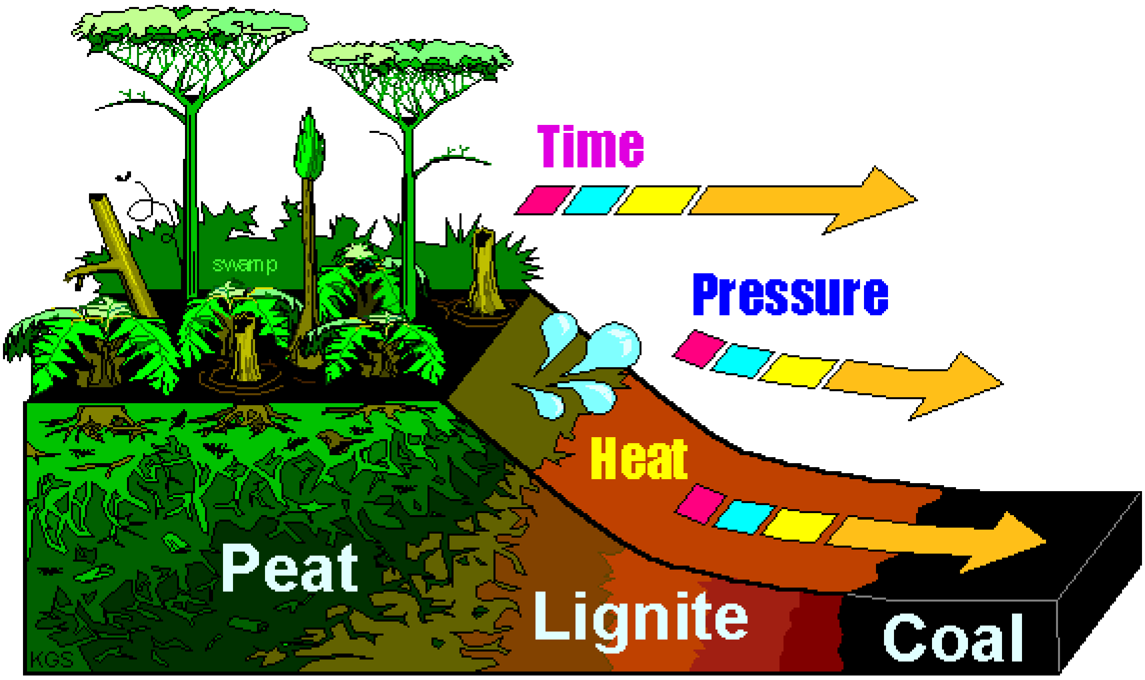
VII. Design and Construction
- Preliminary design and estimation
- Material ordering and contracting
- Water well development
- Access road and site preparation
- Railroad construction
- Power supply and installation
- Facilities and coal handling construction
- Warehouse building and yards
- Coal preparation and loading facilities construction
- Overland conveyor construction

VIII. Mining Preparation
- Stripping machine
- Loader erection
- Support equipment readying
- Manpower recruitment and training

IX. Production Buildup

X. Full Production

How is Coal Formed?

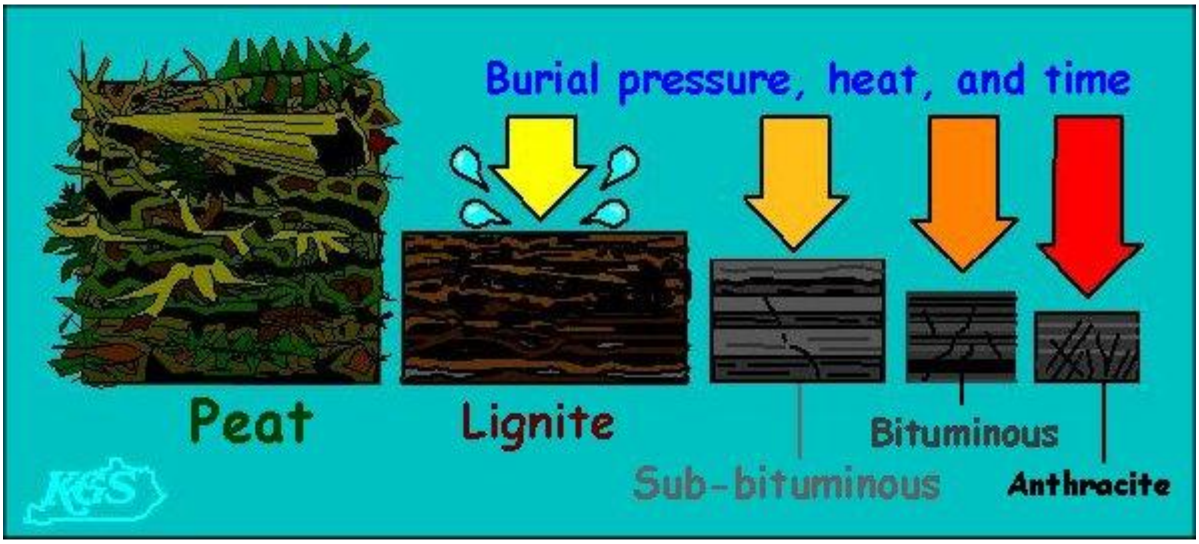


Coal is formed when peat is altered physically and chemically. This process is called "coalification." During coalification, peat undergoes several changes as a result of bacterial decay, compaction, heat, and time. Peat deposits are quite varied and contain everything from pristine plant parts (roots, bark, spores, etc.) to decayed plants, decay products, and even charcoal if the peat caught fire during accumulation. Peat deposits typically form in a waterlogged environment where plant debris accumulated; peat bogs and peat swamps are examples. In such an environment, the accumulation of plant debris exceeds the rate of bacterial decay of the debris. The bacterial decay rate is reduced because the available oxygen in organic-rich water is completely used up by the decaying process. Anaerobic (without oxygen) decay is much slower than aerobic decay.

For the peat to become coal, it must be buried by sediment. Burial compacts the peat and, consequently, much water is squeezed out during the first stages of burial. Continued burial and the addition of heat and time cause the complex hydrocarbon compounds in the peat to break down and alter in a variety of ways. The gaseous alteration products (methane is one) are typically expelled from the deposit, and the deposit becomes more and more carbon-rich as the other elements disperse. The stages of this trend proceed from plant debris through peat, lignite, sub-bituminous coal, bituminous coal, anthracite coal, to graphite (a pure carbon mineral).

Because of the amount of squeezing and water loss that accompanies the compaction of peat after burial, it is estimated that it took 10 vertical feet of original peat material to produce 1 vertical foot of bituminous coal in eastern and western Kentucky. The peat to coal ratio is variable and dependent on the original type of peat the coal came from and the rank of the coal.

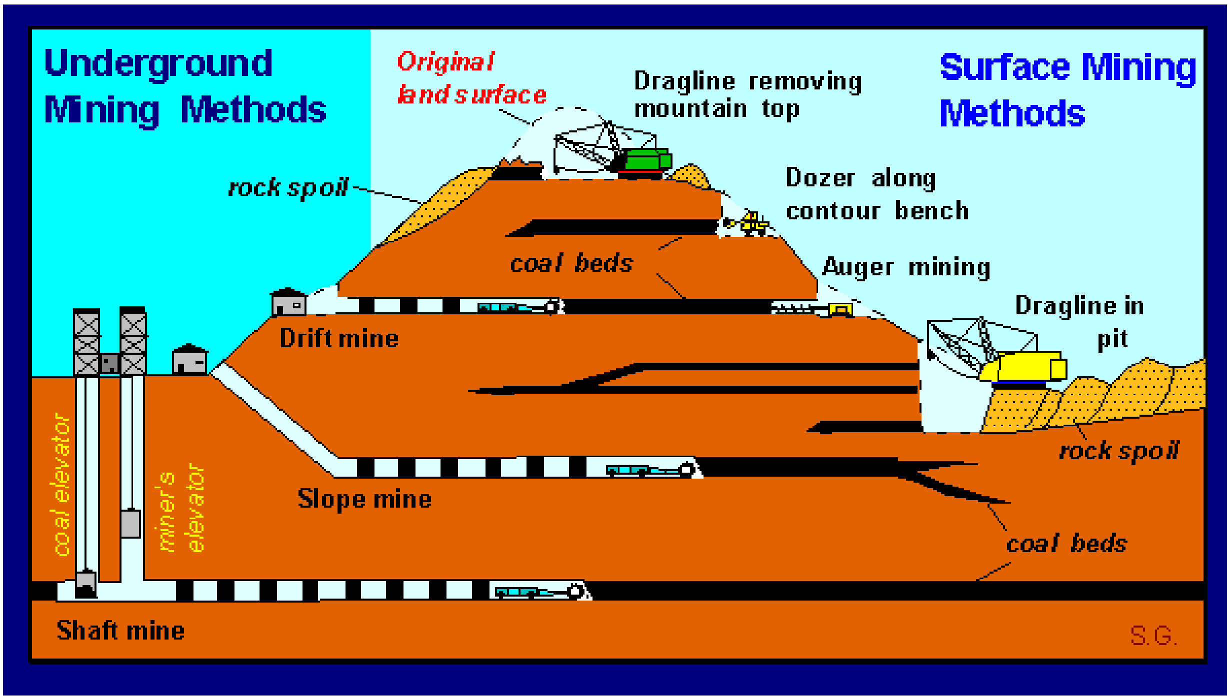
Classification and Rank of Coal



The kinds of coal, in increasing order of alteration, are lignite (brown coal--immature), sub-bituminous, bituminous, and anthracite (mature). Coal starts off as peat. After a considerable amount of time, heat, and burial pressure, it is metamorphosed from peat to lignite. Lignite is considered to be "immature" coal at this stage of development because it is still somewhat light in color and it remains soft. As time passes, lignite increases in maturity by becoming darker and harder and is then classified as sub-bituminous coal. As this process of burial and alteration continues, more chemical and physical changes occur and a the coal is classified as bituminous. At this point the coal is dark and hard. Anthracite is the last of the classifications, and this terminology is used when the coal has reached ultimate maturation. Anthracite coal is very hard and shiny.

The degree of alteration (or metamorphism) that occurs as a coal matures from peat to anthracite is referred to as the "rank" of the coal. Low-rank coals include lignite and sub-bituminous coals. These coals have a lower energy content because they have a low carbon content. They are lighter (earthier) and have higher moisture levels. As time, heat, and burial pressure all increase, the rank does as well. High-rank coals, including bituminous and anthracite coals, contain more carbon than lower-rank coals which results in a much higher energy content. They have a more vitreous (shiny) appearance and lower moisture content then lower-rank coals.

Methods of Mining



According to the Kentucky Department of Mines and Minerals, 131.8 million tons of coal was mined in Kentucky in 2000; 62 percent (81 million tons) was from underground mines and 38 percent (50 million tons) was from surface mines. There were 264 active underground mines and 240 active surface mines in Kentucky in 2000.

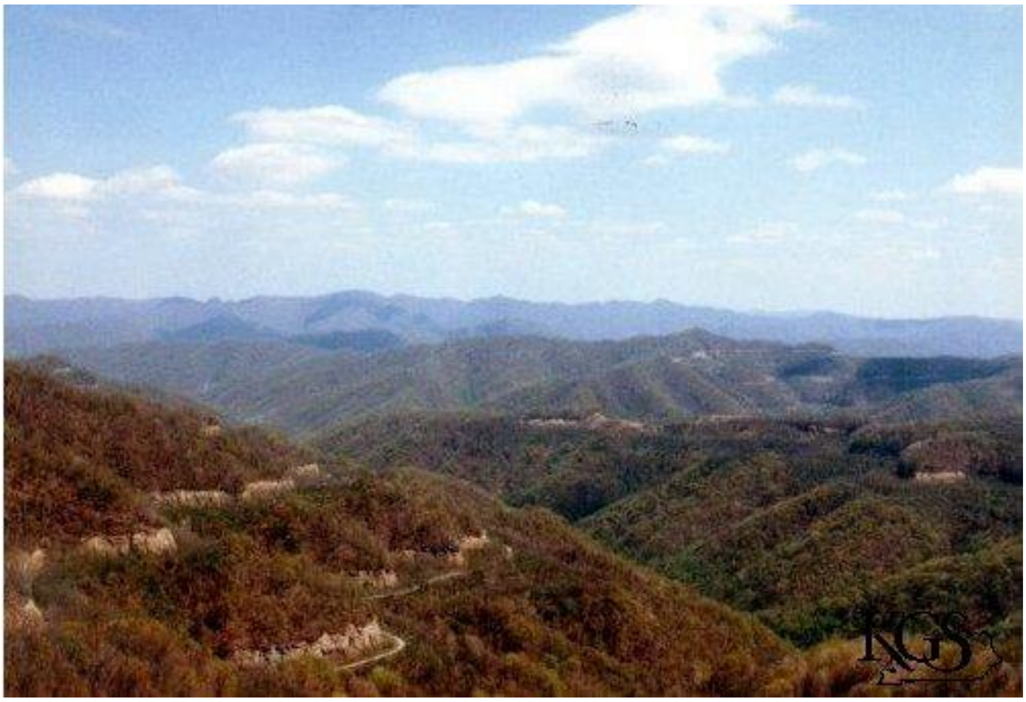
Underground Mining

Underground modes of access include drift, slope, and shaft mining, and actual mining methods include longwall and room and pillar mining. Drift mines enter horizontally into the side of a hill and mine the coal within the hill. Slope mines usually begin in a valley bottom, and a tunnel slopes down to the coal to be mined. Shaft mines are the deepest mines; a vertical shaft with an elevator is made from the surface down to the coal. In western Kentucky, one shaft mine reaches 1,200 feet below the surface.

Surface Mining

Surface-mining methods include area, contour, mountaintop removal, and auger mining. Area mines are surface mines that remove shallow coal over a broad area where the land is fairly flat. Huge dragline shovels commonly remove rocks overlying the coal (called overburden). After the coal has been removed, the rock is placed back into the pit. Contour mines are surface mines that mine coal in steep, hilly, or mountainous terrain. A wedge of overburden is removed along the coal outcrop on the side of a hill, forming a bench at the level of the coal. After the coal is removed, the overburden is placed back on the bench to return the hill to its natural slope. Mountaintop removal mines are special area mines used where several thick coal seams occur near the top of a mountain. Large quantities of overburden are removed from the top of the mountains, and this material is used to fill in valleys next to the mine. Augur mines are operated on surface-mine benches (before they are covered up); the coal in the side of the hill that can't be reached by contour mining is drilled (or augured) out. Drift, contour, mountaintop removal, and augur mining are more common in the Eastern Kentucky Coal Field, and area, slope, and shaft mining are more common in the Western Kentucky Coal Field.

Area mining



Contour mining

DEVELOPMENT DRILLING AND BULK SAMPLING

1. PURPOSE

Development drilling is directed towards determining the grade, volume, and three-dimensional outline of a mineralized zone previously located by exploration. It is distinguished from exploration drilling, which has an objective the discovery of new mineralized areas.

A development drilling and bulk sampling program should furnish the following information:

1. Geology of the mineralized zone.
2. Quantitative data on grade and tons of material within pertinent cutoff limits.
3. Physical size and shape of the deposit.
4. Mineralogical and metallurgical characteristics of the ore.
5. Physical characteristics of the ore.
6. Bulk samples for metallurgical testing and grade check.
7. Data on other factors that could affect mining operations, such as ground water, ground conditions, etc.

2. PROCEDURES

There are numerous methods and techniques that are used to acquire pertinent data on a potential ore deposits. The total cost and accuracy of a development program depends on significant evaluation of the geology, proper selection of a drilling method, through analysis of sample data, and accurate evaluation of all the information.

Knowledge of the geology of the mineral deposit is important in planning the development testing program and in evaluating the data obtained from such a program. Exploration work usually provides some significant information on the general size, shape, thickness, grade, and geology of the deposit. In some cases no further data are needed; in many cases geological complexities suggested by the preliminary work require further geological study. Of considerable importance may be the following:

1. Pertinent geologic framework of the deposit.
2. Variations in grade within mineralized area.
3. Distribution pattern and mineralogical of the economic minerals.
4. Attitude of the ore zone(s).
5. Physical characteristics of the ore and waste.
6. Relation of mineralization to structure, weathering cycles, rock types, alteration, etc.
7. Distribution of ground water.

When the significant geologic features of the mineral deposit are known, a selection can be made of the type of drilling equipment required to obtain the kind of samples that can provide the data for calculation of grade and tons of the possible orebody.

Hole size and hole spacing require evaluation. As much as ore reserve calculations start from drill-sample data, the samples must be representative of the area assigned them in any calculation. In any drilling program it is important that the pertinent data be accurate and assembled in usable form.

Bulk samples of a deposit are taken to provide large representative samples for metallurgical test work and to provide a grade check against drill assays or averages. In the first case, the geology of the deposit and the distribution of values will indicate where representative metallurgical samples should be taken. In the second case, geologic and assay data on grade variations will determine where sample checks should be made.

Ore reserve calculations follow confirmation of grades obtained from drill data. The calculation method used will depend in part on the variations in grade, relation of geology to grade changes, the accuracy required for economic appraisal, etc.

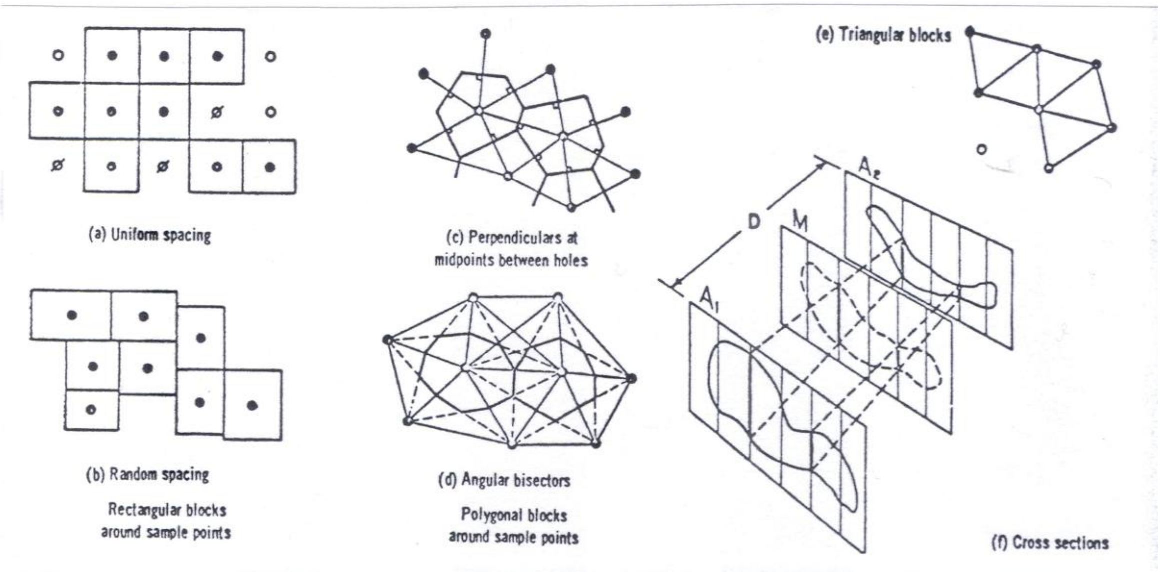
Statistical and mathematical techniques are important in the planning of development drilling, in analyzing the resultant sample data, in establishing confidence levels, etc. Such methods are most useful if the outline of the orebody is well defined (no alternate choices), geological continuity can be expected between drillholes, and the drill samples are representative. Two methods of handling the data statistically are generally used. One method analyzes the assay values regardless of their location or point of origin (unlocated assay). The second method uses the X, Y, and sometimes Z coordinate for each assay (located assay).

3. ORE RESERVE CALCULATIONS

A calculation of grade and tons in a mineral deposit is usually made by an analysis of sample data framed in a polygonal, triangular, cross-sectional, or other modified geometric pattern.

3.1. CONVENTIONAL METHODS

Following figure shows the geometric patterns generally used in assigning areas of influence to drillhole samples.



Geometric patterns used in assigning areas of influence to drillhole samples.

Sketch (a) and (b) illustrate a system of rectangular blocks around each drillhole with the influence of each hole extending halfway to the adjacent holes. Sketch (c) and (d) polygons constructed around each drillhole with influence of each hole extending halfway to the adjacent hole.

In the rectangular and polygonal methods of reserve calculations, it is assumed that the grade of an assay in one drillhole extends halfway to any adjacent drillhole. Obviously, this is an incorrect model of the true situation that exists for the mineralization at the midpoint between the two drillholes. Even if the mineralization were to consist of pod- and lens-like segments, the exploration drilling would not penetrate exactly at the center of each pod or lens, and the drillhole spacing would not equal the pod or lens lengths. In situations where other types of mineralization are present, the polygonal method serves as only a rough approximation of the true grade of ore within its area of influence.

Sketch (e) illustrates the triangular method of estimating grade of ore between drillholes. This method is also an approximation model, because it assumes a linear change of grade of ore in direct proportion to the distance between drillholes.

In the cross-sectional method, sketch (f), it is necessary to draw sections through the orebody, and then to subdivide these sections into areas or blocks for which grade and tonnage estimates can be assigned.

3.2. STATISTICAL METHODS

In the past few years, statistical methods have been used extensively in estimating grade of ore. Techniques are available to permit computation of a confidence interval for an estimate of grade of ore when statistical methods are used.

Various forms of statistical regression analysis are being used to build models of ore deposits which are made up of combinations of mathematical formulas for curves. In many instances, these models tend to represent changes in ore grade better than the polygonal and triangular models.

Statistical response surface techniques can be used effectively for guiding exploration, evaluating grade and tonnage, and for determining pit limits of open pit mining.

ORE RESERVE CALCULATION

1. INTRODUCTION

The estimation of ore reserves is a process that begins with the earliest exploration stages on a property and continues throughout any subsequent evaluation and exploitation of the deposit. During exploration and preliminary evaluation, the results of these reserve estimates constitute the basic data for prefeasibility studies and economic analysis. The decision to continue exploration and development or to abandon a prospect is often based upon these studies.

During the active life of a mine, reserve computations are continuously revised to assist in development planning, cost and efficiency analyses, quality control, and improvement of extraction and processing methods. Accurate reserve estimates are also required when financing a project, purchasing or selling a property, and for accounting purposes such as depletion and tax calculation.

It is important to remember that the reliability of ore reserve estimates varies progressively through time as more and more information becomes available. The lowest order of reliability of estimation of reserves exists at the time of discovery. The maximum level of certainty concerning the ore reserves within a deposit is reached when the deposit is completely mined out. Between these two extremes are variable levels of certainty as to the tonnage and grade of the resource.

In the following discussion, several of the factors affecting ore reserve computation and some of the commonly used methods of calculation are presented. The first part of the discussion is confined to classical methods of hand calculation utilizing level maps and sections.

2. CLASSIFICATION OF ORE RESERVES

Ore reserves are classified with respect to the confidence level of the estimate. Traditionally, ore reserves have been classified as proven (measured), probable (indicated), possible (inferred). Historically, proven ore has been regarded as that which is "blocked out" (i.e., measured, sampled, and assayed on four sides); probable ore as blocked on three sides; possible ore as blocked all two sides; and inferred ore as ore-grade material that is known on only one side.

The U.S. Bureau of Mines (USBM) has introduced the following ore reserve classification:

Measured Ore

Measured ore is ore for which tonnage is computed from dimensions revealed in outcrops, trenches, workings, and drill holes, and for which the grade is computed from the results of detailed sampling. The sites for inspection, sampling, and measurement are so closely spaced and the geologic character is so well-defined that the size, shape, and mineral content are well established.

Indicated Ore

Indicated ore is ore for which tonnage and grade are computed partly from specific measurements, samples, or production data and partly from projection for a reasonable distance on geologic evidence. The sites available for inspection, sampling, and measurement are too widely or otherwise inappropriately spaced to outline the ore completely or to establish its grade throughout.

Inferred Ore

Inferred ore is ore for which quantitative estimates are based largely on a broad knowledge of the geologic character of the deposit and for which there are few, if any, samples or measurements. These estimates are based on an assumed continuity or repetition for which there is geologic evidence.

By the time a deposit is ready for development, there usually exist two ore reserve estimates: a geologic reserve or total resource estimate, and a mining ore reserve. The geologic reserve is an estimate including all known mineralization above a certain grade within the deposit. However, the geologic reserve figure may not be associated with a specific mining cutoff grade. The mining reserve constitutes that portion of the geologic reserve that can be mined and processed at a profit. The mining reserve is always less than or equal to the geologic reserve estimate because a variable proportion of the orebody must be left unmined for a variety of reasons. These reasons include the need for pillars for ground support, metallurgical problems, width of mineralization, or other economic and engineering factors.

