SWAMI VIVEKANANDA SCHOOL OF ENGINEERING & TECHNOLOGY

MADANPUR,BBSR

LECTURE NOTES

ON

SURFACE MINING TECHNOLOGY

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1.0 CHOICE OF OPENCAST MINING

1.1 State factors affecting choice of opencast mining method.

Opencast mining or quarrying of minerals is easier than mining by under ground methods. During quarrying the alluvium and rocks below which the minerals lie, are removed and dumped, in the initial stages, in a place, which is not required in future for quarrying resources or other purposes. The mineral exposed incompletely extracted. Opencast mining is also known as open pit mining, open cast mining, surface mining and also as strip mining. The later term being commonly used in the USA for opencast mining of coal. The overburden and the coal are excavated in long strips of a few meters thickness and hence the operations are termed strip mining. The operation of removing overburden and extracting mineral is done by one of the following method.

SURFACE MINING: Surface mining or open pit mining includes all the excavation made for profit of coal or mineral and all operation are exposed through out the atmosphere.

- ⇨ Surface mining carried out economical when coal seam or ore bodies are out cropping.
- ⇨ The coal seam and are body are at a quarriable depth, that is within the limit of stripping ratio.

Advantages of open cast mining:

1. There is no mineral blockage.
2. There is no need for roof control or ventilation.
3. Early extraction.
4. Quick return of capital.
5. There is less hazard and dangerous.
6. Large concentrated output is possible.
7. High efficiency of workers.
8. No need of artificial lights.
9. Quality control is easy.
10. Female labourer can work.
11. High degree of mechanization is possible.

Disadvantages of open cast mining:

1. The method may be uneconomic for extracting minerals at depth.
2. There may be environmental problem. (Noise pollution, ground vibration, water pollution, dust pollution may occur)
3. Degradation of surface land.
4. After reclamation and there may loss of original agricultural land.
5. Economic depth is less than 90 meter high capital investment.
6. Slope stability of the bench required particular orientation.
7. This method is unsuitable for rainy season, weather affect the working operation.
8. Excessive re-handling of the over burden during open cast mining.

Classification of opencast mining: These are two types.

1. Mechanical excavation.

(a) Open pit mining: In open pit mining the O/B is strip and transport to a disposal area to uncover the coal or mineral deposits.
(b) Opencast mining: In opencast mining the O/B casted or hauled directly in to the adjacent mine of the area.
(c) Quarrying: Quarrying means open cast mining of dimensional store, like granite, metal, stone.

(d) Auger mining: It is the method for surface out crope the excavation is conduct boring opening using auger in to the coal seam bench and the coal seam extraction without removed of O/B.

2. Aqueous mining: It include those method which play on liquid solution for recovery mineral from earth, either by hydraulic action or solution attack.

1.2 Define stripping ratio.

It is generally expressed in the proportion of OB to coal / mineral removed. As the depth of over burden increases more amount of money is spent for removal of the over burden for exposing the mineral body / coal and a time will come when mineral / coal cannot be economically extracted out is called the break even stripping ratio. Factors which play a great role in the stripping ratio calculation are cost of stripping ratio, extracting cost mineral / coal percentage of resects, cleaning cost of coal or ore dressing cost, sale value of clean coal / dressed ore, reclamation cost, coat of transportation, overhead and sale of ore / coal etc. The variation in stripping ratio affect the choice of the mining equipment and requirement of operational efficiency overburden although ultimately it depends upon the economic criteria of the mine. Before going for many surface mining operation or economic comparison is necessary between the underground mining cost per tonne of coal / ore production and surface mining and reclamation cost per tonne of coal / ore production and a stripping ratio is to be established.

$$\text{STRIPPING RATIO} = \frac{\text{WEIGHT OF RECOVERABLE MINERAL RESERVE IN TONNE}}{\text{VOLUME OF OVERBURDEN IN CUBIC METER.}}$$

1.3 Determine overburden / ore ratio.

The soft material like earth and weathered rock is cut by earth cutting picks. A term of workers consists of work member, ore cutting and loading. The removal of OB is a very important operation in surface mining system. The method of excavation of OB depends upon the following main factor. Thickness deep and depth of the overburden rock mass, manner of occurrence of the deposits, i.e. whether the deposits are occurred under the flat surface terrain or over the hilly terrain, the surface topography, the ground condition, the environmental condition, production requirement, geotechnical parameter of rocks like the compressive strength.

1.5 Determine Quarriable limit.

The cost of removing overburden to extract mineral lying below it goes up as the quarrying operations extend to the dip side of the property and the thickness of overburden increase. The stripping ratio, thickness of overburden; thickness of mineral deposit therefore decides the economic working limit of quarrying, i.e. the quarriable limit. The softer the rocks, the less is the expense of overburden removal and higher is the stripping ratio. The wages of labour, the selling price of mineral and the margin of profit and the major considerations in deciding the limiting ratio which is as follows in coal mines :

i.	Manual quarrying	1.5 : 1
ii.	Semi-mechanised quarrying	2 : 1
iii.	Mechanical quarrying :	
	With dipper-shovel, dumper combination :	4 to 5 : 1
	With draglines:	8 to 10 : 1
	With bucket wheel excavators:	3 to 4 : 1

The maximum depth from the surface in existing mines in our country is 120 m but future mines are planned to reach a depth of nearly 480 m.

2.0 Benching:

2.1 Determine bench parameters- height, width & slope.

<u>BENCH:</u> A bench may be defined as a bedge (horizontal space). That form a single level of operation above which mineral or waste material are mixed (excavated). That to the bench face.

It is the part of O/B or coal seam in open cast mining on which drilling, blasting, loading and transportations are carried out.

<u>BENCH HEIGHT:</u> It is the vertical distance between the highest point of the bench or bench crest to the toe of the bench. Bench height is normally Govern by the specification of operating machinery (M/C) such as drilling machinery shovel and dragline.

Note. => The bench height is always less than the bench width.

=> It depends upon the thickness of deposit equipment available nature of rock forming bench.

<u>Geological disturbance:</u>

- ⇨ In abrasion soil bench height should not exceed 1.5m.
- ⇨ In metal ferrous mining bench height should not exceed 6m.
- ⇨ In coal bench height should not exceed 3m.

<u>WIDTH OF BENCH:</u> It is the horizontal between crest of the bench to toe of the bench.

- ⇨ It depends on height of bench pattern of blasting transport system.

<u>SLOPE ANGLE:</u> It is the angle measured in degree between the horizontal is the imaginary line joining toe of the bench between two bench.

(i) It is either equal to or less than angle of repose of bench rock and it depends on plan of weakness, mechanical property of rock, orientation of bedding plane.
(ii) In alluvium soil should not exceed 480°C.
(iii) In metal ferrous should not exceed 60°C.
(iv) In hard and competence rock 750°C.

<u>ANGLE OF SLOPE:</u> It is the maximum slope of which hip loose material will stand without sliding.

<u>THE OVERAL PIT SLOPE:</u> It is the angle at which the wall of an open pit stand and measured between horizontal plane and the plane imaginary plane joining the periphery of the open pit or toe of the bench.

<u>BERM:</u> It is the Horizontal shafe or ledge with in the ultimate pit slope.

<u>ULTIMATE PIT SLOPE:</u> The angle at which a plane joining the crest of the open pit with the toe of the bottm most bench make with the horizontal.

<u>CREST OF THE BENCH:</u> It is the line formed due to intersection bench face or bench slope and with the bench surface.

<u>SLOPE OF BENCH:</u> It is the angle between horizontal plane with the imaginary line joining toe at the bench crest of the bench.

<u>TOE OF THE BENCH:</u> It is the line formed due to intersection of bench slope with bench floor.

3.0 <u>Application of different machineries in opencast mining:</u>

3.1 <u>Calculate bucket capacities of Power shovel, Backhoe, Dragline, Bucket wheel excavator, Bucket chain excavator, wheeled scraper, Bulldozer, Road grader and Ripper.</u>

<u>POWER SHOVEL:</u> A shovel is a equipment which excavates the rock or ore by digging from its operating base to upward.