3. ORE RESERVE PARAMETERS

An ore reserve estimate contains two important parameters: the amount of ore and the average grade or value of that ore. In metal mines, the amount of ore is usually expressed in either metrictons (1000 kg) or short tons (2000 lb).

Grades are normally expressed as a percentage for base metal ores, whereas precious metals may be reported as troy ounces per ton, pennyweights per ton, or grams/metric ton.

The calculation of the tonnage and grade of a deposit requires the collection and documentation of a considerable amount of data. These data include accurate assay information, plans and sections, details of ore controls, the tonnage factor, applicable cutoff grade to be used, potential mineral recovery, and engineering details such as minimum mining width and anticipated dilution. These items are discussed in the following sections.

3.1. Grade Determination

The average grade of an ore deposit or of a specific block within a deposit is based on assays of samples collected within the block or deposit. Sample collection, preparation, and analysis are often the most critical operations in evaluating the reserves for a mineral property. Sampling theory and practice constitute a complex subject in their own right and only some of the more important points are touched upon in the following summary.

Cutoff Grade: Associated with the definition of ore grade is the concept of cutoff grade. The cutoff grade is the minimum ore grade that can be mined at a profit under economic conditions existing at a particular point in time. The cutoff grade can vary with time due to changes in such factors as commodity prices, operating costs, and taxes. The cutoff grade used for any reserve calculation should always be stated.

Sampling: Sampling of an ore deposit is a process of approximation. The objective is to arrive at an average sample value that most closely represents the true average value for the body in question. The importance of attention to detail during the sampling program becomes apparent when it is realized that in the case of a very well sampled block of ore from a vein-type deposit, the actual sample volume may represent only about 0.25% of the block. In other cases, such as the sampling of a porphyry copper deposit by diamond drilling, the sample volume may constitute only about 0.004% of the orebody. To obtain the most accurate grade estimation, it is imperative that the sampling crews and procedures be carefully monitored by members of the geological and engineering staff.

Samples are usually collected at constant intervals down the length of the core, although samples may be taken at shorter intervals through highly mineralized areas or veins. Often, parts of the core known to be barren, or without visible evidence of mineralization, are not split or assayed. In any drilling program, there likely will be areas where drilling is difficult and core recovery is poor. In these zones, it is common practice to collect samples of the drilling fluid, or sludge.

Assaying: Assaying may be done by a commercial laboratory or by an in-house company lab. In any case, a certain percentage of the samples, usually a minimum of 10%, should be assigned a new sample number and resubmitted for a repeat analysis to provide a check on the analytical precision of the laboratory. It is also recommended practice to send a percentage of the samples to a different laboratory for accuracy comparison.

Statistical Analysis of Sample Data: When the assays have been received from the laboratory and the validity of the results has been satisfactorily established, it is often useful to make some simple statistical analyses of the data. Classical statistical techniques are based on two assumptions: that the samples are random and that the data have a normal distribution. The problem of obtaining random samples is somewhat difficult to analyze. Samples collected from an ore deposit are seldom statistically independent of one another.

3.2. Sample Weighting

It is often necessary to compute a value for a composite sample, developing a weighted average for unequal sample widths or lengths. The method of such weighted sample calculation is illustrated in Fig.1, which shows a series of samples collected from a vein exposure in a raise in a hypothetical lead-silver mine. Note that the samples are collected at right angles to the dip of the bed and, in this case, the vein is assumed to extend at right angles to the plane of the page. If for some reason it is impossible to collect samples across the true width of the vein, the measured width should be corrected to the true width by a simple trigonometric calculation, as the true width will be needed for tonnage calculations. Normally, samples are collected at regular intervals, with the interval dependent on the nature of the mineralization. Erratic mineralization, such as is common in epithermal gold-silver deposits, requires sampling at much closer spacing than more regular deposits. Here, samples have been shown at irregular spacing to illustrate the principle of weighting for area of influence. For purposes of calculation, see Table 1.

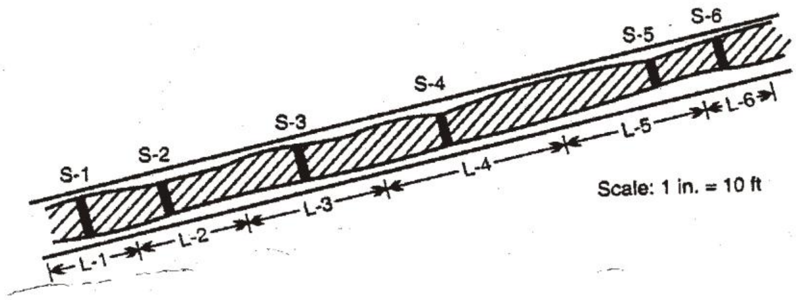


Fig.1 Sample plan of a silver-lead vein.

Table 1. Calculation of weighted average.

Sample No.	Area of Influence, L, ft*	Width, W, ft	W × L, ft	% Pb	Oz* Ag/t	Pb × W × L	Ag × W × L
S-1	6.0	3.0	18.0	6.4	11.3	115.20	203.40
S-2	7.5	2.5	18.75	7.6	14.7	142.50	275.63
S-3	10.0	3.0	30.00	5.6	8.6	168.00	258.00
S-4	12.5	2.3	28.75	8.8	12.9	253.00	370.88
S-5	10.0	2.0	20.0	8.2	13.7	164.00	274.00
S-6	5.0	2.6	13.0	6.7	10.8	87.10	140.40
Total	51.00	15.40	128.50			929.80	1522.31

$$\text{Average grade Pb} = \frac{929.80}{128.50} = 7.2\%\ \text{Pb}$$

$$\text{Average grade Ag} = \frac{1522.31}{128.5} = 11.9\ \text{oz Ag per st*}$$

$$\text{Average thickness} = \frac{128.50}{51.0} = 2.52\ \text{ft}$$

4. Tonnage Determination

The calculation of tonnage for an ore deposit requires that the volume of the mineralized zone and the tonnage conversion factor be known.

4.1. Volume Calculation: The volume of the mineralized zone is calculated by measuring the area of the mineralization and multiplying the area by the corresponding thickness of material above cutoff grade. The area may be estimated by breaking the area into small, regular geometric shapes and calculating the total area by geometry. For very irregular or curved areas, the area is most easily determined by planimetering.

Measuring areas by planimetering is an imponant part of many ore reserve estimations. Unfortunately, all too often insufficient care or attention to detail is given to this aspect of reserve estimation. There are several problem areas in planimetering. One of the major problems is that for smaller areas, small measurement errors can become of sufficient magnitude to seriously affect the area measurement and thus the reserve calculation. Another frequently encountered problem is an erroneous reading of the planimeter. Both these problems are essentially eliminated by the planimetering of all areas at Ieast three times.

4.2. Tonnage Factor Calculation: The tonnage factor provides the mechanism for the conversion from volume of ore to weight of ore. In the English system, the tonnage factor is normally expressed as cubic feet per ton of ore. In the metric system, the tonnage factor is the specific gravity of the ore. The tonnage factor is dependent upon the specific gravity of the ore, and the specific gravity is a function of the mineral composition of the ore. Probably the most accurate method of determining specific gravity of an ore is to calculate an average specific gravity using specific gravities of individual minerals (Table 1.4), provided the relative percentages of ore minerals present are accurately known. For example, if a massive sulfide ore is 10% galena, 35% sphalerite, and 55% pyrite, the specific gravity would be:

$$
\begin{aligned}
7.6 \times 0.10 &= 0.76 \\
4.1 \times 0.35 &= 1.44 \\
5.0 \times 0.55 &= 2.75 \\
&\ 4.95 = \text{sp gr of ore}
\end{aligned}
$$

The specific gravity of an ore may also be computed by weighing a core or specimen of the ore in air, then weighing the same sample suspended in water. The specific gravity is calculated by the following formula:

$$\text{Sp.gr.} = \frac{W_a}{W_a - W_w}$$

where W_a = weight in air and W_w = weight in water.

Table 2. Specific gravity of common rocks and minerals

	Specific Gravity		Specific Gravity
Rocks		**Minerals *(continued)***	
Andesite	2.4–2.8	Chromite	4.5
Basalt	2.7–3.2	Copper	8.8
Diabase	2.8–3.1	Covellite	4.6
Dolomite	2.7–2.8	Cuprite	6.0
Gabbro	2.9–3.1	Feldspar	2.6–2.8
Granite	2.6–2.7	Fluorite	3.1
Gravel (dry)	1.6–2.0	Galena	7.6
Limestone	2.7–2.8	Gold	17.5
Rhyolite	2.2–2.7	Graphite	2.2
Sandstone	2.0–3.2	Gypsum	2.3
Schist	2.6–3.0	Hematite	5.2
Shale	1.6–3.0	Molybdenite	4.8
		Muscovite	2.9
Minerals		Pentiandite	4.8
Anglesite	6.3	Platinum	19.0
Anhydrite	2.9	Pyrite	5.0
Argentite	7.3	Pyroxene	3.3
Arsenopyrite	6.0	Pyrrhotite	4.7
Barite	4.5	Quartz	2.7
Bauxite	4.5	Scheelite	6.0
Bornite	4.9	Sericite	2.6
Calcite	2.7	Silver	10.6
Cassiterite	7.0	Smithsonite	4.4
Cerussite	6.5	Sphalerite	4.1
Chalcedony	2.6	Stibnite	4.6
Chalcocite	5.7	Sulfur	2.1
Chalcopyrite	4.3	Uraninite	9.4

If the ore volume has been computed in cubic meters, the volume multiplied by the specific gravity is the tonnage in metric tons directly. If working in the English system, the tonnage factor is calculated as follows:

Sp.gr. * 62.5 (lb per cu ft water) = lb per cu ft ore

$$\text{Tonnage factor} = \frac{\text{2000 lb per ton}}{\text{lb per cu ft ore}} = \text{cu ft per ton ore}$$

For example, if a porphyry copper ore has a specific gravity of 2.8, then:

2.8 x 62.5 (lb per cu ft of water) = 175 lb per cu ft ore

Tonnage factor = 2000/175 = 11.43 cu ft per ton ore

For purposes of ore reserve estimation, a single or even a few samples of core or ore specimens would not be suitable. Specific gravity determinations would be made of both the mineralization and gangue from many drill hole and other samples.

5. Engineering Considerations

Before proceeding with an explanation of various methods of reserve computation, a brief discussion of pertinent engineering factors is in order.

5.1. Geological Considerations: In many instances, particularly in the exploration stage of a project, it is common practice to project ore extensions based on geologic inference. These projections should never be extended across geological discontinuities such as faults, contacts, unconformities, fold axes, etc. until positive ore correlation data are available on both sides of the discontinuity. Preliminary drilling and other sampling will give an indication of the nature of ore boundaries, whether they are sharply defined or gradational. Lateral or vertical mineralogical zonation, development of discrete ore shoots, and other potential problems will become apparent as exploration work progresses.

5.2. Mining and Metallurgical Recovery: Once the deposit is reasonably well defined as to its limits, shape, and character, consideration can be given to selection of an appropriate mining method, and then an estimate can be made concerning percentage extraction from the deposit. The portion of mineralization above the cutoff value that can actually be exploited constitutes the minable ore reserve .

5.3. Dilution: Dilution is the unavoidable extraction of barren or below cutoff grade material along with the ore. In vein deposits the most common source of dilution is blasting overbreak in the walls of the deposit. Dilution may be handled in various ways. In veins that have at least one gradational or "assay" wall, it is common to cut samples over the normal mining width and use an average grade and tonnage factor for the entire interval. In the case of narrow veins with sharp boundaries, when wall rock must be taken with the vein, the dilution may be calculated as follows:

Ore block	: 30.48 x 15.25 m (100 x 50 ft)	
Average width	: 0.6 m (2.0 ft)	
Average grade	: 10.0% Pb	
Tonnage factor	: 9.0 cu ft per ton ore	
	12.0 cu ft per ton wall rock	
Minimum mining width	: 0.9 m (3 ft)	
Ore tons	: 100 x 50 x 2.0 ÷ 9	= 1111 tons
Waste tons	: 100 x 50 x 1.0 ÷ 12	= 417 tons
	Total tons mined	= 1528 tons
Grade	: 1111 x 10.0%	= 11,110
	417 x 0.0%	= 00,000
		11,110

Diluted grade = 11110 / 1528 = 7.27% Pb

In some instances, it is possible for dilution to drop a block of ore grade material below cutoff grade. In such instances, the block economically ceases to be ore and must be left until the cutoff grade is lowered or greater selectivity in mining can be made.

5.4. Cutoff Grade: As stated before, the cutoff grade is the minimum grade that can be mined at a profit. As economic conditions change, the cutoff grade may increase or decrease. It is common practice to compute the ore reserves of a mine for various cutoff grades and plot the results as a series of grade-tonnage curves. These curves should be updated regularly to aid in mine planning.

6. METHODS OF CALCULATION

In this section, several of the common traditional reserve calculation methods are explained and illustrated by simple examples. The calculation methods discussed are by mining block, by polygons, by triangles, and by section. Many of the other calculation methods presented in the literature are only somewhat sophisticated variations of these methods. No one technique is universally applicable to all deposits. Methods such as mining blocks and sections work well in steeply dipping veins and tabular deposits, whereas polygonal methods have found wide application to disseminated and flat-lying bedded deposits. The method selected for any particular deposit depends upon the geological and engineering elements unique to each deposit, and usually the ore reserves will be calculated several different ways.

6.1. Calculation by Mining Block

Figure 2 shows the method of estimating the tonnage and grade of an ore block in a vein-type mine. Samples have been assumed to be cut at regular intervals. The vein in this area is assumed to exceed the minimum mining width of 0. 91 m (3 ft) .

Figure 2 represents a block of ore between two levels 30.5 m (100 ft) apart, and coordinates 00E and 250E. The upper figure in the block is the average thickness and the lower figure is the average grade in ounces of silver per ton. The tonnage factor is 9 cu ft per ton. The average grade and tonnage for the block are computed in Table 3.

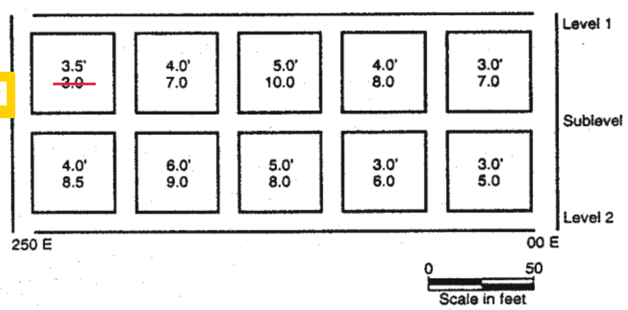


Fig.2. Longitudinal projection of an ore block.

Table 3. Calculation of ore blocks

Length		Width		Thickness	Tonnage Factor		Tons*		Grade		Tons-Grade
50	×	50	×	3.5	9	=	972	×	5	=	4,861
50	×	50	×	4.0	9	=	1,111	×	7	=	7,777
50	×	50	×	5.0	9	=	1,388	×	10	=	13,889
50	×	50	×	4.0	9	=	1,111	×	8	=	8,888
50	×	50	×	3.0	9	=	833	×	7	=	5,833
50	×	50	×	3.0	9	=	833	×	5	=	4,166
50	×	50	×	3.0	9	=	833	×	6	=	5,000
50	×	50	×	5	9	=	1,388	×	8	=	11,111
50	×	50	×	6	9	=	1,666	×	9	=	15,000
50	×	50	×	4	9	=	1,111	×	8.5	=	9,444
							11,246				85,972

Total tons = 11,246

Average grade = $\frac{85,972}{11,246}$ = 7.64 oz* Ag/ton

*Metric equivalents: oz × 0.02834952 = kg; st × 0.9071847 = t.

6.2. Calculation by Polygons

The method of calculation by polygons is often used with drill-hole data. Polygons may be constructed on plans, cross sections, or longitudinal sections. The polygons, once constructed and ranked as to class of ore, are planimetered to determine the area of mineralization. The thickness of above cutoff grade mineralization is applied to the entire polygon to establish the volume estimate. In this method, the average grade of mineralization encountered by the sample point within the polygon is considered to accurately represent the grade of the entire volume of material within the polygon. The construction of polygons is quite simple. The method assumes that the area of influence of any sample point extends halfway to the adjacent points. The procedure for construction of polygons is illustrated in Fig.3.

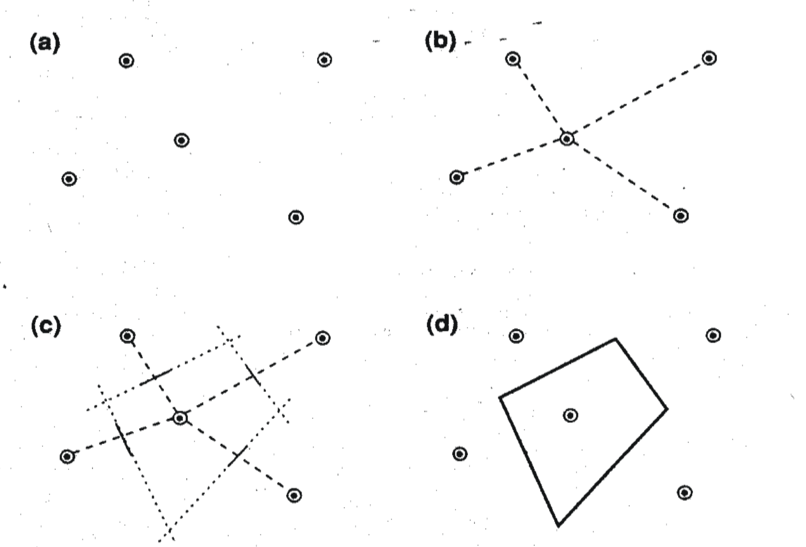


Fig.3. Construction of polygons (a)Drill-hole plan (b) connecting lines for the drill holes (c) the construction of perpendicular bisectors of the lines between adjacent drill holes (d) Construction of final polygon.

The polygon method makes the basic assumption that the area of influence of a drill hole extends halfway to the next adjacent hole: An alternative view-point is that of a circular area of influence for drill-hole intercepts. The concept of circular area of influence about a mineralized intercept can be used to assign the relative classification to the polygon blocks. One important aspect of the circular area of influence for drill holes, shown in Fig.4, is completely covering the area contained within a square grid of holes by the circular area of influence of the holes requiring a drill-hole spacing of (r)*(√2), where (r) is the radius of influence.

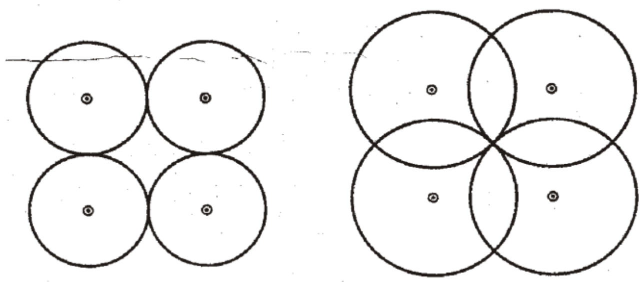


Fig.4. Polygon classification by circular area of influence.
Left, incorrect method; right, correct method.

Once the area of influence of the drill holes corresponding to proven and probable ore has been determined, it can be used to check and rank the polygons. Each drill-hole polygon is superimposed on the center of the "proven" area of influence circle matching the center points. If the polygon falls completely within the "proven" range circle, the polygon is marked as belonging to the proven category. Subsequent checks are made of the polygons not meeting the criteria to be classified as proven blocks using the other area of influence circles until all the polygons are ranked. For record and bookkeeping purposes, polygons are conveniently referenced to the drill-hole number and the section or level being evaluated (e.g., polygon DDH-8-16, level 2080). Figure 5 illustrates the computation of an ore block by the polygon method.

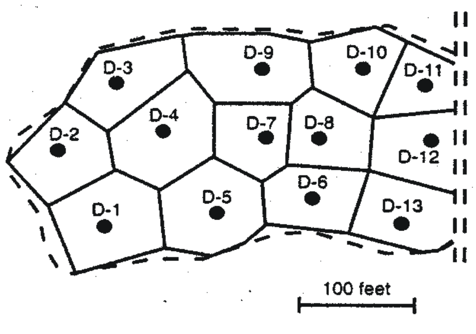


Fig.5. Diamond drill-hole plan of the Bonanza copper deposit.

The method of calculation by polygons is often used with drill sample data. The method makes the assumption that the area of influence of each drill hole extends half the distance to each adjacent drill hole, with appropriate modifications for known geologic factors such as faults, contacts, or mineralization limits. The areas of the polygons may be measured by planimeter or calculated geometrically by breaking up each polygon into a series of triangles. The average grade and thickness of each drill hole may be determined as shown in Table 4 for drill hole D-1. In this example, the mineralization is assumed to be copper, and the cutoff grade is 0.40% Cu.

Table 4. Assay data for drill-hole D-1.

TABLE 1.6 Assay data for drill hole D-1

Interval, ft*	Thickness, ft	Grade, % Cu	Grade × Thickness	
0–100	100	0.31	0.00	(Below cutoff)
100–110	10	0.47	4.70	
110–122	12	0.73	8.75	
122–130	8	0.96	7.68	
130–150	20	1.04	20.80	
150–200	50	0.82	41.00	
200–220	20	0.54	10.80	
220–250	30	0.42	12.60	
250–270	20	0.35	0.00	(Below cutoff)
	150		106.33	Thickness and grade-thickness above cutoff

$$\text{Average grade} = \frac{106.33}{150.00} = 0.71\%\ \text{Cu}$$

Thickness = 150 ft

*Metric equivalent: ft × 0.3048 = m.

The average grade and thickness is determined for each drill hole, and the reserves are calculated as shown on Table 5. Each polygon is labeled for the contained drill hole.

Table 5. Ore reserves for Bonanza copper deposit.

TABLE 1.7 Ore reserves for Bonanza copper deposit

Polygon	Area, A, sq ft	Thickness, T, ft	A × T, cu ft	Tonnage Factor/TF, cu ft*/ton	(A × T)/TF, tons ore	Grade, % Cu	Ton × Grade, ton %
D-1	5,320	150	798,000	12.5	63,840	0.71	45,326
D-2	5,300	135	715,500	12.5	57,240	0.66	37,778
D-3	4,400	180	792,000	12.5	63,360	0.82	51,955
D-4	5,520	175	966,000	12.5	77,280	0.75	57,960
D-5	6,800	155	105,400	12.5	84,320	1.00	84,320
D-6	4,960	180	892,800	12.5	71,424	0.97	69,281
D-7	4,520	250	1,130,000	12.5	90,400	1.21	109,384
D-8	4,640	240	1,113,600	12.5	89,088	1.36	121,159
D-9	5,840	150	876,000	12.5	70,080	0.93	65,174
D-10	4,840	135	653,400	12.5	52,272	0.87	45,476
D-11	3,760	120	451,200	12.5	36,096	0.81	29,237
D-12	4,270	165	637,200	12.5	50,976	0.75	38,232
D-13	4,800	135	648,800	12.5	51,840	0.68	35,251
					858,216		790,553

Tons ore 858,216

Average grade 0.92% Cu

*Metric equivalents: ft × 0.3048 = m; sq ft × 0.09290304 = m^2; cu ft × 0.02831685 = m^3; st × 0.9071847 = t.

6.3. Computation by Triangles

Another method of computing reserves is a modification of the polygon method. In this method a series of triangles is constructed with the drill holes at the apices. This method has the advantage in that the three points are considered in the calculation of the thickness and grade parameters for each triangular reserve block. The construction and calculation of ore reserves by use of triangles are shown in Fig. 6.

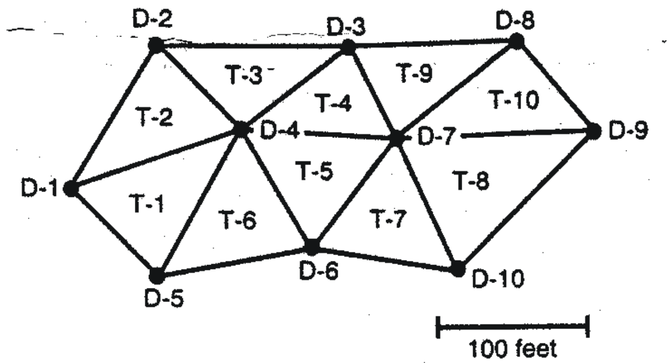


Fig.6. Diamond drill-hole plan of the Ojala copper deposit.

The method of calculation by triangles is a modification of the polygonal method in which the drill area is divided into triangles by connecting adjacent drill holes with construction lines. This method has the advantage that the areas are easily calculated by geometry or by coordinates by use of a computer or programmable calculator. The thickness above cutoff grade and average grade are calculated for each drill hole, as illustrated in the previous example. For the purposes of this problem, each drill hole is presumed to have the average grades and thicknesses shown in Table 6.

In like manner, the tonnage and grade for each triangle can be computed, and Table 7 can be constructed.

Table 6. Assay data for the Ojala copper deposit.

Drill Hole No.	Thickness, ft*	Average Grade, % Cu
D-1	50	0.93
D-2	75	0.77
D-3	60	0.82
D-4	100	1.05
D-5	75	0.72
D-6	60	0.49
D-7	105	1.63
D-8	80	0.91
D-9	70	0.86
D-10	75	0.74

Given these data, the tonnage and grade calculation for Triangle T-1 would be as follows.

Area = 4400 sq ft* (by geometry)

Average Grade-Thickness for Triangle T-1

Drill Hole	Thickness, ft	Average Grade, % Cu	Grade × Thickness, ft %
D-1	50	0.93	46.50
D-4	100	1.05	105.00
D-5	75	0.72	54.00
	225		205.50

$$\text{Average grade} = \frac{205.50}{225.00} = 0.91\%\ \text{Cu}$$

$$\text{Tonnage} = \text{area} \times \text{average thickness} \times \text{tonnage factor}$$

$$= \cancel{440} \times \frac{225}{3} \times \frac{1}{12.5} = 26{,}400\ \text{st*}$$

*Metric equivalents: ft × 0.3048 = m; sq ft × 0.09290304 = m ^{2}; st × 0.9071847 = t.

Table 7. Ore reserves for the Ojala copper deposit.

TABLE 1.9 Ore reserves for the Ojala copper deposit

Triangle	Drill Holes	Tons* Ore	Average Grade	Tons × Grade
T-1	D-1, D-4, D-5	26,400	0.91	24,024
T-2	D-1, D-2, D-4	26,400	0.94	24,816
T-3	D-2, D-3, D-4	22,500	0.91	20,475
T-4	D-3, D-4, D-7	22,260	1.23	27,380
T-5	D-4, D-6, D-7	18,550	1.15	21,332
T-6	D-4, D-5, D-6	27,260	0.79	21,535
T-7	D-6, D-7, D-10	26,240	1.07	28,076
T-8	D-7, D-9, D-10	40,500	1.15	46,575
T-9	D-3, D-7, D-8	24,418	1.20	29,301
T-10	D-7, D-8, D-9	28,917		34,411
		263,445		277,927

Tonnage = 263,445 st

$$\text{Average grade} = \frac{277{,}927}{263{,}445} = 1.05\%\ \text{Cu}$$

*Metric equivalent: st × 0.9071847 = t.

6.4. Calculation by Section

The basis of this method is to calculate a block of ore that is bounded by regularly spaced cross sections (see Tables 8 and 9) .The following equation illustrates the detailed calculation of a typical block of ore by the cross section method. The ore outline of each bounding section is divided into areas of influence based on the drill-hole or other sample data. The areas of influence are the neither planimetered or calculated geometrically. The individual areas are totaled for each section and the volume calculated by the average and area formula:

$$V = (A_1 + 2A_2 + 2A_3 + ...+ 2A_{n-l} + A_n)*L / 2$$

where A_n is area of section n and L is a constant section spacing, or when using only two adjacent sections:

$$V = (A_1 + A_2)*L / 2$$

The volume is then converted to tons by application of the appropriate tonnage factor.

Table 8. Assay data for Section 100N, Big Rat copper vein.

Sample No.	Area of Influence, sq ft*	Grade, % Cu	% Cu × sq ft*
T-1	A = 510	0.80	408
DDH-1	B = 1000	2.55	2550
DDH-2	C = 1040	1.66	1726
C-1N	D = 710	1.70	1207
	3260		5891

Total Area = 3260 sq ft*

Average grade $= \frac{5891}{3260} = 1.81\%$ Cu

*Metric equivalents: sq ft × 0.09290304 = m²; cu ft × 0.02831685 = m³.

Table 9. Assay data for Section 200N, Big Rat copper vein.

Sample No.	Area of Influence, sq ft*	Grade, % Cu	% Cu × sq ft*
T-2	A' = 848	0.92	780
DDH-4	B' = 1792	2.32	4157
DDH-3	C' = 1280	1.59	2035
C-2N	D' = 976	1.63	1591
	4896		8563

Total area = 4896 sq ft*

Average grade $= \frac{8563}{4896} = 1.75\%$ Cu

*Metric equivalents: sq ft × 0.09290304 = m²; cu ft × 0.02831685 = m³.

Figure 7 shows two cross sections spaced 30.48 m (100 ft) apart. These sections show a tabular dipping vein sampled by a surface trench, two drill holes per section, and one crosscut per section. The vein is assumed to be copper ore with a tonnage factor of 9.5 cu ft per ton.

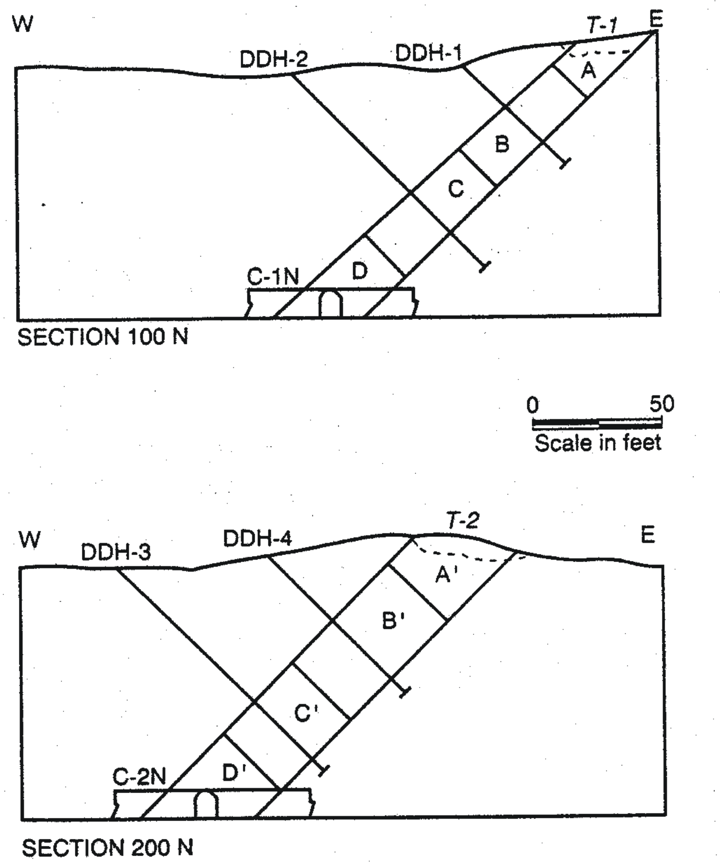


Fig.7. Cross sections of the Big Rat copper vein.

Block Average Grade:

Section	Area, sq ft	Average Grade, % Cu	Amount $\%Cu \times ft^2$
100N	3260	1.81	5901
200N	4896	1.75	8563
TOTAL	8156		14,463

Average block grade = 14463 / 8156 = 1.77% Cu

Volume of Ore Block:

v = [(area section 100N + area section 200N) x section spacing] / 2

v = [(3260 + 4896)x100] / 2 = 407 800 cu ft

Tonnage = 407800 / 9.5 = 42,926 st

The geologic reserve of this deposit between 100N and 200N is 38,934 t (42,926 st) with an average grade of 1.77% Cu. Similarly, the reserve calculations can be extended north and south to cover the entire minable strike length of the vein by adjacent pairs of sections.

6.5. Documentation

It is extremely important, no matter what ore reserve estimation methodology is used, to carefully document the method used and the limits applied. if the method is documented, anyone using the reserve figures will have a fuller understanding of the confidence limits inherent in the stated reserves. Also, in case of any change of mine personnel, it will be possible for future calculations to be consistent with previous reserve data. Adherence to this policy also allows for future critical review of the applicability of the methodology to the specific deposit. Should a problem be encountered with the application of the reserve estimation method to the deposit, it may be possible to correct the problem without completely recalculating the entire reserve inventory.

Writing up the reserve estimation methodology also forces the staff to fully identify potential problem areas and to define methods of handling such problems in a consistent manner. The documentation should be prepared in the form of a manual keyed to the specific property by examples. Adequate space should be provided for notes to be added by the staff as problems or questions arise or ideas for improving the reserve calculation method are encountered. Such an ore reserve preparation manual provides each new staff member with a set of uniform procedures and guidelines to be used on the specific property.

The cutoff limits used -those related to both mining method and grade- should be specified in the reserve statements. Examples of such qualifying statements include:

proven 3,795,000 tons averaging 2.15% Cu

probable 5,600,000 tons averaging 2.69% Cu

possible 2,901,000 tons averaging 2.80% Cu

based on a minimum mining width of 3.04 m (10 ft) of material above a cutoff grade of 0.80% Cu.

Note (1) all reserves in the "possible" category are located below the 3500 level, (2) all reserve figures are reported to the nearest 1000 st, and (3) additional inferred reserves exist at depth below current mining limits. These reserves presently are poorly defined and hence are not reported.

Problem : CALCULATION OF FEASIBILITY OF A COAL DEPOSIT

At final stage of pre-project studies done for a coal deposit, it is proposed to afford the following investments (The project is highly mechanized).

Amount of deposit	$R=10*10^6$ ton
Annual production rate	$T=1*10^6$ ton/year
Mine life planned	n= R/T = 10 years
Size of main investment (main preparations+shaft construction+ventilation units+others…)	$Y_a=150*10^6$ TL
Period to renew machinary and others	$n_y=5$ years
Investment to renew once each 5 years	$Y_y=300*10^3$ TL
Credit interest	i=0.10 q=1+i=1+0.10=1.1
Average annual sale income	$G_s=30$ TL/t*$1*10^6$ t/y = $30*10^6$ TL/y
Average annual expenses (labor+material+energy+eng.+others…)	İ=$7.5*10^6$ TL/year
Average annual gross profit	$G=G_s-M = G_s- (M_a + M_y + İ)$

According to dynamic investment analysis, determine whether the project is "profitable" or not. If not change the inputs to make it yes. (Scrap value of machinary at the end of project is not considered)

Firstly let's determine annual main and renewing investments.

Main investment, $Ma = Y_a * \dfrac{q^n(q-1)}{q^n-1} = 150x10^6\left[\dfrac{(1.1)^{10}*(1.1-1)}{(1.1^{10}-1)}\right] = 24.4*10^6 \quad TL/y$

Renew investment, $My = Yy * \dfrac{q^{ny}(q-1)}{q^{ny}-1} = 0.3x10^6\left[\dfrac{(1.1)^5*(1.1-1)}{(1.1^5-1)}\right] = 0.079*10^6 \quad TL/y$

Total annual expenses, M = $M_a + M_y$ + İ = $24.4*10^6 + 0.079*10^6 + 7.5*10^6$
M = $32*10^6$ TL/y

Average annual gross profit, G = G_s-M = $(30-32)*10^6 = -2*10^6$ TL/y

If G<0 then the project is not profitable. In this situation, it must either rejected or some investment parameter has to be changed.

If we charge expenses on unit coal produced;

Main investment = $(24.4*10^6) / (1*10^6)$ (interest+capital)	=	24.4 TL/t
Renew investment = $(0.792*10^6) / (1*10^6)$ (interest+capital)	=	0.079 TL/t
Annual expenses = $(7.5*10^6) / (1*10^6)$	=	7.5 TL/t
	+	
Production cost of unit coal	=	32.0 TL/t

It is also seen that the production cost is higher than the sale price about 2 TL for per ton of coal. It means that the entrepreneur will have a deficit in this project.

First consideration to make the project profitable is to increase the production capacity. If it is increased about 25% (by keeping the investment costs the same, but not average annual expenses). Let's determine the parameters again for this new condition.

İ=6.8*10^6 TL/year (assumption)

G_s=30*1.25*10^6 t/y = 37.5*10^6 TL/y

Mine life, n = R / T = (10*10^6 t) / (1.25*10^6 t/y) = 8 years

Total annual cost can be determined similarly;

Main investment, $Ma = 150x10^6 \left[\frac{(1.1)^8 * (1.1-1)}{(1.1^8 - 1)} \right] = 28.12 * 10^6 \quad TL / y$

Renew investment, M_y = 0.079*10^6 TL/y

Total annual expenses, M = M_a + M_y + İ = 28.12*10^6 + 0.079*10^6 + 6.8*10^6
M = 35*10^6 TL/y

Average annual gross profit, G = G_s-M = (37.5-35)*10^6 = 2.5*10^6 TL/y

G>0 then the project is profitable.

Production cost of unit coal = M/T = (35*10^6 TL/y) / (1.25*10^6 t/y)
= 28 TL/t

which is lower than the sale price. Therefore the project is profitable.

It can be concluded that production rate can be increased as much as possible which affects the economic situation of projects in positive manner.

Problem : CALCULATION OF ECONOMIC STRIPPING RATIO

The results of feasibility study belong to a mine are given for both open-pit and underground mining productions. According to these, determine economic stripping ratio.

	Open-pit	Underground
Production rate (t/year)	2 000 000	1 000 000
Investment Cost (TL)	3 000 000 000	5 000 000 000
Number of labor (wage/day)	150	1 500
Labor Cost (TL/wage)	3 000	3 500
Energy Consumption (kWh/t)	15	35
Energy Cost (TL/kWh)	2	2
Other Expenses (TL/t)	100	200
Mine Life, n (year)	5	5
Annual Interest Rate, i (%)	20	20
Stripping Cost (TL/m^3)	300	

To find open-pit mining cost, M_a ;

Investment Cost $Y_m = \frac{0.2(1+0.2)^5}{(1+0.2)^5-1} * 3x10^9$ = 1 003 139 110 TL/year

Labor Cost = 150 w/d*3000 TL/w*365 d/y = 164 250 000 TL/year
Energy Cost = 15 kWh/t*2000000 t/y*2 TL/kWh = 60 000 000 TL/year
Other Cost = 100 TL/t*2000000 t/y = 200 000 000 TL/year

TOTAL 1 427 389 110 TL/year

Then M_a = 1427389110/2x10^6 = 714 TL/t

To find underground mining cost, M_y ;

Investment Cost $Y_m = \frac{0.2(1+0.2)^5}{(1+0.2)^5-1} * 5x10^9$ = 1 671 898 516 TL/year

Labor Cost = 1500 w/d*3500 TL/w*365 d/y = 1 916 250 000 TL/year
Energy Cost = 35 kWh/t*1000000 t/y*2 TL/kWh = 70 000 000 TL/year
Other Cost = 200 TL/t*1000000 t/y = 200 000 000 TL/year

TOTAL 3 858 148 516 TL/year

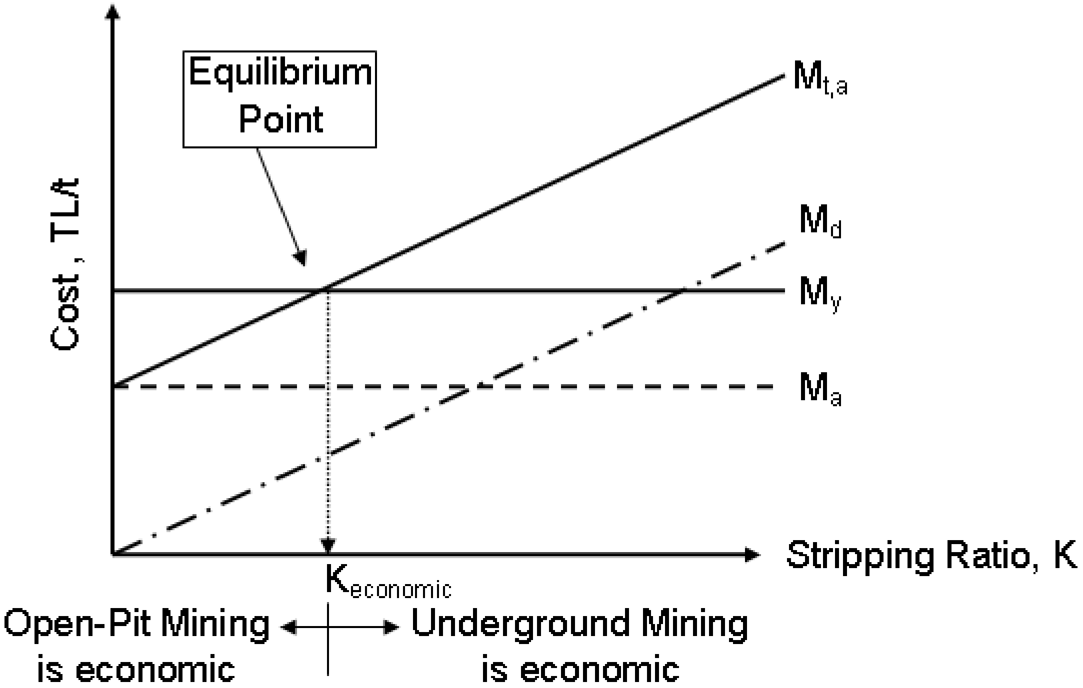
Then M_y = 3858148516/1x10^6 = 3858 TL/t

Economic stripping ratio regarding the unit costs can be defined as;

$K_{economic} = (M_y - M_a) / M_d$

In this equation,

$K_{economic}$: Economic stripping ratio, m^3/t
M_y : Underground mining cost, TL/t
M_a : Open-pit mining cost, TL/t
M_d : Stripping cost, TL/m^3



From figure above, $M_y = M_{t,a} = M_a + K.M_d$

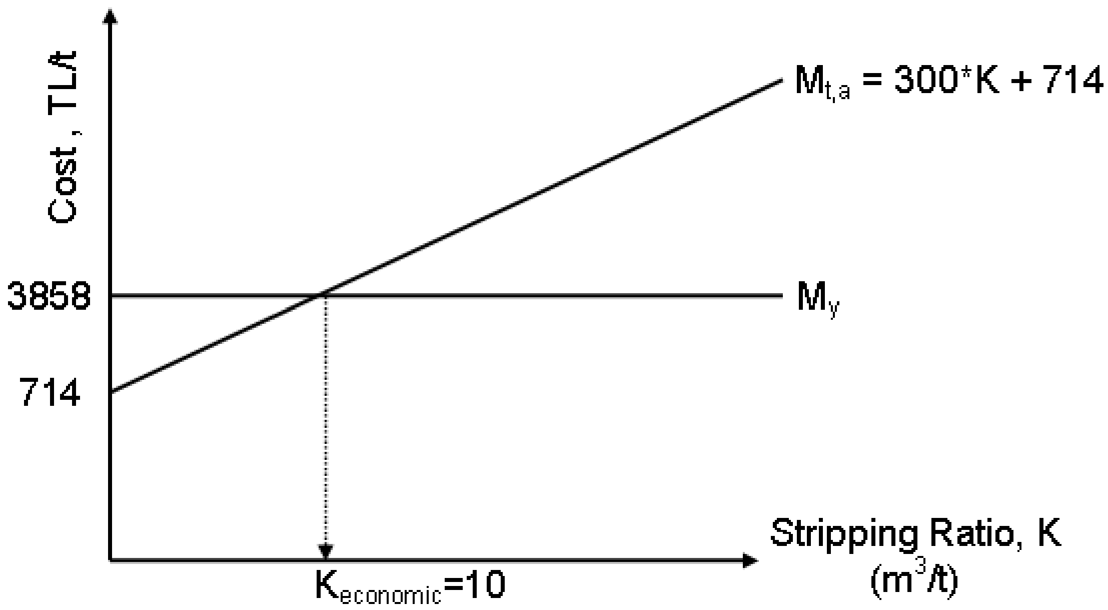
$M_{t,a}$: Total Open-pit mining cost, TL/t

If the equation is solved for stripping ratio at the equilibrium point, we get;

$K_{economic} = (M_y – M_a) / M_d$

According to the costs determined for our mine

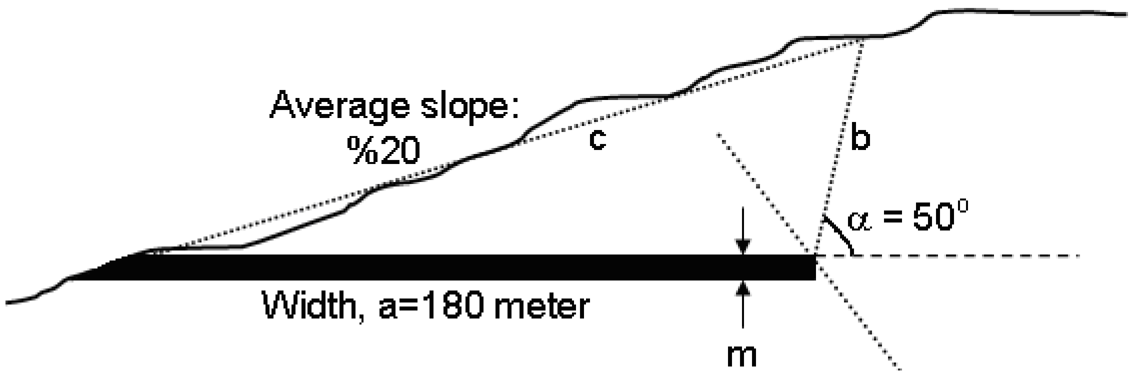
$K_{economic} = (3858 – 714) / 300 \cong$ **10 m³/t**



Problem : CALCULATION OF SEAM THICKNESS

Vertical section of a horizontal coal seam and some related parameters are given below. For givens, determine first the critical (economical) stripping ratio on the basis of costs and then seam thickness (m) goes with the ratio. By using a curve, show how the stripping ratio changes with the seam thickness.

Stripping cost	: 5 TL/m^3
Open-pit mining cost	: 8 TL/t
Underground mining cost	: 30 TL/t
General pit slope	: 50^o
Unit weight of coal	: 1.5 t/m^3
Length of seam in 3rd direction	: 500 m



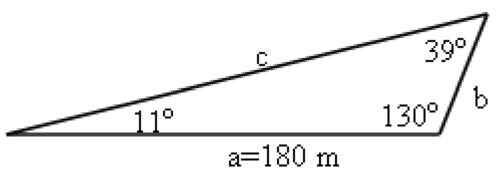
$K_{economic} = (M_y – M_a) / M_d = (30-8)/5 = 22/5 = 4.4\ m^3/t$ (overburden/coal)

In this equation,

$K_{economic}$: Economical (critical) stripping ratio, m^3/t
M_y : Underground mining cost, TL/t
M_a : Open-pit mining cost, TL/t
M_d : Stripping cost, TL/m^3

If the slope is 20% (=20/100), then β = arctan 0.2 = 11^o

$\alpha = 50^o \Rightarrow \alpha' = (180-50) = 130^o$ $\lambda = 180^o - (11^o + 130^o) = 39^o$



On the given triangle

$\frac{180m}{Sin39^o} = \frac{b}{Sin11^o} \Rightarrow$ b = 54.6 meter

$\frac{180m}{Sin39^o} = \frac{c}{Sin130^o} \Rightarrow$ c = 219.1 meter

If the lengths of all sides of a triangle is known, then to find its area;

p = (a+b+c)/2 = (180+54.6+219.1)/2 = 226.85 m.