<u>CLASSIFICATION OF SHOVEL:</u> Shovels are classified considering the following factors-working agent, bucket size and working mechanism. According to working agent they are regarded as:

(a) Diesel shovel (b) Electric shovel (c) Hydraulic shovel

According to bucket size they are:

(a) Commercial type power shovel, bucket capacities in the range 0.275 cu m to 1.8 cu m.
(b) Quarry mine type shovel, bucket size 1.8 cu m to 15 cu m or more.
(c) Large quarry mine type shovel, bucket size 15.0 cu m to 75 cu m or more.

According to working mechanism they are:

(a) Rope shovel (b) Non-rope shovel.

<u>DIESEL SHOVEL:-</u> This is one of the oldest types of shovels used in open cast projects. Various designs are available nowadays. Which consists of the following units:

i) Prime mover, (diesel engine)
ii) Power transmission system including crowd and hoist mechanism, travel mechanism, swing mechanism
iii) Undercarriage unit.
iv) Front attachment.

Undercarriage unit is also known as the crawler travel unit. This unit consists of two shovel track assemblies fitted with a driving sprocket, an idle sprocket, track rollers and support rollers. The whole of the track assembly is mounted on a track frame. There is a provision for adjusting the tension on the track near the idler of the assembly.

Front attachment consists of a boom, a dipper stick and a bucket. The bucket is rigidly attached to the front end of the dipper stick, while the dipper stick and bucket combination is held by rope at one end and the stick is supported at the other end by the crowd pinion. There are two such crowd pinions, which are mounted on the shipper shaft or the crowd shaft. The crowd shaft is powered by a chain sprocket arrangement receiving power in turn from another chain sprocket arrangement.

<u>ELECTRIC SHOVEL:-</u> The shovel under this category are used widely in Indian mines and are manufactured by Heavy Engineering Corporation. Ranchi. The most popular model of them is the excavator EQC-4.6. This consists of the following mechanisms:

i) Motor generator sets, ii) Crowd mechanism, iii) Hoist mechanism,
iv) Swing mechanism, v) Travel mechanism, vi) Pneumatic control system,
vi) Undercarriage unit.

The crowd mechanism consists of a crowd motor, which delivers power through the pinion to the overload clutch. The overload clutch delivers power to a pinion which is in consists mesh with the gear. This gear drives two crowd pinions on both its sides which finally reciprocate the dipper stick with the help of rack rigidly fixed in the underside of the dipper stick.

Hoisting of the boom and bucket of the shovel is performed by mechanism. One end of the motor shaft is connected to the reducer through a coupling. The reducer delivers power in constant mesh with the hoistgear. This provides movement to the hoist drum, which lifts the bucket.

A swing mechanism consisting of two units has a vertical motor with its flange resting on the housing. At the end of the vertical motor shaft, there is a pinion in mesh with gear. There is a common shaft in the reducer over which both the pinion and the gear are mounted.

The travel mechanism is driven by a motor cylindrical-cum-bevel reducer. The reducer is connected to the motor by flexible coupling. The bevel pinion receives power from the motor through the coupling and the bevel gear is in mesh with the pinion. Ultimately power from the reducer through a gearing arrangement is given to the right hand side and left hand side dog clutch.

Pneumatic control system consists of a twin cylinder reciprocating compressor driven by an AC motor and delivers air to a tank through air check valve. The tank is fitted with a pressure guage and a safety valve. Air from the tank is distributed to the swing mechanism brake cylinder, which are controlled through three separated solenoid valves and there is one sire operated by air through another solenoid valve. The remaining end is connected to a T-valve for any other auxiliary use.

Undercarriage unit perform the same function as those of a diesel shovel. The constructions are also similar to that of a diesel shovel. The only changes are the shape of the driving sprocket of the crawler unit. Instead of driving sprocket it has cam wheels which are in constant connection with the cat belt and the endless chain is supported by a number of wheels mounted on the main frame.