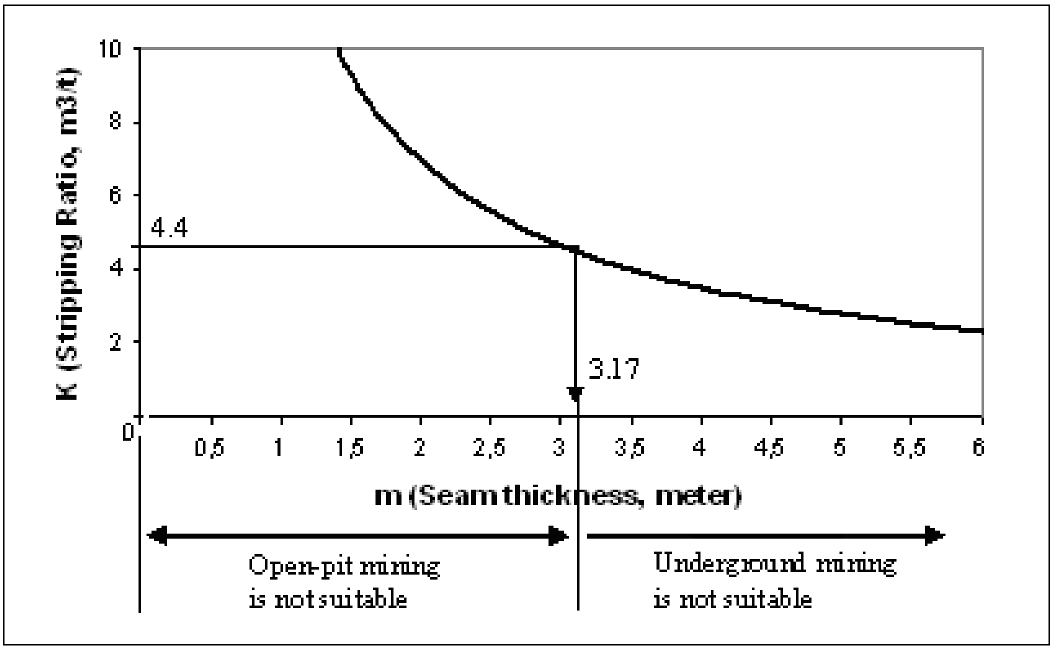
Area, F = $\sqrt{226.85(226.85-180)(226.85-54.6)(226.85-219.1)} = \sqrt{14187612.3} = 3766.65\ m^2$

Stripping volume, $V_d = 3766.65*500 = 1883322\ m^3$

Coal amount, $T_k = m*a*L*\gamma = m*180*500*1.5 = 135000*m$ ….. t/m of thickness

$K = V_d / T_k = 1883322/(135000*m) = 4.4$

Seam thickness, m = 1883322/(135000*4.4) = 3.17 meter


K (Stripping Ratio, m3/t)
10
8
6
4
2
0
4.4
3.17
0,5
1
1,5
2
2,5
3
3,5
4
4,5
5
5,5
6
m (Seam thickness, meter)
Open-pit mining
is not suitable
Underground mining
is not suitable

DRILLING (ROCK PENETRATION)

1. GENERAL

Drilling is performed for one of the following reasons;

- to construct space in the rock,
- to exploit the material being excavated, or
- to use the drilled holes for special purposes other than blasting.
    - water well drilling
    - prospect drilling
    - oil drilling
    - anchor hole drilling
    - pipe and cable drilling
    - pumping hole drilling

2. CLASSIFICATION OF METHODS

On the basis of form of rock attack, or mode of energy application. Since drilling occupies only one category in the classification, the more general term *rock penetration* is employed in referring to all methods.

Method	System or Machine
Mechanical (drilling)	
Percussion	churn, cable tool, rock drill, channeler
Rotary, drag	auger, wire rope, chain, rotary saw
Rotary, roller	rolling cutter
Rotary-percussion	drag, roller
Thermal (flame)	jet piercer, channeler
Fluid	hydraulic monitor, pallet impact
Sonic (vibration)	high frequency transducer
Chemical	shaped charge, capsule, softener
Electrical (arc)	
Light (laser)	
Nuclear (fission, fusion)	

2.1. Mechanical Attack : The application of mechanical energy to rock can be performed in only two ways: by percussive or rotary action. Combination of the two methods is termed as rotary- percussion.

2.2. Thermal Attack : Flame attack with the jet piercer or channeler. Used not only to produce blastholes but to chamber them as well and to cut dimension stone.

2.3. Fluid Attack : To produce a directed hole with a fluid from an external source, jet action or erosion appears to be most feasible, but application is limited.

2.4. Sonic Attack : Attractive but not presently commercial.

2.5. Chemical Attack : Chemical reaction may be more attractive as an accessory rather than a primary means of penetration.

2.6. Other Methods of Attack: The remaining methods must be classified as in the hypothetical or research category at present.

3. THEORY OF PENETRATION

Since the vast majority of rock penetration is carried out by mechanical attack systems, the following parts are devoted entirely to the fundamentals of penetration by drilling.

3.1. Operating Components of System

The three main functional components of the drill system are:

1. *Drill (source)*: is the prime mover, converting energy from its original form into mechanical energy to actuate the system.
2. *Rod (transmitter)*: transmits energy from the prime mover or source to the bit or applicator.
3. *Bit (applicator)*: is the applier of the energy in the system, attacking rock mechanically to achieve penetration.

In commercial drilling machines, much attention of late has been focused on reduction of energy losses in transmission.

3.2. Phases of Rock Drilling

A drilling (or any penetration) system must perform two separate operations in order to achieve advance into rock: 1) fracture of material in the solid and 2) ejection of the debris formed. The first phase is actual penetration, while the second is cuttings removal.

3.3. Mechanics of Penetration

There are only two basic ways to attack rock mechanically – *percussion and rotation*. Known commercial drilling methods utilize these principles or combinations of them.

Causing rock to break during drilling is a matter of applying sufficient stress with a tool to exceed the strength of rock. This resistance to penetration to rock is termed its drilling strength.

The significant parameters of the two basic methods – i.e., percussion and rotary types – are the geometry of cutting tools, the applied force and the responses of rock.

4. FACTORS INFLUENCING DRILLING

A number of factors affect rock penetration or cuttings ejection in drilling process. These in turn largely determine the performance of a given drilling machine.

The various factors may be grouped in six categories: 1) drill, 2) rod, 3) bit, 4) circulation fluid, 5) hole dimensions, and 6) rock.

Those factors in categories 1-4, components of the drilling system itself, are referred to as design or operating variables. They are dependent (controllable) within limits being selected to match the environmental conditions reflected by category 6.

The hole geometry factors of category 5, drillhole size and depth, are dictated primarily by outside requirements and are independent (uncontrollable) variables.

The environmental factors of category 6 include:

a) Rock properties (resistance to penetration, porosity, moisture content, density, etc.)

b) Geological conditions (petrological and structural – bedding, folds, faults, joints, etc.)

c) State of stress (overburden pressure and formation fluid pressure; unimportant in shallow holes).

These are often called the *drillability factors*.

Another group of factors is external to the drilling process itself and may be referred as job or service factors (i.e., labor, job site, scale of operations, weather, and supervision). These may exert a considerable influence on drill performance.

4.1. Rock Characteristics which Affect Drilling

Some of the important engineering properties of rock material that have an overall effect on the drilling techniques:

Hardness: A rock’s hardness indicates how much stress is necessary to cause failure within the rock. Following table presents the degree of hardness as a function of uniaxial compressive strength.

HARDNESS	Mohs	σ_c (MPa)
Extremely hard	>7	>200
Hard	6 – 7	120 – 200
Medium hard	4.5 – 6	60 – 120
Quite soft	3 – 4.5	30 – 60
Soft	2 – 3	10 – 30
Extremely soft	1 – 2	<10

Abrasiveness: is a time-dependent parameter for drill bit wear, and it depends on the mineral composition of the rock, which the drill bit wear is proportional to.

Texture: Texture refers to the grain structure of the rock and can be classified by such properties as porosity, looseness, density and grain size. All these have a definite relationship to the drilling speed.

Structure: Structural properties, such as faults, joints, bedding planes, schistosity and rock type contacts, dip and strike all affect hole linearity deviation, drill bit penetration and structural strength of rock material.

Breaking Characteristics: Breaking characteristics describe the rock behaviour when struck with the hammer; each rock type has a typical manner and degree breakage related to its texture, mineral composition and structure.

5. PERCUSSION DRILLING

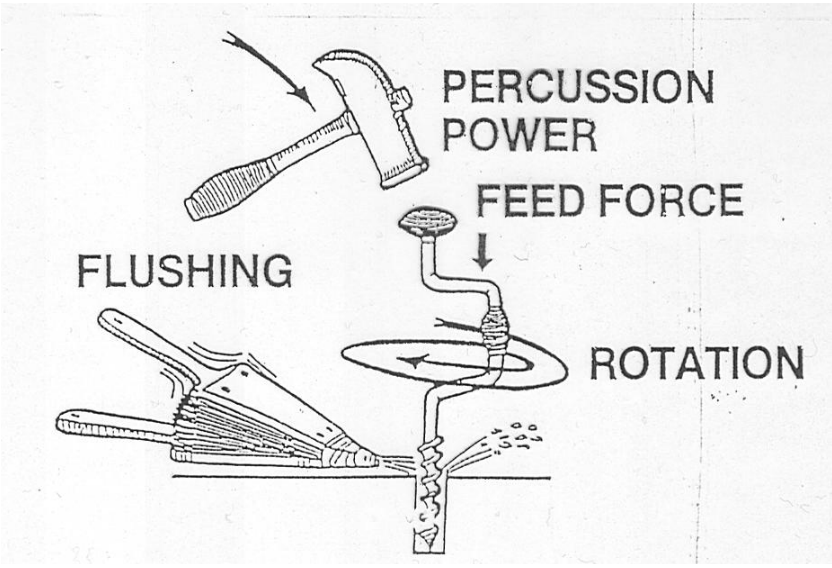


Figure – The principle of percussion drilling.

Percussion drilling combines percussion, feed, rotation and flushing.

Percussion: Impact energy and impact frequency – varying between 2000-3500 blows per minute – together create the percussion output power. The impact energy is developed by a piston that strikes the bit or drill steel and produces a high-energy stress wave that is transmitted to the bit and crushes and chips the rock when the blows are of sufficient magnitude. The output power is directly proportional to the pressure (fluid or air) in the system. Generally the higher output power gives the higher penetration rate.

Feed: In order to obtain sufficient drilling, the impact energy transmitted into the rock has to be maximized. This implies that the bit must constantly be in contact with the bottom of the drillhole. It is performed either by chain, by screw or by cylinder. The feed force varies mainly according to properties of the rock drilled. Increased feed force will give higher penetration up to a certain level.

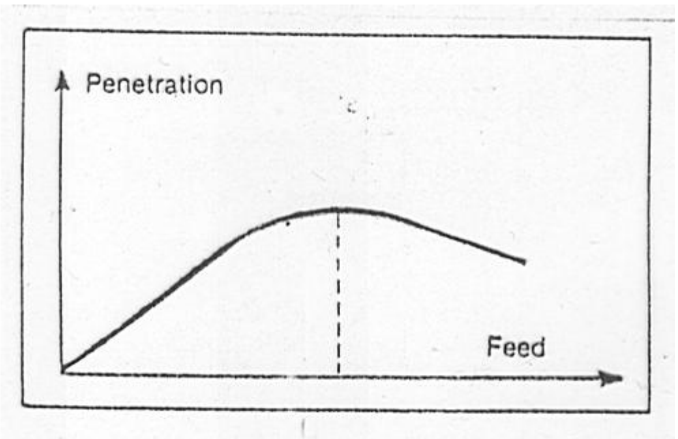


Figure – Feed force.

Rotation: The main function of rotation is to turn the drill bit to a new position for a new energy blow; the optimum speed varies according to rock nature and other drilling parameters. Too low a rotation speed results in low penetration whereas with too high rotation speed energy will be lost and drill steel will wear.
The rotation between consecutive blows must be adapted to produce as many cuttings as possible. The rotation speed is normally 80-250 revolutions per minute.

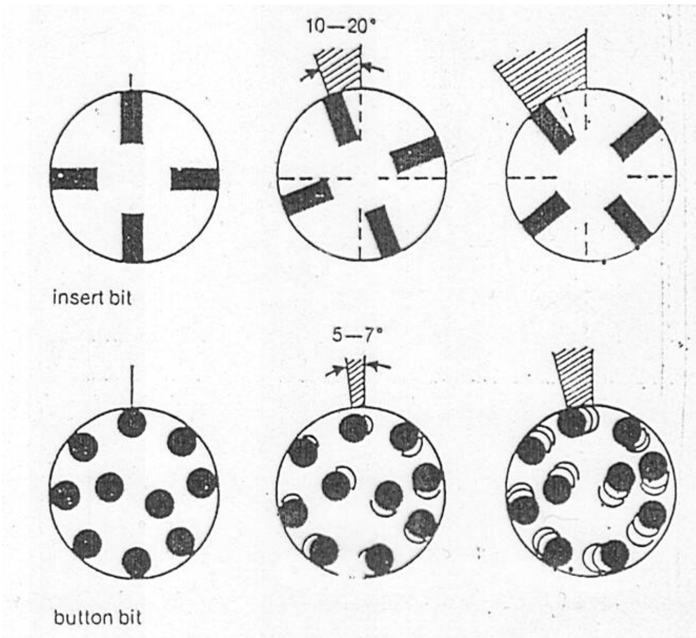


Figure – Rotation speed and bit turn between the consecutive blows.

Flushing: Flushing (by air or water) is needed to remove the cuttings from the drillhole. If it is not sufficient, regrinding takes place in the hole and thus causes smaller penetration and excessive drill steel wear. With air flushing the penetration rate is 10-30% higher than with water flushing.

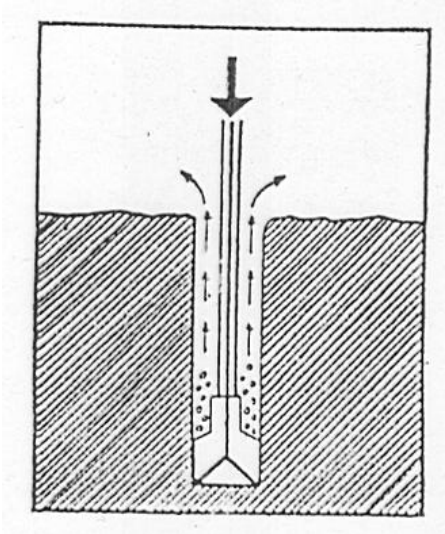


Figure – Principle of flushing.

5.1. Mechanism of Rock Breakage in Percussion Drilling

Rock breakage by drilling is based on the formation of a heterogeneous stress field under the concentrated loads produced by the bits. The sequence of crater formation under a single blow is described as: (1) rock is elastically deformed, (2) main subsurface cracks form, radiating downward, (3) secondary cracks propagate, and (4) broken particles are ejected, to form a crater.

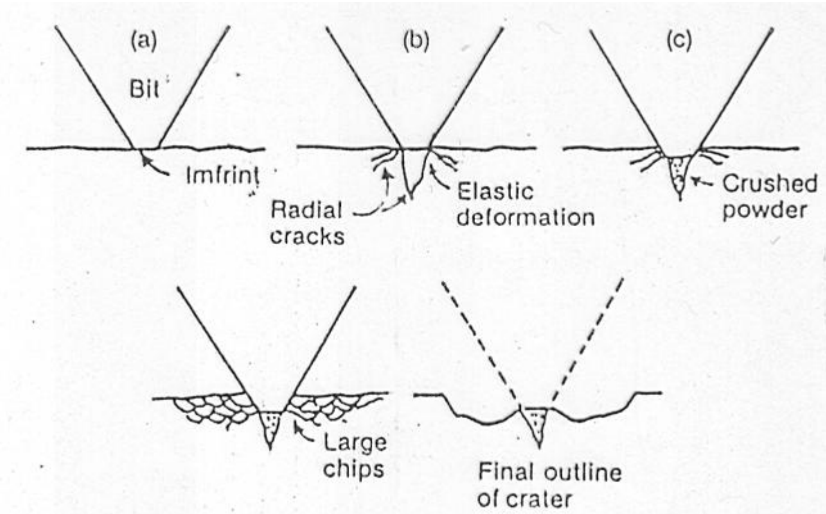


Figure – Mechanism of rock breakage. Fractured and crushed zone.

5.2. Percussive Drilling Bits

They are of two major types: chisel and button. They are subdivided into different designs to better adapt to all kinds of percussive drilling.

1. Cross bits : four sintered tungsten-carbide inserts set at right angles.
2. Cross bits : four sintered tungsten-carbide inserts with a special rake angle for rotary-percussive drill.
3. X-bits
4. Chisel bits
5. Button bits

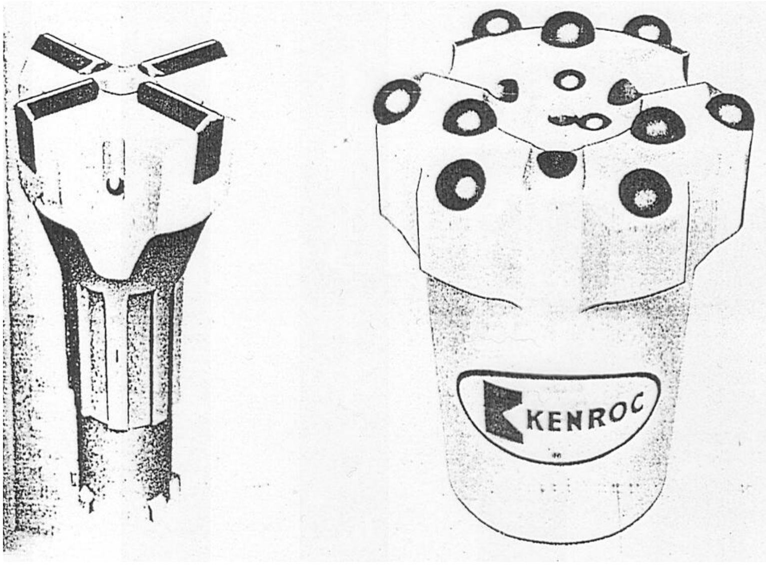


Figure – X-bit and button bit.

Percussive drills are most commonly used to drill small-diameter holes in hard rock and larger diameter holes where heavy rotary rigs cannot be used or are not available. For holes 2 to 12 inches in diameter to depths of 100 ft.

The primary advantages of percussive drilling are: (1) small-diameter holes can be drilled in the hardest rock, (2) percussive drill rigs are lighter, less expensive and more manoeuvrable, (3) can be operated by one man and can drill at any angle, (4) it is economical in a wide range of rock types and hole diameters.

6. ROTARY DRILLING

Rotary drills attack rock with energy supplied to the bit from the rotating action and thrust; the cemented carbide roller buttons are pressed into the rock and rotated.

Rotating action: The bit is rotated to continuously work on a new part of the hole. Rotary speeds of 50 to 90 rounds per minute are normally used. Sufficient rotary power must be supplied to the bit for all drilling conditions. The most common power sources for rotation of the bit on blasthole drills are electrical (either AC or DC), hydraulic and pneumatic. This power is converted to mechanical power in the drill stem.

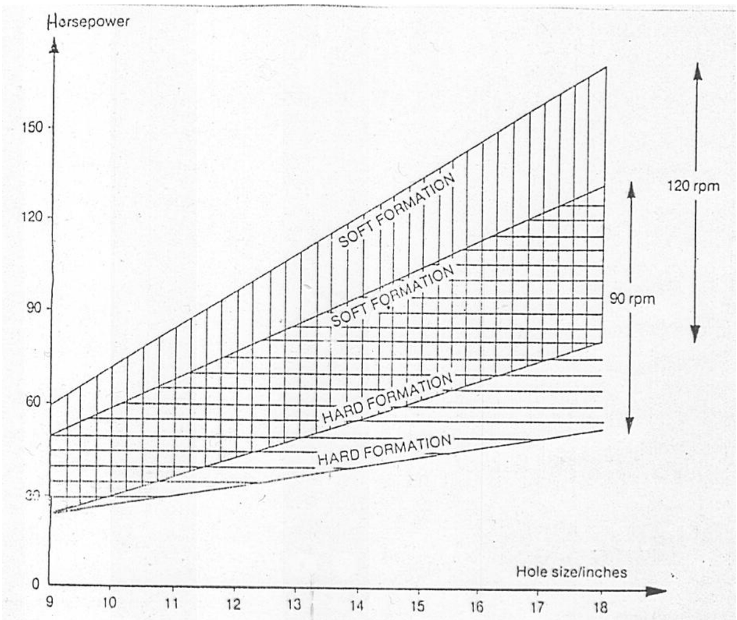


Figure – Required rotary power versus hole size.

Thrust: The pull down force is the thrust that the drilling applies to the bit; this pushes the roller bit inserts into the rock. The pull down force can vary from 0.5 ton per inch of bit diameter in soft material to 4 ton per inch of the bit diameter in hard rock. Down feed speed range varies from 1 cm per minute in hard rock to 3 meters per minute in soft material.

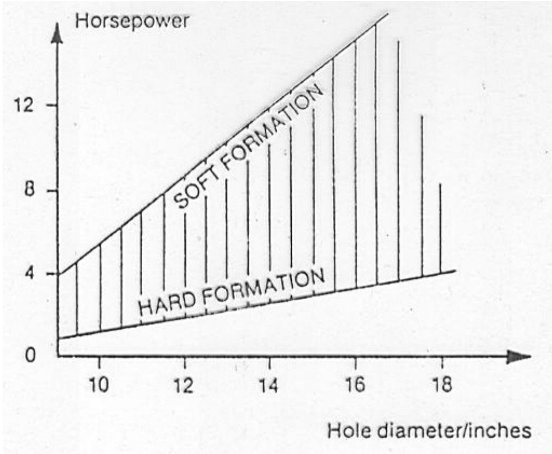


Figure – Required pull down horsepower versus hole size.

Flushing: Drill cuttings in blasthole are generally removed by compressed air. 10 % of it is routed through the bit bearings for cooling.

6.1. Mechanism of Rock Breakage in Rotary Drilling

In rotary drilling, the bit moves forward by the effect of torque and thrust simultaneously applied to the rock surface. The mechanics of bit penetration are dependent upon the basic designs of the bit and the configurations of the cutting components.

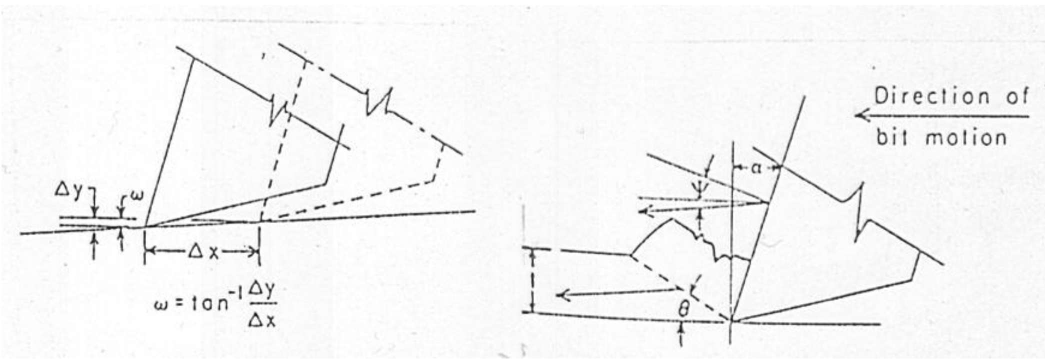


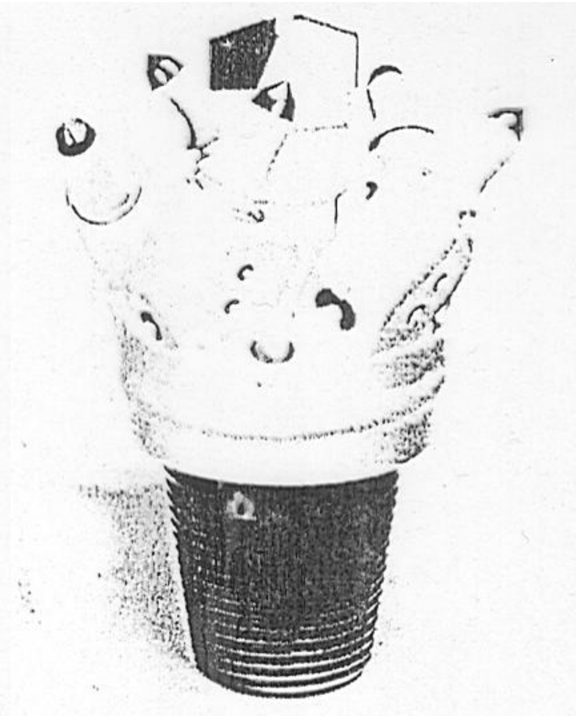
Figure – Geometry of drag bit and cutting force.

6.2. Rotary Drill Bits

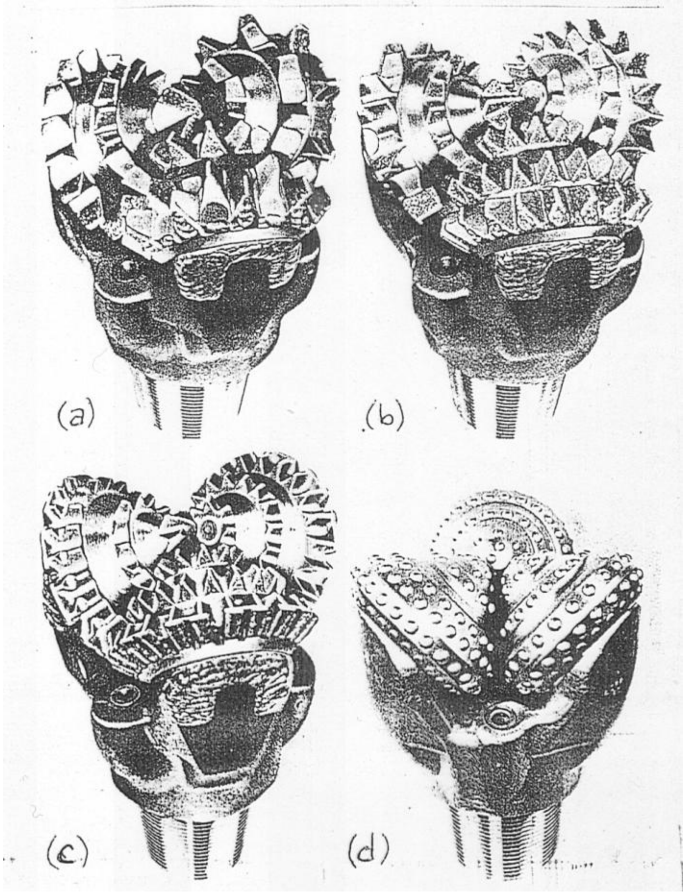
The two main types of bits used in rotary drilling are drag and rolling cutter. Drag bits depend on a ploughing or scraping action to fragment material and are confined to drilling soft material.

Drag bits: They are two general types: integral blade and replaceable cutter element. They are used in soils, soft shales, poorly cemented sandstones and other soft sediments.

Roller cutting bits: The rolling cutter bit is of either two- or three-cone type. They are classed as soft, medium, hard and very hard formations. Hard formation bits generally have smaller teeth.



Replaceable finger-type rotary drag bit.



Tricone bits: (a) soft formation, (b) medium formation (c) hard formation, (d) very hard formation.

7. THERMAL DRILLING OF BLASTHOLES

Drilling with high-energy heating is a proven and new method of preparing blastholes. In some types of hard rock, the stress induced by heating results in localized flaking of the heated zone, a process known as thermal spalling. Rocks containing minerals with high thermal expansion properties generally spall well under the effects of surface heating. The following are good examples of good spalling rock types: taconite, quartzite, dolomite, granite, sandstone, rhyolite and diabase.

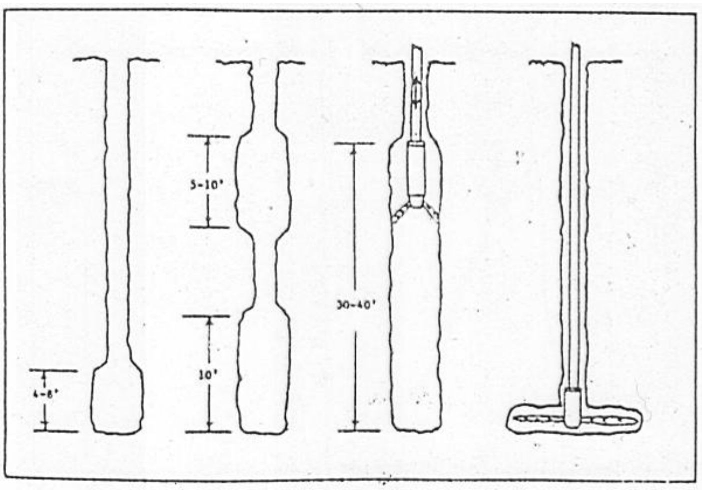


Figure – Examples of blasthole chambering configurations.

Enlarging and chambering blastholes improve fragmentation efficiency. Increasing the blasthole volume to hold a larger explosive charge reduces the number of holes required.

8. DRILLING SELECTION

The following table presents the suitable drilling machine for different groups of rocks. They are sub-grouped according to their hardness and abrasiveness definitions.

Table – Drilling suited for different rocks. a) igneous, b) sedimentary, and c) metamorphic.

a)

	Hardness and abrasiveness			
	Abrasive	Intermediate	Less abrasive	Decomposed
ROCKS	Rhyolite Aplite Felsite Granophyre Grano diorite Pegmatite Quartz porphyry Granite	Olivine basalt Dacite Danite Olivine gabbro Quartz diorite	Andesite Basalt Trahyte Dolerite Diorite Gabbro Syenite	Serpentine "Red" bvasalt Kaolinized granite
DRILLING	Heavy sinkers (small ø holes) Heavy drifters ø 50—230 mm DTH machines ø 102—150 mm Heavy rotary drills ø 150 mm		Heavy or medium sinkers and drifters or DTH machines Heavy rotary drills ø 150 mm	Ripping Rotary drilling. Percussive drilling necessary if fresh rock occurs.

b)

	Hardness and abrasiveness		
	Hard + Abrasive	Intermediate	Softer
ROCKS	Granulite Quartz schist Quartzite Gneiss	Hornblendeschist Mica schist Dolomite Marble	Slate Phyllite Chlorite schist Marble
DRILLING	Heavy percussive drills Very heavy rotary drills	Medium to heavy percussive drills. Medium to heavy rotary drills in softer rocks.	

c)

	Hardness and abrasivenes				
	Abrasive Hard	Abrasive Less hard	Abrasive Friable	Non-abrasive Hard	Non-abrasive Soft
ROCKS	Flint Chert Sed. Quartzite Greywacke Quartz conglomerate	Siltstone Volcanic ash Siliceous limestones Tuff Gritstone Agglomerate	Friable sandstone Calcareous sandstone Some grits	Limestone Mudstone Freestones	Marl Mudstone Shale Chalk Coal Oolite
DRILLING	Heavy percussive drills Very heavy rotary dr.	Medium heavy percussive drills or heavy rotary drills rotary rigs	Light to medium heavy rotary drills	Medium to heavy percussive drills, DTH drills, or heavy rotary drills	Drag-bit, Rotary-drills

ENGINEERING BLASTING OPERATION

1. INTRODUCTION

Rock is blasted either to break it into smaller pieces such as in most mining and quarrying operations or large blocks for dimensional stone mining and some civil engineering applications. Precise engineering of blasting operations are needed to achieve the desired objectives. The engineering of blasting operations needs clearly defined objectives, materials, skilled techniques, the necessary theoretical background of the process of rock fragmentation and effect of rock conditions and experience in combining them.

2. BLASTING OBJECTIVES

In mining and quarrying, the main objective is to extract the largest possible quantity at minimum cost. The material may include ore, coal, aggregates for construction and also the waste rock required to remove the above useful material. The blasting operations must be carried out to provide quantity and quality requirements of production in such a way that overall profits of mining or quarrying operation are maximized. Large blocks needing secondary breakage or an excess of fines, can result from poorly designed blasts due to adverse geological conditions. A well designed blast should produce shapes and sizes that can be accommodated by the available loading and hauling equipment and crushing plant with little or no need for secondary breakage. While optimizing the fragmentation, it is also important, for safety and ease of loading, to control the throw and scatter of fragments. However, other times controlled displacement is provided, as in the case of casting of overburden or explosive mining, where part of the overburden is thrown to such a distance that it need not be handled again.

In civil engineering, rock is removed to create tunnels or caverns, or deep excavations at the ground surface for road cuts, foundations, or basements. The emphasis is not on high rates of production, although the job must be done as quickly and as cheaply as possible, but on creating space and leaving behind stable rock walls that are either self-supporting or require little reinforcement and lining. Requirements for smooth walls and long-term stability exist also in mine shafts, crusher stations and in mine development drifts that must remain open for moderate periods. Special blasting techniques are applied in carrying out controlled blasting to produce smooth walled excavations.

In some civil engineering operations large blocks are needed for many operations such as dam construction, and break water construction. To meet these requirements techniques and explosive materials used are modified as practiced for conventional blasting.

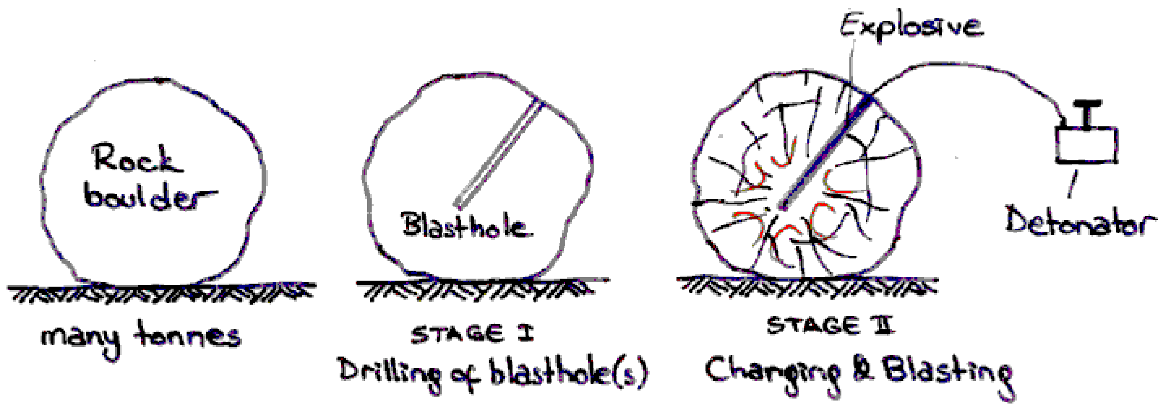
The size of blasting operations varies greatly from those needing a small charge to those requiring several hundreds of kilograms of explosives and also from those involving a few cubic meter rock to millions of cubic meter of rock. For surface blasting operations such as quarries, foundations, trenches, the environment needs to be protected. Blast damage due to vibrations and noise needs to be avoided by careful blasting. Controlled blasting is often used to limit the projection of flyrock. For underground blasting operations care needs to be taken so that fumes and dust produced is minimal to protect persons working underground. Care needs to be taken of extraneous hazards while handling and during storing and also while carrying out blasting operations to prevent accidental explosions.

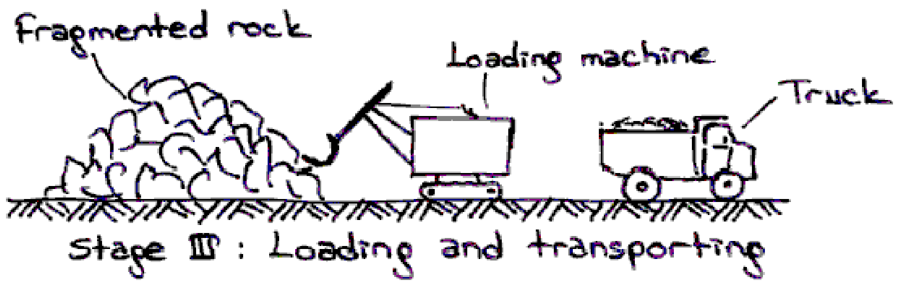
Mining and quarrying operations have production as a primary goal, but precautions must be also taken to avoid damaging the rock left behind. In terms of mining economics, the optimum excavating method is one that maximizes production and safety and minimizes dilution, excavation costs and environmental costs.

3. THE PROCESS OF FRAGMENTATION

The processes involved in rock fragmentation by blasting have been the subject of large amount of experimentation and studies. The difficulty is that at the time of fracture, fragmentation and displacement of rock, persons or sensors may not be close to the point of action. Nevertheless, without basic theoretical considerations and understanding of the phenomena involved precise blasting results are difficult to achieve.

When an explosive detonates in hole the pressures can exceed 10 GPa (over 100 000 kg/cm^2, 1 GPa=10197 kg/cm^2), sufficient to shatter the rock near the hole, and also generate a stress wave that travels outward at a velocity 3000-5000 m/s. The leading front of the stress wave is compressive, but it is closely followed by the tensile stresses that are mainly responsible for rock fragmentation. A compressive wave reflects when it reaches a nearby exposed rock surface, and on reflection, becomes a tensile strain pulse. Rock breaks much more easily in tension than in compression, and fractures progress backward from the free surface. The gas pressure generated during the process also act to widen and extend stress-generated cracks or natural joints. The fragmentation process which takes place is the combined effect of the above two, the role of each is dependent on the rock conditions, blasting geometry, explosive materials and initiation systems. The relative role of the stress waves and gas pressure is not fully understood and hence an accepted process of rock fragmentation by blasting is lacking.





4. MECHANISMS OF ROCK BREAKAGE

During the detonation of an explosive charge inside rock, the conditions presented are characterized by two phases of action:

- ***1st phase :*** A strong impact is produced by the shock wave linked to the Strain Energy, during a short period of time.
- ***2nd phase :*** The gases produced behind the detonation front come into action, at high temperature and pressure, carrying the Thermodynamics or Bubble Energy.

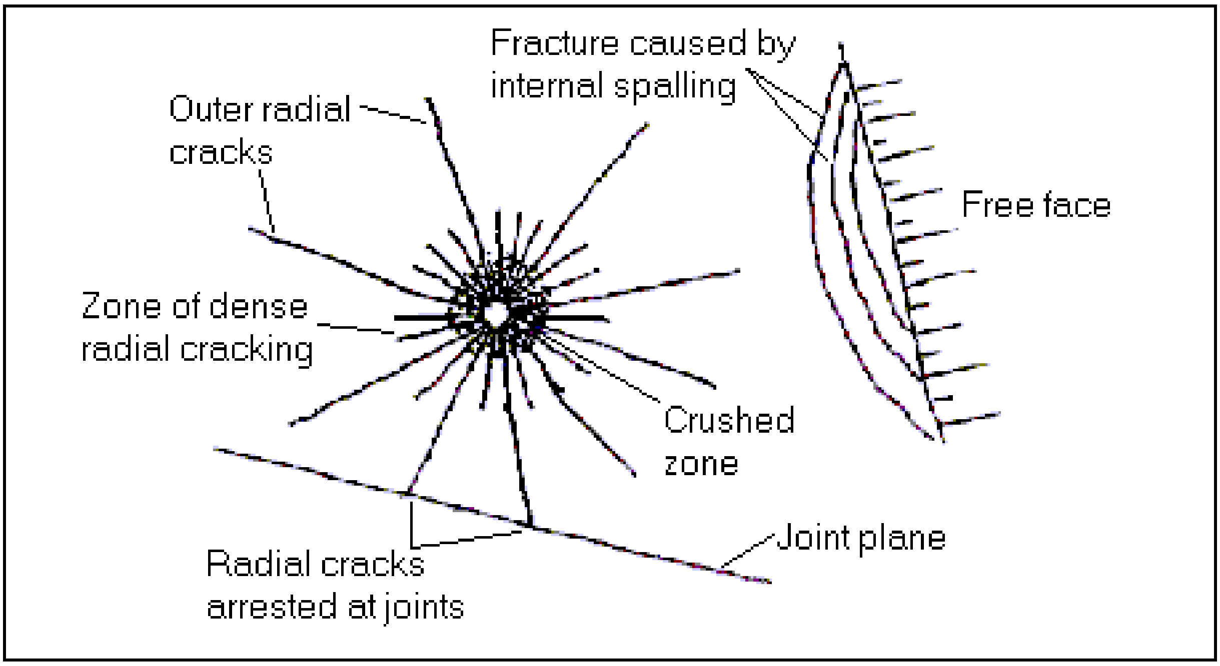
To get an idea of the strength of an explosive :
Consider a thermal plant with 550MW of power installed.

1kW=0.238 kCal/s ➔ 550MW=131*10^3 kCal/s

1 kg of gelatine explosive is 1200 kCal. If it is placed in a column of 1 m in length and detonated with a detonation velocity of 4000 m/s, releases a strength of:

1200*4000/1=4800*10^3 kCal/s, which is 37 times higher.

In the fragmentation of rocks with explosives at least eight breakage mechanisms are involved, with more or less responsibility, but they all exert influence upon the results of the blasting.



4.1. Crushing of rock

In the first instants of detonation, the pressure in front of the strain wave, which expands in cylindrical form, reaches values that well exceed the dynamic compressive strength of the rock, provoking the destruction of its inter crystalline and inter granular structure. The thickness of the so called crushed zone increases with detonation pressure of the explosive and with the coupling between the charge and the blasthole wall. The size of crushed zone is about 2 and 4 times of the hole diameter.

According to Hagan, this breakage mechanism consumes almost 30% of the energy transported by the strain wave, only contributing a very small volume to the actual rock fragmentation, around 0.1% of the total volume corresponding to the normal breakage per blasthole.

4.2. Radial fracturing

During propagation of the strain wave, the rock surrounding the blasthole is subjected to an intense radial compression which induces tensile components in the tangential planes of the wave front. When the tangential strains exceed the dynamic tensile strength of the rock, the formation of a dense area of radial cracks around the crushed zone that surrounds the blasthole is initiated.

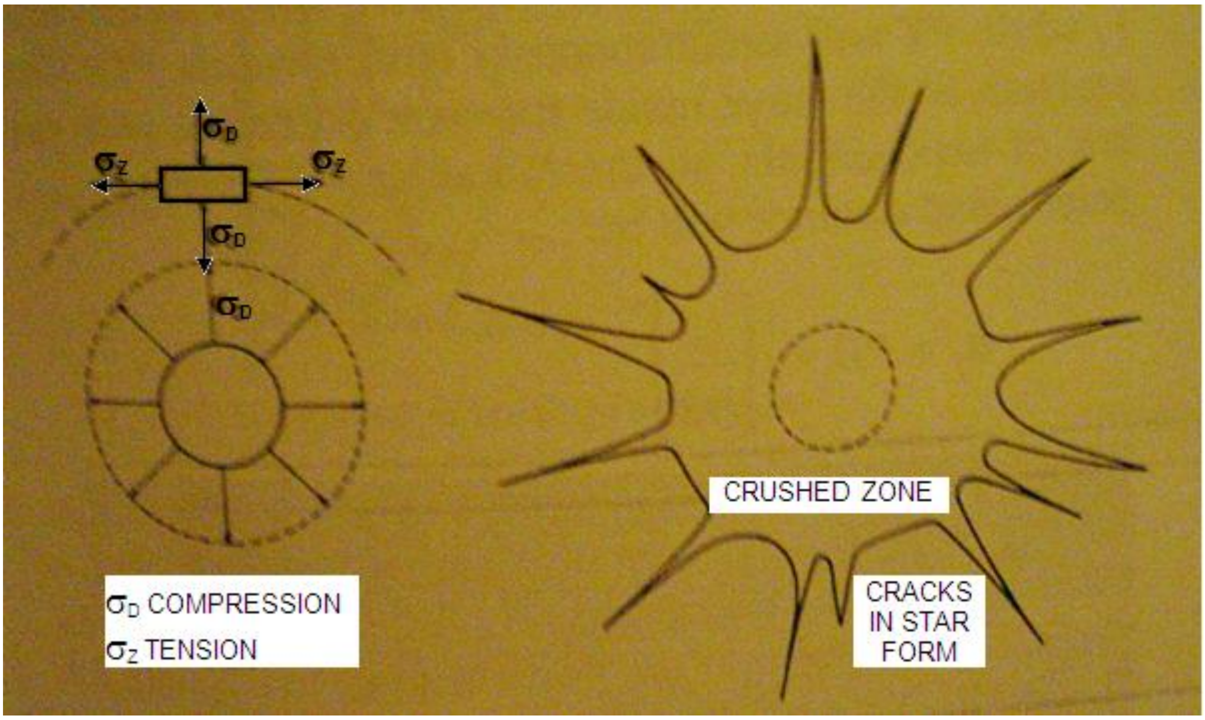


Figure. Radial fracturing.

The number and length of these radial cracks increase with:

1. The intensity of strain wave on the blasthole wall or on the exterior limit of the crushed zone, and
2. The decrease in dynamic tensile strength of the rock.

Beyond this inner zone of intense fracturing, some of the cracks extend noticeably and are systematically distributed around the blasthole. The propagation velocity of the cracks is from 0.15 to 0.40 times that of the strain wave, although the first micro cracks are developed in a very short time, around 2 ms.

When the rock has natural fractures, the extension of the cracks is closely related to these. If the explosive columns are intersected lengthwise by a pre-existing crack, these will open with the effect of the strain wave and the development of radial cracks in other directions will be limited. The natural fractures that are parallel to the blastholes, but at some distance from them, will interrupt the propagation of the radial cracks.

4.3. Reflection breakage or spalling

When the strain wave reaches a free surface two waves are generated, a tensile wave and a shear wave. This occurs when the radial cracks have not propagated farther than one third the distance between the charge and the free face. If the tensile wave is strong enough to exceed the dynamic strength of the rock, the phenomenon known as spalling will come about, back towards the interior of the rock. The tensile strengths of the rock reach values that are between 5 and 15% of the compressive strengths.

4.4. Gas extension fractures

After the strain wave passes, the pressure of the gases cause a quasi-static stress field around the blasthole. During or after the formation of radial cracks by the tangential tensile component of the wave, the gases start to expand and penetrate into the fractures. The radial cracks are prolonged under the influence of the stress concentrations at their tips. The number and length of the opened and developed cracks strongly depend upon the pressure of the gases, and a premature escape of these due to insufficient stemming or by the presence of a plane of weakness in the free face could lead to a lower performance of the explosive energy.

4.5. Fracturing by release-of-load

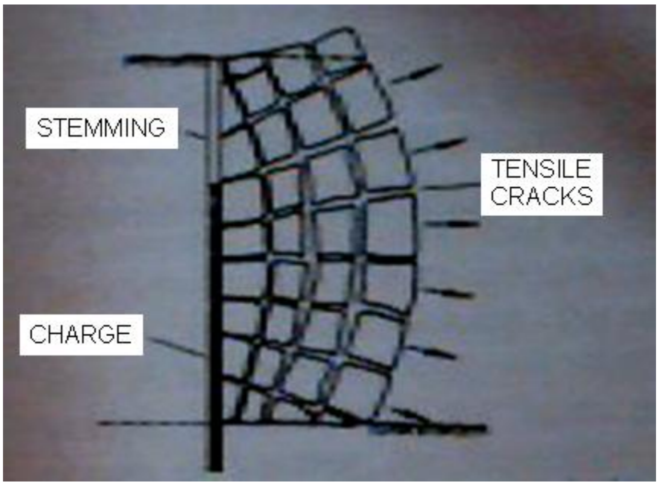
Before the strain wave reaches the free face, the total energy transferred to the rock by initial compression varies between 60 and 70% of the blast energy. After the compressive wave has passed, a state of quasi-static equilibrium is produced, followed by a subsequent fall of pressure in the blasthole as the gases escape through the stemming, through the radial cracks and with rock displacement. The stored stress energy is rapidly released, generating an initiation of tensile and shear fractures in the rock mass.

4.6. Fracturing along boundaries of modulus contrast of shear fracturing

In sedimentary rock formations when the bedding planes, joints, etc., have different elasticity modulus or geomechanic parameters, breakage is produced in the separation planes when the strain wave passes through because of the strain differential in these points.

4.7. Breakage by flexion

During and after the mechanisms of radial fracturing and spalling, the pressure applied by the explosion gases upon the material in front of the explosive column makes the rock act like a beam embedded in the bottom of the blasthole and in the stemming area, producing the deformation and fracturing of the same buy the phenomena of flexion.



4.8. Fracture by in-flight collisions

The rock fragments created by the previous mechanisms and accelerated by the gases are projected towards the free face, colliding with each other and thereby producing additional fragmentation.

t = 0 ms

t = 150 ms

t = 360 ms

t = 540 ms

t = 840 ms

PHOTOGRAPHS TAKEN FROM A BENCH BLASTING OPERATION

ONLY 15% OF TOTAL ENERGY GENERATED IS USED AS A WORKING TOOL IN THE MECHANISMS OF ROCK FRAGMENTATION

STRIPPING METHODS

1. SELECTION OF STRIPPING METHODS

The size of an orebody and the distribution of values within that orebody normally will limit the variety of economical stripping methods which need to be considered. Much may depend upon the selectivity required in the mining due to the relationship between ore and overburden, as well as the character of the overburden itself.

The following factors concerning the geologic nature and environment of an orebody, as well as the production requirements, must be determined before any selection of equipment is made:

1. The size of the orebody and distribution of the values within that orebody. Is the ore massive or scattered, bedded or disseminated, thick or thin?
2. The nature of the overburden to be removed. Is it a hard dense rock, bedded rock, friable material, earth, sand, clay, marsh, etc.?
3. The character and significance of geologic structures (fractures, faults, shear zones, etc.) associated with the ore occurrence. Are there water bearing formations with resulting water disposal problems?
4. In considering the nature of the overburden, including its alteration products, the physical or chemical conditions which may render certain equipment inoperable during unfavorable seasons.
5. The life and expected production rate of the operation. Is the production to be continuous or intermittent?
6. The calculated capacity of, and haulage distance to, each disposal area.
7. The future use of the equipment. Is it to be utilized to mine the orebody as it is developed, or is it to be used for stripping only? What is the effect of ore blending requirements on equipment size when it is also to be used to mine ore?