<u>HYDRAULIC SHOVEL:-</u> The implementation of this concept started as early as 1950. As many as 36 versions are available throughout the world now a days. The undercarriage units are very similar to those of other shovels. But the superstructure and front attachment are different from those of other shovels. The equipment consists of the following

i) Prime mover ii) Superstructure and its attachments
iii) Hydraulic mechanism iv) Undercarriage unit

Both diesel engines and electric motors are used as prime movers. In regards the electric motors, AC induction motors are usually preferred in the present case.

Like all other shovels the superstructure is mounted on a turntable. This consists of space for the prime mover, the pymp, the hydraulic tank, hydraulic circuits, etc.

The front attachment of the bucket, the stick and the bom. The whole assembly is mounted on pin joints at the front part of the equipment. The boom moves in a vertical plane and is operated by two boom piston cylinders. The stick, on the other hand, is mounted on the top of the boom by a pin joint, which is also capable of moving in a vertical plane by one stick piston-cylinder arrangement.

Undercarriage unit is very similar to that of the diesel shovel undercarriage unit having a track and carrier roller attached to the crawler frame. The only different is the shape of the track shoe. Hardened grousers and provided at the exposed side of the shoe so that a more tractive effort can be obtained during operation instead of the flat surface to the shoe as in diesel and electric shovels.

SELECTION OF SHOVEL:- Selection of shovel in done by following factors:

1. Requirement of daily production.
2. Type and quality of material.
3. Total mineral reserve, compressive strength, share strength, abrasiveness, dip of the mineral body.
4. Bucket fill factor.
5. Swell factor.
6. Working cycle time (cycle = digging time + dumping time + return time for next digging)
7. Weight and maximum long size of the material.
8. Height of the bench and height of the cut.
9. Weather the shovel is workout in overburden or mineral.
10. Weather it is used for production or overburden or reclamation.
11. No. of faces to be worked.
12. Nature of terrain.
13. Capacity of haulage equipment.
14. Operating parameters, like digging height, digging radius, dumping height, dumping radius.
15. Bucket size and capacity.
16. Overall dimension. Rear and clearance angle of boom inclination, height of excavator.
17. Availability of electric power.
18. Drainage condition.
19. Working gradient.
20. Equipment availability.
21. Climating condition.
22. Reliability of the machine.
23. Capital cost of equipment.
24. Operating and maintenance cost.
25. Availability of spare parts.
26. Facilities of after sale service.

COMPARISON BETWEEN ROPE SHOVEL AND HYDRAULIC SHOVEL:

Rope Shovel	Hydraulic Shovel
1. Higher initial cost, longer life, lower depreciation rate.	1. Lower initial cost, shorter working life.
2. Usually have a good resale value.	2. Resale value is usually poor.
3. Heavier basic machine weight.	3. Lower basic machine weight.
4. Very high capacity shovel are available.	4. Less capacity shovel available.
5. Heavier weight cause for wider heavier track pad to reduce ground pressure.	5. Light weight makes narrow pad.
6. Very good performance.	6. Poor performance.
7. Reliability under heavy duty condition.	7. Reliability under heavy duty condition is less.
8. In pollution free atmosphere and high attitude driven by electric drivers gives very good performance but efficiency is less.	8. In pollution free atmosphere and high attitude drivers by electric drive gives best performance.
9. Penetration in the ground in achieved combination of pushing force of the	9. Penetration into the ground is achieved by the combination of three

dipper arm and lifting the help of the boom.	forces, bucket crowding, dipper, arm swinging.
10. It cannot be used as back hoe.	10. It can be used a bucket.
11. Utilization of power is poor to moderate.	11. Utilization of power is highly efficient,
12. Operation is jerky and cycle time is more.	12. Operation is smooth and cycle time is less.
13. Steering and movement is difficult.	13. Steering and movement is less difficult.
14. Level floor clean up is limited.	14. Level crowd permits better floor clean up.