The character of the terrain at and in the vicinity of the orebody and the proximity of the orebody to waste disposal areas will greatly influence the cost of stripping and the selection of equipment. The possible need to reclaim the land following mining may be necessary if legislative trends continue. Such conditions may strongly influence equipment choices.

Unit capacities of earth moving equipment of all types have increased tremendously in recent years. With a variety of equipment available, the selection of a stripping method becomes a problem requiring careful analytical methods.

In the selection of a stripping method preliminary considerations, such as type of material to be moved, accessibility, size of job, volume per day, and type of power available at the site, will usually narrow the field to one or two possibilities. These should then be examined in detail using standard cost analysis techniques. A listing of the attributes of the various types of equipment available should help refine the selection possibilities.

EXCAVATORS

Shovels

1. Can give high production.
2. Can handle all types of material including large blocky material.
3. Are limited to fairly rigid operating conditions.
4. Require supporting equipment for waste disposal except in some strip mining.

Draglines

1. Have the ability to dig well above and below grade.
2. Can function under less rigid operating conditions than shovels.
3. Are only 75 to 80% as efficient in production as a shovel of comparable size due to less precise motions.
4. May or may not require supporting waste haulage equipment.
5. Are normally used for handling unconsolidated and softer material, but larger units can handle blasted rock

Scrapers

1. Have excellent mobility.
2. Are limited to fairly soft and easily broken material for good production.
3. Usually require pushers to assist in loading.
4. Usually are operated without supporting disposal equipment where the distance to the dump area does not exceed one mile.

Bucket-Wheel Excavators

1. Must be operated under very rigidly engineered conditions.
2. Have very high capital cost.
3. Are limited to fairly easy digging.
4. Are capable of high production rates.
5. Require auxiliary disposal systems.

HAULAGE EQUIPMENT

Bulldozers

1. Are economically limited to a fairly short operating radius of about 500 ft.

Scrapers

1. Require good roads to minimize tire costs.
2. Are fast but are economically limited to an operating radius of approximately one mile.

Trucks

1. Require good roads to minimize tire costs.
2. Can negotiate steep ramps.
3. Are usually limited by economics to an operating radius of about 2.5 miles.
4. Are very mobile.

Trains

1. Are high-volume, long-distance, low-unit-cost carriers.
2. Track requires careful conformity to engineering specifications.
3. Have a high initial capital cost.
4. Cannot handle adverse grades much greater than 3%.
5. Can handle coarse, blocky material.

<u>Conveyors</u>

1. Are high-volume, long-distance, low-unit-cost carriers.
2. Are difficult and costly to move.
3. Have a high initial capital cost.
4. Can handle steep adverse grades (up to about 40%).
5. Require material broken into fairly small pieces for good belt life.

Some of the principal factors affecting the cost of open pit mining are the size of the operation, the kind of material mined, and the distance it is moved. As a rule, the cost per ton tends to decrease with increased production, larger equipment, decreasing haulage distances, and easier handling material. There are many other factors, of course, which enter the cost picture.

Cost variations of drilling, blasting, and loading generally amount to only a few cents per ton. Haulage, on the other hand, not only accounts for a substantial portion of the direct mining cost but also the most variable single cost item.

Open Pit Unit Mining Cost	$/ton mined	
Drilling	0.157	15%
Blasting	0.127	12%
Loading	0.135	13%
Hauling	0.226	21%
Roads & Dumps	0.183	17%
General Mine	0.085	8%
General Maintenance	0.017	2%
Supervision & Technical	0.125	12%
Total Mining, ($/t)	1.056	100%

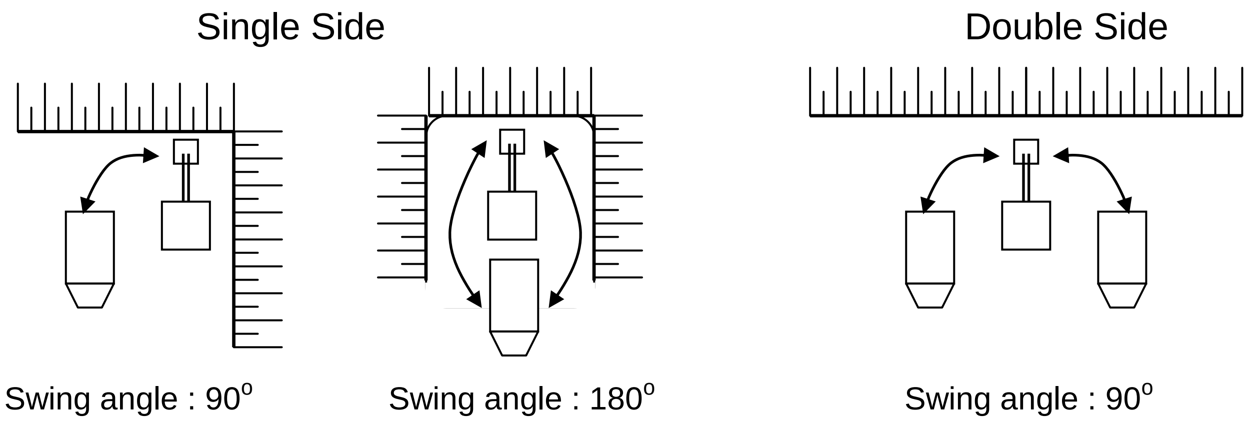
The major haulage systems are rail, conveyor, truck, and scraper; skips and pipelines are additional, but limited systems. Costs vary according to distance, although not in direct proportion. In general, rail is cheapest for very long hauls, conveyors for long hauls, trucks for short hauls, and scrapers for very short hauls.

2. SHOVEL-TRUCK STRIPPING

The shovel-truck combination is commonly selected for one or more of the following reasons:

1. The overburden is rock which breaks into large angular pieces.
2. There is limited access room.
3. Hauls involve short steep grades.
4. Extreme mobility is required.
5. Haulage is of medium length.

According to working conditions, two main kinds of loading can be figured as :

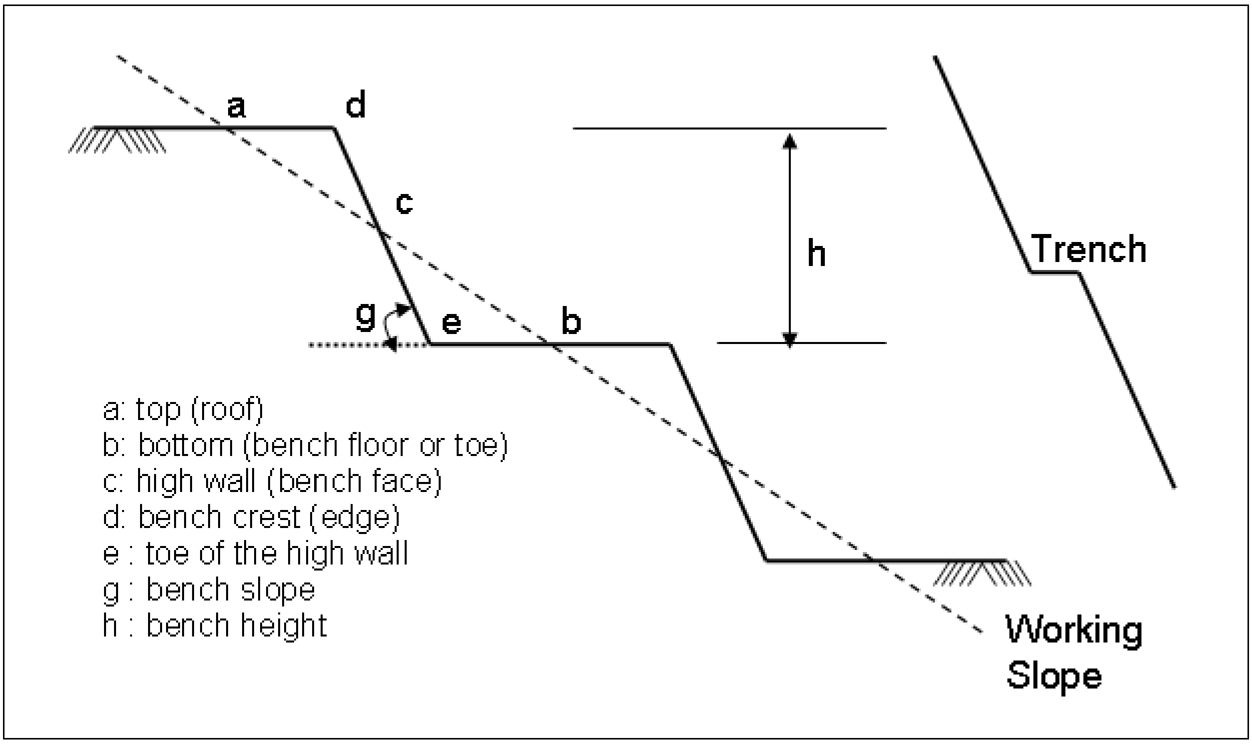


3. SHOVEL-TRAIN STRIPPING

The use of trains as the haulage unit for a stripping operation should be considered when the following conditions exist:

1. The operation will last long enough to amortize the high initial investment.
2. The haul is long (In established rail system the average haul is usually more than two miles).
3. Grades can be held to a minimum, usually not to exceed about 4% in favor of a load and about 3% adverse to a load.
4. The rigidly engineered haulage system will not seriously impair the progress of the stripping.
5. The material to be hauled is in a large, rough and blocky form.

Basic elements of an open-pit bench;



Problem : CALCULATION OF SHOVEL AND TRUCK NEED FOR AN OPEN-PIT STRIPPING OPERATION

An amount of $3*10^6$ m^3 (bank measure) overburden material covers a lignite deposit is planned to be stripped. Because of climatic conditions only 240 days can be worked annually and 2 shifts (8 hours/shift) work per day. If shovel-truck combination is planned to remove the overburden material, determine the required number of shovels and trucks (Use the following inputs).

Shovel capacity, α : 3.5 m^3
Truck capacity, α_t : 21 m^3
Loaded truck speed, V_d: 30 km/hr
Shovel cycle time, t : 25 sec
Distance to dumping area, L : 3 km.
Percent swell, θ : 0.3 (=30%)
Empty truck speed, V_b : 50 km/hr
Total manoeuvre time, t_m: 3 min

Daily and hourly stripping amount (in-situ) according to annual amount of working days:

$M_{dg} = 3*10^6 / 240 = 12500\ m^3/day$ $M_{ds} = 12500 / (2*8) = 781.25\ m^3/hr$

Practical hourly production capacity (in-situ) of shovel is determined by using the following equation which takes swell factor, organization efficiency (overall) and shovel productivity (as time) factors into account:

$$V_{pr} = \frac{\alpha}{1+\theta} * \frac{3600}{t} * R_{şan} * R_{zam} \ ....m^3/hr$$

In this equation,

θ is percent swell of excavated material. Its value greately changes with material properties and size distribution. It is generally taken between 10% and 50%.

t is shovel cycle (loading+swinging to truck+dumping+swinging to face) time. It depends on machine abilitiy, overburden blasting efficiency and face conditions. Thus the cycle time generally changes up to 40 seconds.

3600 is constant used to convert hour into second (…3600 sec/hr)

$R_{şan}$ is organization efficiency factor. If it is not given, it can be valued as 0.8 (means 80% efficiency) in the equation.

R_{zam} is a factor which used to define efficienct working time of a shovel (or truck) in a time interval (i.e., shift). If it is not given, it can be valued as 5/6 (means 50 mins in an hour) in the equation.

$$V_{pr} = \frac{3.5}{1+0.3} * \frac{3600}{25} * 0.8 * \frac{5}{6} = \frac{50400}{195} = 258.46 \quad m^3/hr$$ bank measure

Then the number of shovel needed is:

$n_e = 781.25/258.46 = 3.02$ is taken as 3 shovels.

To determine number of trucks for per shovel, we use:

$$n_k = \frac{3600 * \alpha * R_{şan}}{t * t_ç * \alpha_t} * \left(\frac{60 * L}{V_d} + \frac{60 * L}{V_b} + t_m\right)$$

In this equation,

$t_ç$ is truck running time (efficient). It can be taken as 50 min/hr (=5/6).

t is shovel cycle (loading+swinging to truck+dumping+swinging to face) time. It depends on machine abilitiy, overburden blasting efficiency and face conditions. Thus the cycle time generally changes up to 40 seconds.

t_m is total manoeuvre time in minute. It is sum of loading time (t_y), manoeuvre time at dump site (t'_m) and dumping time (t_b).

60 is a constant used to convert hour into minute (…60 min/hr)

$(60*L)/V_d$ is loaded travelling time of truck to dump site, in minute.

$(60*L)/V_b$ is empty travelling time of truck to excavation site, in minute.

$$n_k = \frac{3600*3.5*0.8}{25*50*21}*(\frac{60*3}{30}+\frac{60*3}{50}+3) = \frac{10080}{26250}*(6+3.6+3) = \frac{10080*12.6}{26250} = 4.84$$

As a result, 5 trucks are determined for one shovel. But one additional truck must be assigned for each shovel to prevent any interruption of loading because of fuel needs, simple failures, etc. Hence the total number of trucks needed for the operation is;

n_k = 3 shovels*(5+1 trucks/shovel) = 18 trucks.

SURFACE EQUIPMENTS

– Bulldozers
  • Tracked
  • Rubber Tire Dozer (RTD)
– Scrapers
– Haul Trucks (Heavy Haulers)
  • Rear Dump
  • Bottom Dump
  • Side Dump
  • Water Trucks
– Front-End Loaders (FELs)
– Hydraulic Excavators
  • Hydraulic Shovels
  • Hydraulic Hoes

– Electric Shovels
  • Stripping Shovels
  • Loading Shovels
– Draglines
– Bucket Wheel Excavators (Continuous Miners)
– Blast Hole Drills
  • Rotary Drills
  • Rotary-Percussion Drills
– Dredges

Bulldozers

• A dozer is a crawler or wheel driven tractor with a front mounted blade for digging and pushing material
• Dozers can come with numerous attachments to increase versatility, such as a rear-mounted ripper
• There are also a number of different blade designs

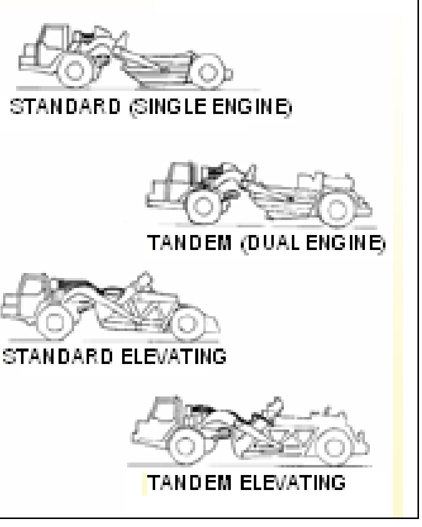
- Generally move at low speeds
- Have varying horsepower dependent on make and model
- Can be rigid framed or articulated (tracked or wheeled, respectively)
- Can grade to approximately 45°
- The capability of the dozer is dependent on the weight and power of the particular machine but is limited by traction; track slip will occur before maximum power is achieved
- One of the simplest pieces of equipment in terms of its basic operations and operator training
- High operational versatility
- Good maneuverability and mobility
- Good stability
- Highly reliable

    - Used to both excavate and transport material over short distances
    - Dozer applications include:
        - Earthmoving
            - Land clearance
            - Stripping overburden
            - Grading and leveling
            - Feeding conveyors
            - Trapping for loaders
            - Reclamation
        - Ripping (fragmenting and loosening consolidated material)
        - Pushing other equipment such as scrapers or other dozers
        - Utility work
        - Stockpiling and blending
        - Snow removal
        - Road maintenance

Scrapers

- Scrapers are unique in their ability to excavate material in thin horizontal layers, transport the material considerable distances, and then discharge the material in a spreading action
- Modern scraper units have rubber tires and are self-propelled high-powered machines with unusually high transport speeds and good maneuverability
- Load capacity varies depending on the machine

• Typical scrapers are of two axle articulated body design
• There are four basic scraper configurations
  1. Standard (single, front mounted engine on tractor, two wheel front axle drive)
  2. Tandem (front mounted engine on tractor and rear engine on scraper body, two axle four wheel drive)
  3. Elevating (single, front mounted engine on tractor, front axle two wheel drive)
  4. Tandem elevating (front mounted engine on tractor and rear engine on scraper body, two axle four wheel drive)

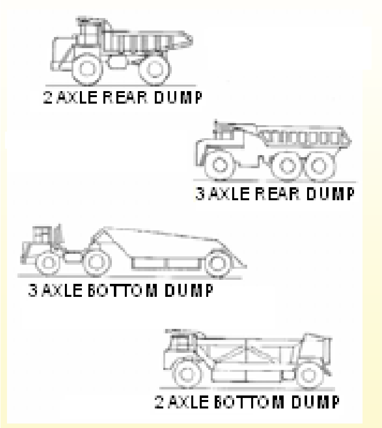


• Scraper applications are versatile, and include:
  – Topsoil removal
  – General reclamation
  – Overburden removal with or without prior ripping
  – Ore removal with or without prior ripping
  – Parting removal
  – General utility work
    • Stockpiling
    • Reclaiming
    • Site preparation
    • Road construction
    • Construction of ponds and ditches

• If the power demand of efficient scraper loading exceeds that generally available a scraper may be teamed with a dozer for push loading, or two scrapers may pair together for push-pull operation utilizing their combined power
• Hauling and returning speeds are limited by:
  – Road conditions
  – Traction
  – Safe speeds
  – Traffic
  – Grades (tandem scrapers can handle up to 25% grade when empty, but only 12% when loaded)
• Wet conditions restrict operations
• Tire wear can be severe if operating on ripped or blasted rock

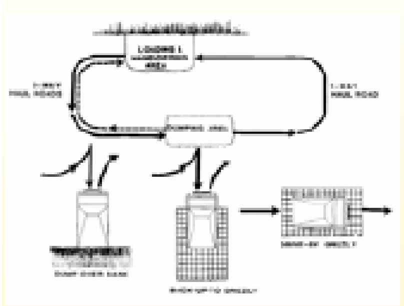
Haul Trucks

- A haul truck is simply a mobile piece of equipment for hauling material
- The two basic truck designs utilized in surface mining include the rear dump and the bottom dump
- Rear dumps generally have rigid frames and heavy-duty bodies, front axle steering, and dual wheels on the rear axle(s)
- Bottom dumps are generally tractor-trailer three axle units or rigid frame two axle units with various combinations of single or dual wheels.



• Horsepower and vehicle weight vary from model to model
• Capacity varies greatly and is the focus of much research
• Trucks are filled (loaded) by front-end loaders, hydraulic excavators, shovels, and/or bucket wheel excavators, while the truck is positioned so as to optimize the operation cycle of the specific excavating equipment
• If the units are properly matched cyclic loading machines normally require three to five passes to fill the truck (Anything over six passes is not economic)

• The haul cycle includes:
  – Waiting and positioning time at the load site
  – Loading time
  – Loaded traveling time
  – Waiting and positioning time at the dump site
  – Dumping time
  – Empty traveling time
• Minimization of waiting and positioning time increases effectiveness and decreases costs
• Consideration must be made for the different traveling speed and maneuverability of a loaded or empty truck as well as variation in fuel consumption



• Truck performance and economy are tied closely to haul road quality factors such as:
  – Grades
  – Curve radii
  – Width
  – Surface conditions
  – Number of crossings
• Top transport speeds are limited by:
  – Safety
  – Traffic
  – Visibility
  – Weather conditions
  – Grade
  – Haul road surface conditions
  – Tire heating
  – Speed limits may be reduced at night. Trucks have a relatively low center of gravity and therefore comparatively low tripping speeds.

• Tires are a major operating cost item so their selection, care, and maintenance are prime factors in tire life
• Liners, wear bars, and plates are sometimes added to extend body life and reduce maintenance
• Due to the size of some of the larger mine trucks operator visibility can be a problem
  – Care must be exercised to restrict the movement of people and smaller vehicles in truck maneuvering areas
• Productivity per hour generally increases with truck size

Water Trucks

• Used to control dust on haul roads (– Improves visibility)
• Variation of haul truck. Typically a modified hauler with a large water tank instead of a box

Front End Loaders

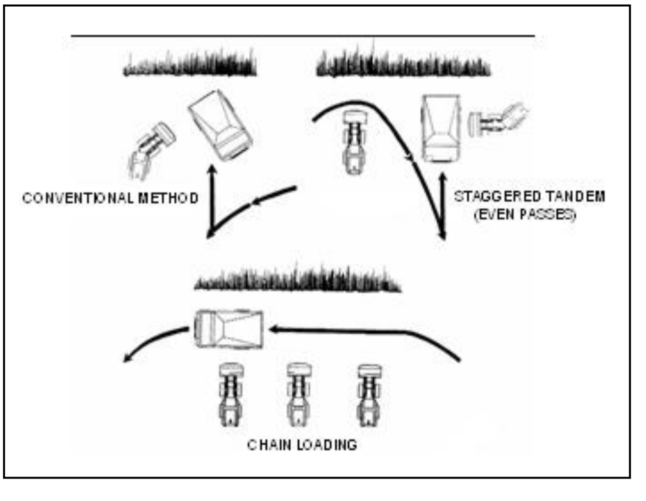
- Wheel or crawler mounted tractor with a front mounted bucket
- Utilized in:
  - Excavating
  - Loading
  - Transporting material
- Found in a wide variety of mining applications
- Articulated wheel mounted loaders are now common and have a larger bucket capacity than crawler loaders

- Typical units have articulated frames (70-90° steering)
- Four wheel drive
- Have varying capacity and horsepower depending on make and model
- Excellent maneuverability and mobility
- May be limited by operating surface conditions

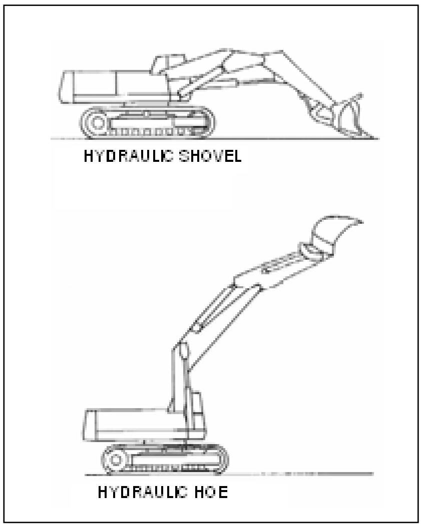
• *<u>Used for:</u>*
  - Loading / transporting topsoil, ore, overburden, and waste
  - Pit clean up
  - Stump and boulder removal
  - Road maintenance
  - Snow removal
  - Logging (with special attachments)
  - General utility

• *<u>Advantages:</u>*
  - Can operate at relatively high speeds
  - Good (small) turning radius
  - High break-out forces while digging
  - Good blending capabilities
  - Good cutting efficiency
  - Easy dumping action
  - Leave a clean floor
  - Can dig material selectively with a minimum of waste
  - Wide buckets permit handling of large pieces of rock

• *<u>Operational limitations include:</u>*
- Requirement of prior preparation of consolidated materials (material must be ripped or blasted before the front-end loader can handle it)
- Reduction of performance in poor surface conditions (wet ground, soft ground, clay surfaces)
- High tire wear
- Minimal reach
- Reduced stability in load and carry position
- Relatively poor visibility rearward while maneuvering



Hydraulic Excavators

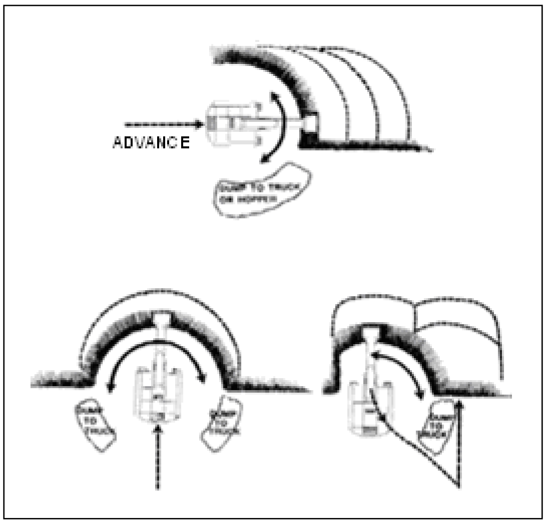


- Hydraulic hoes and shovels, originally designed for construction projects, have now increased in size to the point where units are common in surface mining applications
- The difference between a hoe and a shovel is the orientation of the bucket and front-end geometry

- Basic operating cycle consists of:
  – Cutting pass through the bank
  – Loaded swing to the discharge area
  – Dumping
  – Empty swing back to the digging face
- As a hoe, the machine digs downward and back towards itself
- As a shovel, it digs upward and away from itself
- The bucket is wristed (rotated) to orient the teeth for the desired penetration of the face
- The upper part of the machine rotates on the crawler base so that the machine can swing to the right or the left from the digging face and discharge the material from the bucket into a hopper, truck, or stockpile

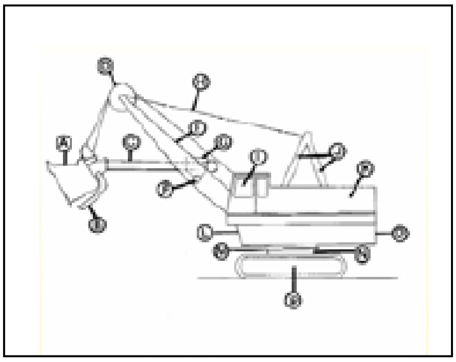
- Hydraulic machines are employed in overburden removal and ore loading
- Smaller versions may be used for utility work such as blending
- The hydraulic shovel is primarily an excavating and loading device
- Hoes have the same uses as shovels but have below grade digging capacity that makes them particularly suited to tasks such as trenching or excavating under water
  – Useful when floor conditions warrant keeping machines off the bottom of the pit

• *Advantages:*
– Excellent positioning capabilities (spin turns) with independent track drives
– Good ground clearance
– Good stability
– High swing speeds
– High breakout forces through wristing
– Versatility in bucket orientation for face penetration
– Smooth, low-shock dumping (dumping action can be controlled)
– Compact size
– Low weight
– Good operator safety (good visibility)



- Narrow bucket size can be an advantage (mining narrow veins) or a disadvantage (more passes required for wider ore bodies)
- Excavators can work in close quarters and load directly into trucks with good load placement
- They do, however, have a high initial cost

Electric Shovels



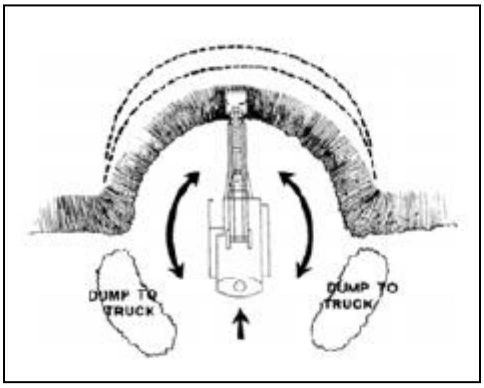
- Two types of electric shovels in service are the loading shovel and the stripping shovel (the primary difference being the size of the machine)
- Stripping shovels are no longer used
  – They have been superceded by the large walking draglines
- Electric shovels are used primarily for loading large off highway trucks, or mobile hoppers


LOADING SHOVEL


STRIPPING SHOVEL

- Typical units are crawler mounted
- Have full revolving upper works
- Cable hoist system
- Fully enclosed machinery house
- Quite large and heavy
- Basic machine operation includes:

– Periodic propelling for relocation
– Digging (a combination of vertical hoisting and horizontal motion 'crowd')
– Dumping (releasing the bottom of the dipper)



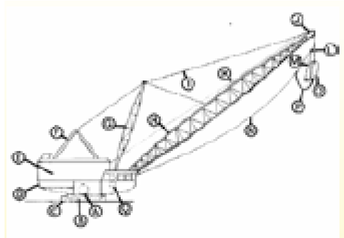
- Rotating upper works allow swinging the boom and dipper to the right or left for discharge
- Applications of electric shovels are similar to those of hydraulic shovels, but electric shovels are considered more suited to more severe digging conditions
- Are available in larger sizes and have a proven service record in multishift mining operations as well as longer range capabilities

• Advantages:
– Long cutting heights and dump heights
– High sustained production capability
– Rugged construction suited to tough digging conditions
– Requirement of minimal blasting
– High reliability
– Good stability
– Good blending vertically at the face
– Optional front-end lengths for different digging geometry

• Disadvantages:
– Limited capability for selective loading
– Low propelling speeds (less than 1 mile per hour)
– Limited mobility between faces
– Relatively fixed digging paths
– Hard dumping action on trucks
• Altitude, temperature, and humidity may affect electrical efficiency

- Perform well on bad floor conditions
- Have the capability to dig below floor level
- Good operator safety
- Require a highly trained crew of one or two
- Maintenance is performed on site (in the pit)
- Require a trailing cable which must be kept clear of the truck maneuvering area
- Have a long service life (over twenty years)
- Low operating cost
- High capital cost

Draglines



- Draglines have both long reaches and the ability to dig to substantially below their own base
- Have broad applications in irrigation projects and in surface mining
- Are now the largest pieces of mobile equipment manufactured

  - Two types of dragline are available: crawlers and walkers
  - Walking draglines are generally larger and are electric drive with trailing cable
  - They propel themselves forward at very low speeds
  - Can weigh substantially over a million pounds
  - Machine mobility is restricted by large machine size and the trailing cable
  - The walking device consists of large rectangular shoes attached by a cam arrangement to either side of the main deck
  - Located behind the machine center of gravity

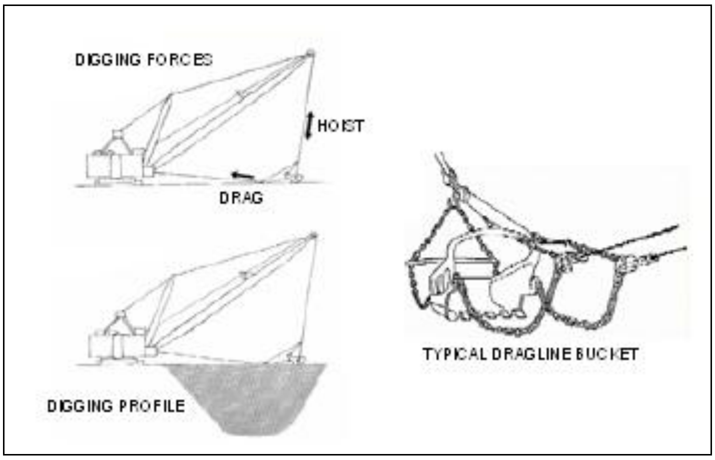
- To move the machine the cams are rotated to lower the shoes to the ground and then lift the rear of the machine and slide it forward
- The cam then lowers the machine to the ground, lifts the shoes, and shifts them back to reposition them for another step sequence
- Each walking step takes about 40 seconds
- During a step about 70-85% of the machine's weight rests on the shoes and drags
- The unit can be rotated to walk in any direction
  – Walking process takes place in reverse, with the boom pointing in the wrong direction

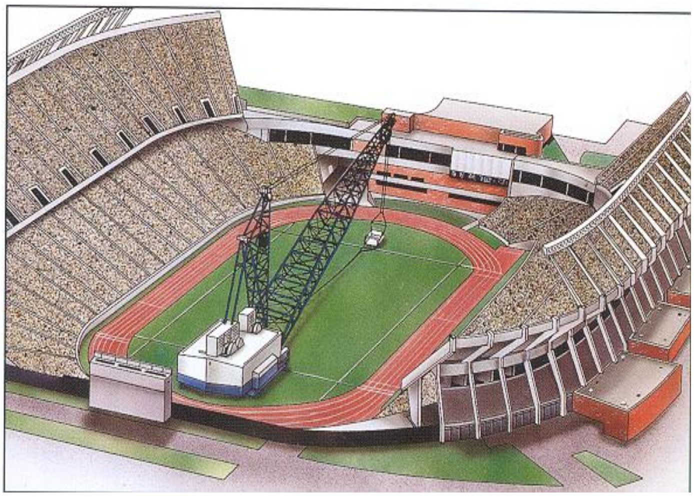
  - Applications of the dragline include material excavation wherever the material being excavated only needs to be transported a short distance (such as strip mining)
  - The dragline is considered both an excavating and a transport system as it can transport material the length of the boom
  - This can eliminate secondary haulage requirements associated with other systems
  - Due to its essential stationarity the dragline is not significantly impacted by adverse weather conditions
  - Smaller draglines (such as the crawler type) are often used for truck loading, excavation of ponds / basins, reclamation leveling, and removal of partings in multiple coal seam mining operations

- Manned by a highly trained crew (one or two people)
- Have:
  - Good operator visibility
  - Removable counterweights
  - Full walk-in machinery house
  - Medium service life
  - Full revolving upper works
- Perform well in medium to hard digging conditions
- Maintenance is performed at the site of the machine

- *Disadvantages:*
  - High initial cost
  - Limited mobility
  - Inconvenience caused by the trailing cable (with electric walkers)

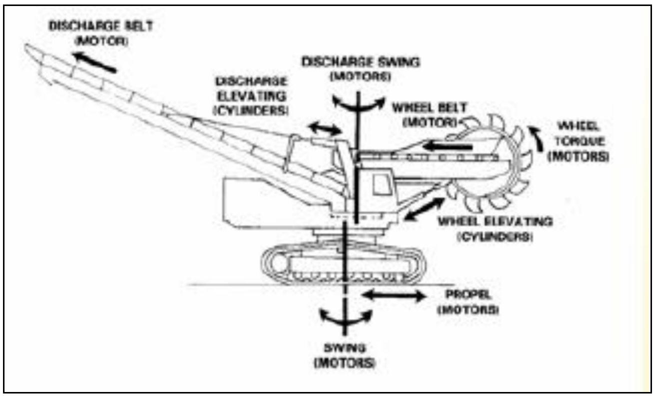
- Auxiliary equipment may be required to do cleanup at floor level and prepare flat operating benches if equipment is to be used at the bottom of a pit created by a dragline
- Walking draglines are unique in their design in that each powered function is an independent module with (in most cases) multiple motor drives
- This allows broad componentry standardization in the motordrive systems, and, in the event of a single drive failure, continued operation
- However, the way draglines are generally put to use means that mining is dependent on operation of the dragline
- As such, in the event of equipment failure the mining operation ceases
- This means additional planning must be done and stockpiles prepared to prevent total shutdown of operations if the dragline has a problem requiring more than quick repairs

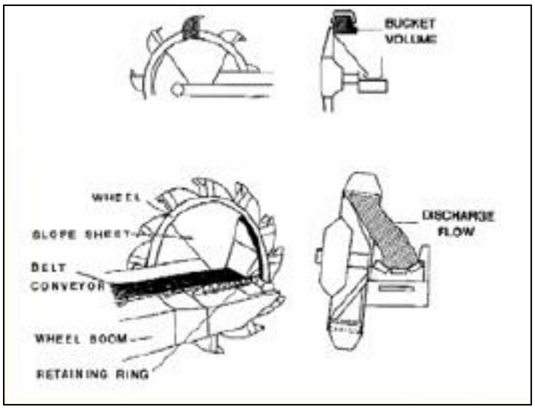




Bucket Wheel Excavators

• Wheel excavators dig with a rotating bucket wheel
• This wheel discharges the material onto a belt conveyor, which then transports the material until it is discharged from the machine
• Unlike loaders, shovels and draglines (all cyclic excavators), bucket wheel excavators are continuous mining systems
• Have been used extensively in Germany (but not as much in North America)
• Three types of bucket wheel excavators, classified by bucket size (large, medium and small)
• Large wheel models are used for pre-stripping for large shovels
• Other types are used for full-face mining operations

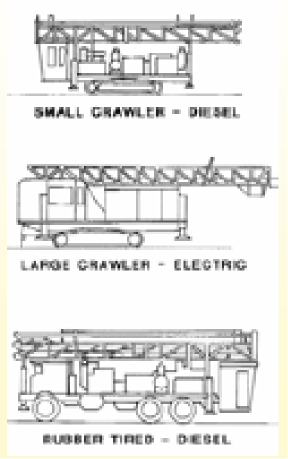




- All units are crawler types with digging height, capacity, and power requirements dependent on the wheel size
- Multiple small buckets mounted on the circumference of a rotating wheel structure take progressive slicing side cuts as the wheel is swung back and forth across the digging face (slewing)
- Wheel rotation produces an upward cutting action
- Material collected in each bucket is initially retained by a fixed internal ring segment that acts as the back (or bottom) of the bucket
- This ring segment only closes off the buckets behind the digging sector of the wheel (where actual cutting occurs)
- As the wheel rotates, the material in each bucket falls out when it reaches the top position, where there is no closing ring
- There is a relatively large discharge zone for each bucket to dump its load while the wheel continues to rotate
- There is sufficient overlap between succeeding buckets to effectively produce a continuous flow
- Cut depth is controlled by swing speed
- Actual output is about one half the theoretical capacity
- Applications include:
  - Overburden removal
  - Large earthmoving projects
  - Coal excavation
  - Reclamation leveling
  - Topsoil removal

- Bucket wheel excavators can dig medium hard consolidated materials without prior blasting
- Are generally applied in stable materials with low cutting resistance (such as potash)
- Cannot economically handle large boulders, frozen ground, sticky material or hard rock
- Design is highly customizable (wheel size, bucket number, discharge belt setup, swing ability, etc)
  - Bucket wheel excavators can be specifically designed for a particular mining situation
- Units are electric with trailing cable for power supply (they have relatively uniform power demand due to continuity of mining)
- Buckets and bucket teeth are removable/replaceable–maintenance is performed in the pit
- Material can be discharged above or below the working level
- Good operator visibility with respect to the wheel
- Auxiliary means are required for viewing discharge (such as television systems)
- Crew size is dependent on bucket size
- Other disadvantages:
  - Low propelling speeds and limited pit mobility
  - Requirement of a level operating bench
  - Requirement of auxiliary equipment for floor clean up

Blast Hole Drills



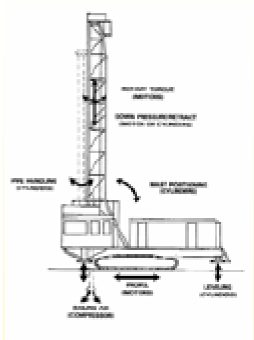
- When material is too hard or consolidated to be mined using any of the equipment previously mentioned it must first be broken
- This is generally done with explosives
- Explosives are placed in holes drilled into the formation to depths roughly equal to the depth of the planned excavation
- These holes have a diameter that will accommodate the characteristics of the explosive being used, which, in turn, is matched to the power required to produce the desired fragmentation in the specific formation

- Blast hole drills are used in surface mining operations to produce the holes for explosive placement
- Small percussion drills may be used or more popular rotary units (crawler type and wheel mounted units)
- Crawler mounted units can drill larger diameter holes than wheel mounted units
- Wheel mounted units have commercial truck chassis

- *Basic operation consists of:*
  - Maneuvering the machine into position
  - Raising the mast which supports the drilling mechanism
  - Horizontal leveling (so holes are drilled at a continuous specified angle, or vertically
  - Raising the wheels / crawlers off the ground to ready for the drilling process
  - Drilling (forcing a rotating bit into the ground with a high down pressure and rotary torque)
  - The reach of the cutting bit may be extended during the drilling process, as needed, by adding lengths to the drill pipe
- Chips formed by cutting are removed from the hole by high velocity air
- Brought back up to the surface and can be caught for sampling purposes
- Once the hole is completed the drill pipe and bit are brought up and the cycle begins again at the site of the next hole
- There is much variation in blast hole equipment depending on manufacturer as well as on the designed blast characteristics
- Dust control can be a problem, but dust control systems are available

- *<u>Advantages:</u>*
  - Good mobility under poor ground conditions
  - One man operations and require no support equipment
  - Medium to long service lives
  - Rugged construction
  - Enclosed operator stations
- Initial investment is system dependent
- Drill bit configurations vary depending on the material to be drilled
  - Soft such as shales, calcite, clay
  - Medium such as gypsum or sandstone
  - Medium hard such as limestone, dolomite, or porphyry copper
  - Hard such as granite, iron ores, or quartzitic materials
- As the operator cannot see the bit or the chips coming up from the hole, the machine's gauges must record all the information necessary to run the hole

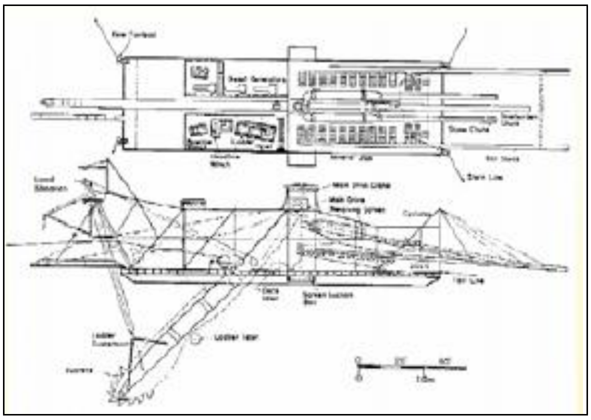
- Machine selection considerations include:
  - – Material properties (hardness, compressive strength, abrasiveness and toughness
  - – Hole size capability
  - – Machine size (larger machines are more rugged and can generally drill in harder formations)
  - – Drill pipe length
- A machine that can handle drill pipe long enough to permit single pass drilling can significantly improve productivity
  - – Production rate
  - – Operating cost
  - – Power requirements
  - – Mobility
  - – Dust control
  - – Level of automation
  - – Reliability

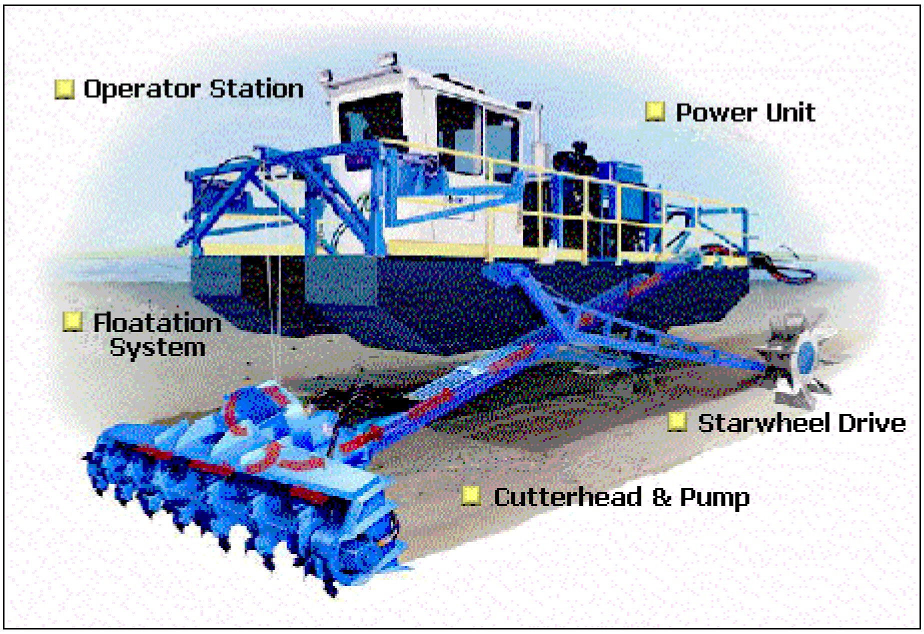


- Options which can be added to the basic machine include:
  - – Stabilizers
  - – Anti-shock drill coupling
  - – Automatic monitoring equipment
  - – Centralized automatic lubrication
  - – Depth of drilling indicator
  - – Night lighting
  - – Special mufflers
  - – Low temperature starting capabilities
  - – Vandalism protection systems
  - – Fire extinguishers
  - – Operator comfort / convenience items

Dredges

- A typical dredge is essentially a floating bucket wheel excavator
- Scoops slurry material from the bottom of a water body and brings it on board for processing
- Waste material is dumped off the back end of the dredge
- Very high productivity
- High efficiency
- Low mining cost
- Very limited field of applications





EXAMPLES RELATED TO SHOVEL-TRUCK COMBINATION

Example 1:

A truck is loaded 80 tons of coal by a shovel of 20 yd^3 capacity. If the density (loose) of coal is 1500 lbs/yd^3 and the fill factor of shovel is 0.8 how many dipperfulls are required to fill each truck?

$C_d = 20\ yd^3$
F = 0.8
d = 1500 lbs/yd^3
Truck capacity = (80*2000)/1500 = 106.67 yd^3 where 1 tons (short) = 2000 lbs

# of dippers = 106.67 / (20*0.8) = 6.67 ➔ 7 dipperfulls

Example 2:

A truck is loaded 85 tons of coal by a shovel of 15 yd^3 capacity. If the density (loose) of coal is 1400 lbs/yd^3, and the fill factor of shovel is 0.9 how many dipperfulls are required to fill the truck? If the cycle time of shovel is 32 sec and time availability is 50 sec/min how many trucks can be loaded per hour?

F = 0.9
d = 1400 lbs/yd^3 (loose)
t_s = 32 sec
E = 50/60 = 0.83
Truck capacity = (85*2000) / 1400 = 121.43 yd^3 (loose volume)

# of dippers = 121.43 / (15*0.9) = 8.99 ➔ 9 dipperfulls

The first major step in shovel selection is the determination of DIPPER SIZE

$$C_d = \frac{Q.t_s.S}{3600.E.F.D.A}$$

C_d = Dipper capacity (yd^3 or m^3)
Q = Production required (bank volume/hour)
t_s = Shovel cycle time (sec.)
E = Efficiency factor (time utilization factor)
F = Dipper fill factor (dipper efficiency)
D = Depth of cut correction
A = Angle of swing correction
S = Swell factor

C_d = (Q*32*S) / (3600*0.83*0.9*D*A) = 15 yd^3 then
Q = 1260.6 yd^3/hr (Bank volume) (S,D,A=1 assumed)

If S=1 assumed, truck capacity is valid for bank measure as well, then;
# of trucks per hour = 1260.6 / 121.43 = 10.38 ➔ 11 trucks/hr

Example 3:

If the followings are given:

Stripping required per year = $13.5*10^6$ m^3 (Bank measure)
Shovel dipper capacity = 15 yd^3
Truck capacity = 65 tons (short)
Working days in a year = 270 days
Daily working time = 18 hours (≈7 hrs/shift, 50 min/hr)
Cycle time for shovel = 33 sec
Fill factor of shovel = 0.9
Bank unit weight of overburden material = 2.50 tonnes/m^3
Percent swell of overburden = 60% (Swell factor = 1.6)
Overall efficiency = 0.8
Spotting time for truck = 35 sec
Hauling time = 220 sec
Return time = 190 sec
Dumping time = 70 sec

a) Find the number of dippers required to fill the truck
b) What would be the actual truck output per hour?
c) What would be the actual shovel output per hour?
d) Find the number of trucks required for one shovel
e) Find the total number of shovels and trucks required

Working hours/year = 270*18 = 4860 hr/year
Truck capacity = 65*0.907 = 58.96 tonnes 1 tons (short) = 0.907 tonnes
Bank volume = 58.96/2.5 = 23.58 m^3
Loose (Swell) volume = 23.58*1.6 = 37.73 m^3
Shovel capacity = 15*0.7645 = 11.47 m^3 (Loose volume) 1 yd^3 = 0.76455 m^3

a) # of dippers = 37.73 / (11.47*0.9) = 3.65 ➔ 4 dippers

b) T = 35+190+70+220+132 = 647 sec (loading time = 4 dippers * 33 sec = 132 sec)

$P_t = (3600*C_t*E) / T = (3600*37.73*0.8) / 647 = 168$ m^3/hr (Loose) or
$P_t = (3600*C_t*E) / T = (3600*23.58*0.8) / 647 = 105$ m^3/hr (Bank)

c) $C_d = (Q*33*1.6) / (3600*0.8*0.9*1*1) = 11.47$ m^3 then (D,A=1 assumed)
Q = 563.07 m^3/hr (bank measure)

d) Number of trucks per shovel = 563.07/105 = 5.36 ➔ 6 trucks/shovel (0.36>0.2)

e) Yearly production of a shovel = 563.07*4860 = 2736520.2 m^3/shovel (Bank volume)
# of shovels = ($13.5*10^6$ m^3/year) / 2736520.2 m^3/shovel = 4.9 shovels
5 shovel+1 spare = 6 shovels
6 trucks/shovel*6 shovel = 36 trucks + 5 spare = 41 trucks

Example 4:

In an open pit operation 85 tons (77 tonnes) truck is selected to haul overburden from the 835 m elevation to 970 m. The length of the haulage road is 1506 m. If the followings are known, find whether the truck can go uphill both loaded and empty.

Empty vehicle weight: 119000 lbs
Payload: 85 tons
Load on drive axle (empty): 59000 lbs
Load on drive axle (loaded): 134000 lbs
Coefficient of traction: 0.3
Rolling resistance: 2%

traction
a. The act of drawing a vehicle over a surface and the force exerted in so doing. Traction is the friction developed between tracks or tires and the surface of the ground on which they are moving.
b. The total amount of driving push of a vehicle on a given surface.

coefficient of traction
Represents the percentage of the total engine power that can be converted into forward motion by means of the friction between tire and track.

rolling resistance
a. The sum of the external forces opposing motion over level terrain.
b. The tractive resistance caused by friction between the rails and wheels, which forms the major resistance on level tracks.

Grade Resistance = (970 – 835) / 1506 = 0.0896 ~ 0.09 = 9%
Gross vehicle weight (GVW) = 119000 + 85*2000 = 289000 lbs

R_eR = GVW*TR/100 TR = GR + RR = 9% + 2% = 11%
= 289000*11/100 = 31790 lbs (loaded)
UR = 134000*0.3 = 40200 lbs (loaded)
UR > R_eR ➔ Truck can go uphill (loaded)

R_eR = 119000*11/100 = 13090 lbs (empty)
UR = 59000*0.3 = 17700 lbs (empty)
UR > R_eR ➔ Truck can go uphill (empty)

DEWATERING AND FLOOD CONTROL

1. SOURCE OF WATER

Precipitation in the form of rain or snow is the original source of water that enters most surface mines. The precipitation, which does not escape to the atmosphere, either becomes surface run-off or infiltrates into the ground to become ground water. Evaporation and surface run-off can be measured. The water that infiltrates the ground that disappears from view and is more difficult to evaluate. The infiltrating water descends trough the soil and rock by whatever passageways permit flow until the water table or top of the zone of saturation is reached. Ground water may become surface run-off again at a lower point, and likewise, surface run-off lakes and screams can infiltrate to become ground water.

Should the ground water be confined to a permeable zone overlain by an impermeable barrier, a pressure is created. Such water is said to be artesian if the pressure head is sufficient to raise the confined water to the surface. These conditions are more commonly found in relatively flat-lying bedded formations.

2. RECOGNITION AND ESTIMATION OF WATER

Dewatering and flood control should be considered by the design engineer at any early stage in all surface mine planning. Circumstances, which should lead to investigation, are,

i. High rainfall
ii. Experience of other mines in the area
iii. Evidence of water and caving in drill holes
iv. Presence of springs and artesian wells
v. Geological evidence of aquifers.

The first step is to appraise the planned excavation in relation to the climate, topography and geology. Records of annual rain and snowfall should be examined for a number of years for the watershed in which the mine lies. It is important to determine seasonal variations and maximum short-duration precipitation rates, as these will indicate the size of peak surface flows. In colder climates, the rate at which snow melts must also be considered.

Collection of data on all stream flows in the mine watershed, together with a study of the topography, will indicate the amount of surface water that could flow into the mine and what can be done to divert it. Some forms of flow gauges can be placed on streams. A preliminary estimate of the amount of water that infiltrates and moves trough the ground is obtained by subtracting the evaporation and surface run-off from the precipitation. If a sizable proportion of the precipitation goes to feed the ground water, serious dewatering problems are indicated.

The factors affecting the flow of ground water into an excavation below the water table are given in a modified form of Darcy's eqn.

$Q = P\ I\ A$ where; Q is the quantity of water
P is the coefficient of permeability
I is the hydraulic gradient
A is the area of cross section

Permeability is difficult to evaluate quantitatively. However a reappraisal of the geology relative to permeability, porosity and impervious structure may be helpful. Rock with open joints and fissures are especially capable of transmitting large quantities of water. Uniform strata of great lateral extent make it possible to obtain permeability and other related data from well-pumping tests in order to estimate water flows.

The hydraulic gradient that provides the pressure for ground water flow is dependent on the elevation of the water table relative to the depth of the mine. The elevation of the water table can be determined from bore holes, existing water wells or deep excavations, geophysical surveys, locations of springs, and levels of lakes. Holes drilled for the exploration of the deposit should be checked for water and some should be retained as observations holes if water is found.

The area of cross section for water flow provided by the mine walls and even the pit floor is very large.

The flow rate, together with surface inflows and precipitation inside the mine perimeter; dictate the pumping rate to avoid flooding.

When dewatering is started, the water table is first lowered in the vicinity of the mine or wells where pumping takes place. The rate of pumping, less the rate of recharge, gives the rate at which water can be removed from the pore spaces in the rock. The maximum possible rate of pumping is dependent upon the flow rate, Q. The rate of recharge depends on the precipitation and permeability. If pumping exceeds recharge, the water is drawn from the formations, and the water table is lowered; but in effecting this, the water table must be lowered an increasingly larger basin. To lower the water table requires a higher pumping rate than that required maintaining it in a lowered position.

3. DETRIMENTAL EFFECTS OF EXCESS WATER

Excessive water in the mines can cause serious operating delays and increased expenditures. The heavy equipment used in surface mining does not operate efficiently under muddy or wet conditions. The cost of maintenance increases due to loss of lubricants and the abrasive action of muddy water. Ground water in blast holes cause hole caving and necessitate the use of more expensive slurry explosives or waterproof liners.

Excessive moisture in the ore and stripping necessities increased handling and processing expenditures, not only from the weight of the water but for many other reasons. Drainage and pumping are cheaper than drying ore prior to processing. Wet ore builds up on conveyors, sticks in crushers and in cold climates increases frost build-up.

Seepage and a high water table in the walls of deep open pits decreases bank stability considerably. Surface water can cause serious erosion of pit walls. In open cast operations, spoil banks may slump if drainage is not provided. Even the removal of water may have detrimental effects in adjoining property by causing surface disturbance and changes in subsurface drainage.

4. METHODS AND SELECTION OF METHODS

Methods used for dewatering include ditching, pumping from sumps, diversion to underground sumps, dewatering tunnels, shafts and drifts, and wells. Selection of the method or combination of methods is governed by the cost relative to the savings affected.

4.1. DITCHING

Diversion of surface water away from the edge of a pit is cheaper than pumping. Elaborate drainage ditches, dikes and even surface sumps are rapidly paid for if they prevent substantial flows from entering the pit.

When the coal lies above the natural drainage, the water is collected in ditches; it is carried by gravity to either or both ends of the mining area and discharged into natural drainage. When coal lies below the natural drainage, the collection system is similar; however, levees must be constructed at the discharge points to prevent water flood from entering the pit.

Ditcher is dug with a dragline and/or tractor on a gradient towards the pit ends. Usually 0.1 or 0.25 % is sufficient to prevent silt build up. Their cross-sectional area will be determined by the depth of cut, maximum rainfall and run-off factor of the topography.

Shallow ditches, following the contour around hills and discharging into the main ditches, are dug at regular intervals. These contour ditches are usually constructed 2 to 4 ft deep with a tilt-blade dozer or a road petrol grader.

4.2. PUMPING FROM SUMPS

Surface water collects in the lowest areas of the pits, and it is normal practice to dig a sump to confine the water, to provide sufficient depth to effectively operate pumps and to allow settling of solids prior to pumping. The size of the sumps required depends on the estimated peak inflow rate and duration relative to the size of pumping equipment. In semi permanent sumps, a stationary platform above the sump provides the most convenient and safe place to install the pumps. Pumps mounted on pontoons with a flexible hose connection to the pipeline are more quickly installed.

4.3. DIVERSION TO UNDERGROUND SUMPS

Many surface mines are connected to underground workings and it is found expedient to drain water to underground sumps. Sump pumping, while offering one of the cheapest forms of water control, can have disadvantages. They require generous operating room, they freeze in cold weather and the water is usually dirty and contains abrasive material. The discharge of discolored water from mines is objectionable in settled areas.

4.4. WELLS

Dewater trough wells (drill holes) is the common method employed for advanced dewatering provided it can be justified economically. While much can be learned from the literature on water supply wells that can be usefully applied to dewatering wells, there are important differences between the two applications. Dewatering wells must be designed to lower the water table as far as the work requires, as economically and fast as possible, and to hold it there during mining. Wells are grouped close together so that they produce an overlapping effect on the water table and lower it more rapidly than a single well. Water with impurities must often be pumped and pumping must be continuous.

4.5. PIPELINES

Pipeline systems to carry water from the pits should be carefully planned as an integral part of the pumping systems. Pipes of adequate size and carefully laid to a planned grade with few sharp bends, can reduce the friction losses and lower the effective head on pumping equipment. In cold climates pipeline systems must be planned to allow rapid and complete draining to avoid frozen lines.

5. PUMPS AND PUMPS SELECTION

5.1. PUMPS

Essentially all pumps finding application in surface mine dewatering and flood control are electric driven, centrifugal pumps. Three types find common usage in mines:

i. Horizontal centrifugal pumps
ii. Vertical turbine pumps
iii. Submersible motor-vertical turbine pumps.

5.2. PUMP SELECTION

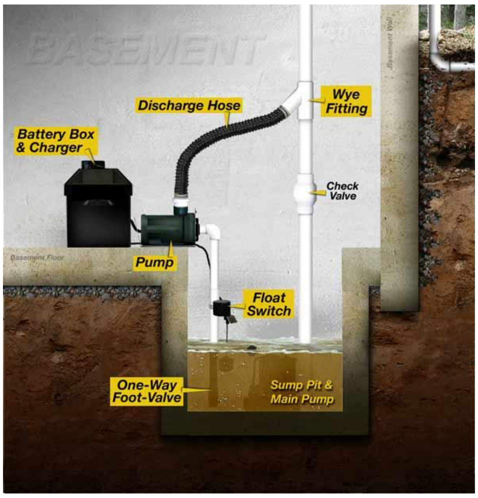
Basic criteria that must be supplied to order a pump are; quantity of water to be pumped, total head to be overcome, water temperature, acidity and content of silt and grit, type of electric power supply, conditions of service, and in the case of a well pump, depth and inside diameter of well.

Care should be taken in selecting pumping units so that the first units purchased will be of use in the future when the excavation is larger and deeper. Increased head can be overcome by adding booster pumps along the line, adding additional stages to vertical pumps, or placing horizontal units in series.

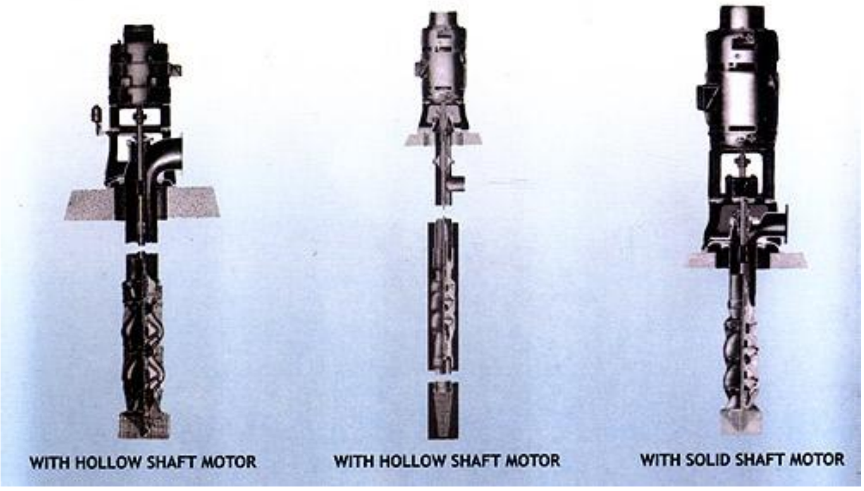
In selecting pumping units, functional efficiency is not measured so much by numerical values, but by economical performance, minimum service, and repair costs and long life.

Sump Pumping: For sump pumping, the horizontal centrifugal pump is most widely used. Advantages of it for sump pumping include:

i. They are better for handling water with abrasives.
ii. Maintenance is simplier as the entire pump is above the water
iii. Equivalent units are less expensive, especially in smaller sizes for low-head conditions.



Advantages of vertical turbine pumps for sump pumping (short coupled installation);



i. No priming is required as the pump suction is submerged. This is an advantage especially in automatic operations.
ii. This type of pump is very flexible. Additional stages can be added to increase head. For high-head applications to over 1000 ft, all stages are combined in one installation, thus simplifying the system.
iii. A minimum of space is required for foundations, and they fit well on pump floats.
iv. The motor and discharge head can be located well above water level to avoid danger of flooding during power failures.
v. Motor efficiency is better, and a slightly smaller motor can be used.

An advantage of submersible pumps for sump pumping is:

i. Simple connections make for rapid installation with a minimum of trouble for the operator.

Well Pumping: For well pumping, the vertical turbine pump is used almost exclusively.

Advantages:

i. Where pump settings in wells are deeper than 350 ft, the initial cost should be less because of the elimination of line shaft and bearings.
ii. A pump house is not required on the surface.
iii. More rapid installation with no periodic maintenance and lubrication is possible.

Disadvantages:

i. They are not recommended for corrosive waters or where abrasives are present.
ii. There is a danger of over heating the motor if installed in wells where surging takes place.
iii. They are more expensive for well setting down to 250 ft.
iv. They require factory service, and the electric power cable requires careful testing.

Airlifts are the least efficient means of pumping, but they have some characteristics that make them useful in special well pumping situations. Two pipes, one inside the other, are lowered into the well; compressed air is then forced down the smaller, inner pipe, and a column of mixed air and water rises up the other. This system has no moving parts, and so is able to handle water with abrasives and to operate in broken and crooked wells where pumps could not.

6. DRAINAGE

Good drainage is essential to all open pit operations. Water in the pit must be held to a minimum to achieve maximum efficiency. Drainage control in and around the pit may be likened to a preventive-maintenance program in machine operation. Neglected maintenance will contribute to breakdowns of equipment; likewise, neglected drainage control will curtail pit operations since flooding sometimes shuts down the entire pit. There is a tendency to overlook the importance of drainage control, especially during dry seasons when stripping conditions are most favorable.

Studies of the mining area, in order to select the best layout, would start with obtaining data from the nearest weather bureau on frequency of maximum rainfall. Armed with this information and available topographic maps showing the natural drainage pattern and position of coal, both above and below the high water mark of the main streams, a satisfactory system of drainage can be designed.

SURFACE DITCHES: The prevention of surface run-off into the pit should be the first consideration. The problem is not complicated when coal lies above the natural drainage. Once the water is collected in ditches, it is carried by gravity to either or both ends of the mining area and discharged into natural drainage. When coal lies below the natural drainage, the collection system is similar; however, levees must be constructed at the discharge points to prevent floodwater from entering the pit.

Conditions may dictate that the pit be worked across the direction of general flow of existing drainage. Main ditches are then dug in advance of, and parallel to, the high wall. They are dug with a dragline and/or tractor on a gradient towards the pit ends. Usually 0.10 or 0.25 % is sufficient to prevent silt build up. Their cross-sectional area will be determined by the depth of cut, maximum rainfall, and runoff factor of the topography. In crossing the flow line of ravines between hills, it is necessary to construct a bank of material downstream in order to assure containment during maximum flow.

Shallow ditches, following the contour around hills and discharging into the main ditches, are dug at regular intervals. These contour ditches are usually constructed 2 to 4 ft deep with a tilt-blade dozer or a road patrol grader. As the mining progresses these contour ditches are cut into first. Later, as the main ditches are approached, a new system is made in advance.

DIVERSION DAMS: Sometimes it is better to alter the mining plan by advancing the pit parallel to the direction of existing drainage flow. This is advantageous in that approximately one-half of the area need not have drainage control. That half of the valley sloping away from the pit will drain to its natural course. Before the flow line of the valley is reached, one or more diversion dams are constructed, together with main ditches on the opposite hillside, to carry the flow at a higher elevation during the time of pit advance across the valley bottom. The pit cut at the flow line of the valley should be made as wide as possible. This is done by loading out a double coal pit. The next overburden cut in filling the wide coal pit is laid with a low vee between spoils, creating a new watercourse through the spoils behind the pit. Next, a watercourse is opened at the diversion dam allowing the water to flow through the vee in the spoils (Fig.7.1). This can be accomplished as one operation or by approaching the valley at an angle and making the transfer in several steps. Services of an auxiliary machine, preferably a dragline, will be required during the transfer. Operations should be coordinates so that the stripping equipment can continue using its maximum digging time.

PIT DEWATERING: A small amount of runoff water always escapes the collection system and finds its way into the pit. In addition, seepage from the high wall and rainfall ill the spoil area collects in lows on the bottom of the pit. This water is collected in sumps by small, low-head gathering pumps or gravity ditches cut ill the under clay.

Pumping must be relied upon to dewater all pits below natural drainage and to some extent in pits above drainage. Economy requires sumps to be located near a discharge point that will give a low pumping head; a location near the haulage ramps into the pit is usually desirable as well as being readily accessible. Whenever possible, drain-ways should be held open through the spoils for the purpose of gravity flow or pumping discharge.

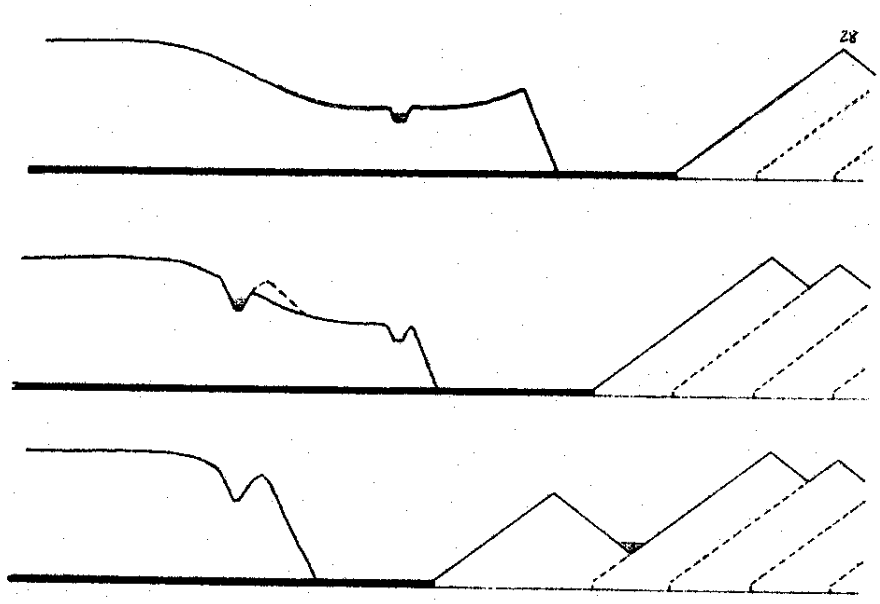


Fig.7.1. Diverting stream flow around working pit.

Another pit dewatering problem that the stripping industry has faced is encountered when approaching workings of abandoned underground mines. Some operations have unexpectedly cut into old works full of water with disastrous results. To prevent this from happening, a search in the county records for underground mille maps should be undertaken. Next, a thorough test drilling of the area should be made to check the location of the workings, as well as to reveal the amount of water they contain.

Stages of Reclamation

1. Planning the area usage
2. Reconstruction, Conditioning
3. Reclamation
4. Observation and Treatment

An Old Quarry (Estonia)

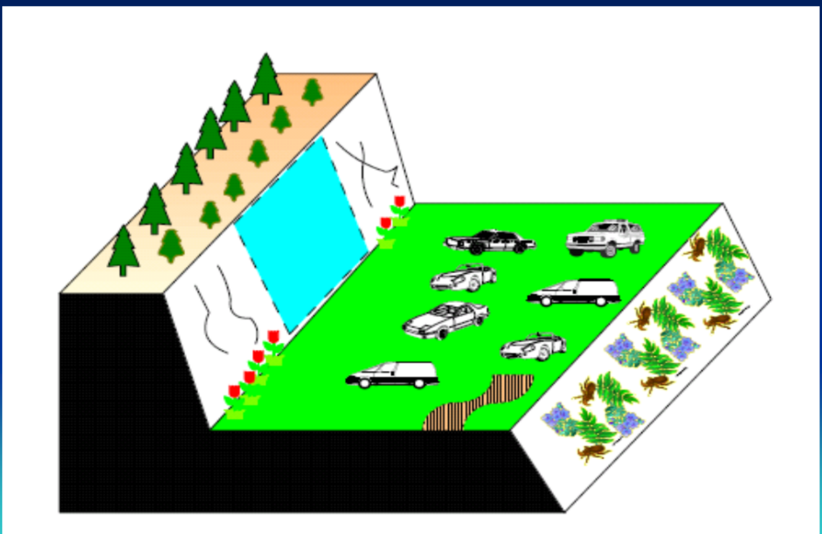
Garden Design (Germany)

Open Pit Coal Mine (USA)

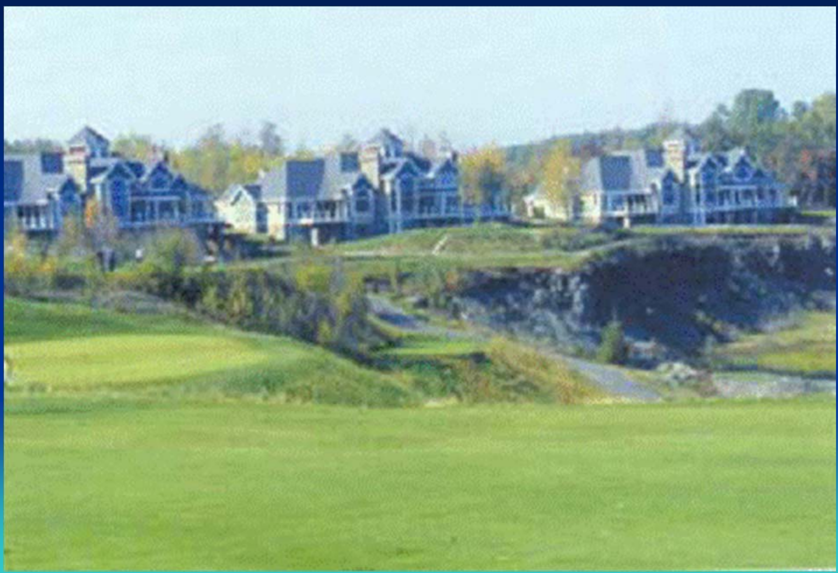
BEFORE

AFTER
(Amfitheatre)

Open Pit Slope, Drive-in Movie



Mountain Houses – (Michigan, USA)



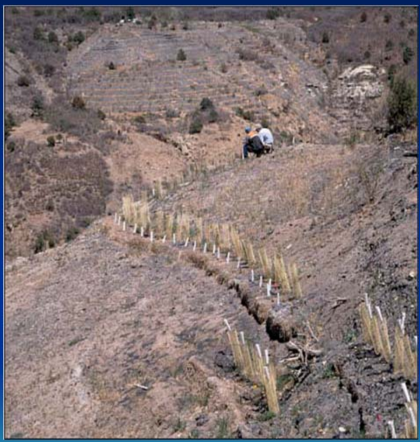
Recultivation (GLİ, Turkey)

Wild Life (Arizona)


Photo No. 1
Victory Mine Site
Main Area
Before Reclamation
Photo No. 6
Victory Mine Site
Post Reclamation
Start of the NCAA Cross Country
Championships
Reclaimed Victory Abandoned Mine Site
Photo No. 4
Borrow Pit Lake
Northern Area
Title V
Reclamation
Main Area with
Cross Country Course
and Other Facilities
World Champion-
ship
Hovercraft Course
Reclaimed Gob and
Landfill Area
Sports Center Office
and
Indoor facilities
Reclaimed Slurry
Pond


West Virginia

Coal Mine Reclamation (New Mexico)



Coal Mine, Revegetation (Indiana)

HOTEL Construction (PRC)

Çin'de inşaa edilen **otel** görenleri kendisine hayran bırakıyor. Projesi bile insanda heyecan uyandıran lüks otel, Tanrı Dağı eteklerinde bulunan ve 100 metre derinliği olan eski taş ocağı alanında kuruluyor. Etrafı büyük bir tema park tarafından çevrelenecek otel yer üstünde 3, yer altında da 16 kat olmak üzere 19 kattan meydana gelecek. 380 oda bulunacak mekanda ayrıca bir sualtı restoranı da olacak.