<u>CAPACITY OF SHOVEL:</u>

Shovel Capacity is given by the following expression.

$$Q = \frac{B \times f \times u}{t \times Sf} \ m^3/s$$

Where B = Bucket capacity in cu m.
f = Bucket fill factor.
u = Shovel utilization factor.
t = Shovel cycle time in seconds.
Sf = Swell factor or Rock loosening factor

So shovel capacity on hourly basis is given by

$$Qh = \frac{3600 \times B \times f \times u}{t \times Sf}$$

There is another capacity i.e. technical capacity of shovel, where technical factor is taken into consideration is given by

$$Qt = \frac{3600 \times B \times f \times u \times k_t}{t \times Sf}$$

Values of fill factor "f", swell factor or rock loosening factor "Sf" and technical factor "k_t"

<u>BUCKET CAPACITY OF SHOVEL:-</u>

- Spotting time dumper
- Loading time = No. of pass required by shovel to fill the dumper X Cycle time by shovel.
- No. of pass = $\frac{\text{tonnage reacting of the truck}}{\text{Tonnage per pass by shovel}}$

Tonnage per pass by the shovel= Capacity of bucket X Bucket factor X Bank density

<u>BACKHOE / PULL SHOVEL / DRAG SHOVEL:</u>

This is known as a hoe, back hoe, drag shovel, pull shovel. It is used for loading dumpers and its best application is for digging below the level on which it stands. The shovel and the dumpers can stand as a higher level free from water and mud of the quarry floor. As the attachments to the bucket are by dipper stick and not by cables, the bucket is under positive control of the operator and therefore suitable for hand digging.

The shovel is used for stripping top soil, and making shallow cuts and trenches up to a depth of 3.5 to 6m. Compared to dipper shovel, the hoe is slower in digging and less efficient for loading trunks.

In deep digging, the face should be kept fairly straight and the shovel should be as far back from the edge as possible, otherwise there may be danger of caving of the edge.

⇨ It digs soil, rock or ore below the level of the bench on which it stands and unload excavated material over a truck or railway wagon.

⇨ It can be deployed efficiently where the mine is very much wetted by the prolonged rain or where seepage of water through the ground strata is very heavy.

⇨ Generally the backhoe is very good for trenching, shallow depth cutting and for basement excavations.

⇨ The hoe bucket is attached to a dipper stick at its lower and facing towards the machine.

⇨ The middle of the dipper stick is hinged to a boom where as the top of the same is attached with a rope puller.

⇨ A drag rope is connected to a drag drum passing through the sheave attached to the side of the main boom.

⇨ The dipper is lowered down into position by the hoist rope so that bucket bites into ground and the cutting operation is achieved by the pulling the drag rope ties it becomes fully.

⇨ Presently hydraulic backhoe are also available in market.

⇨ It is efficiency is around 60 to 70% of that of dipper shovel of identical size.

⇨ Although it has longer cycle time and less efficiency is discharging materials over the trucks but it can nicely be used for removing top soil or overburden.

DRAGLINE:

This is also intermittent discharge-type excavator. The machine is well known for having the longest boom among excavator. The boom length varies from 9 to 96 m. Unlike other excavators, the bucket is not rigidly held by the frame structure. Filling of the bucket is done by pulling or dragging it against the material towards the machine because of which the machine is named a Dragline. One end of which is attached with the revolving unit of the machine and the hanging end in other sides carries a large sheave for the cable attached with the bucket. It is made of lattice construction by the structural steel which is lowered down or raised up by the cable of boom.

SELECTION OF DRAGLINE: The factor to be consider during selection of dragline.

1. It is general used for handling softer unconsolidated material.
2. It has a greater ability to dig well above and below grade.
3. It has a greater digging reach and dumping radius.
4. Since dragline boom length can be varied, its working range also can be varied.
5. May or may not require other equipment for disposal.
6. Although it is less efficient compare to shovel it can function well under less a rigid operating condition.
7. Since there located on the top of the bench surface bench slides, water seepage small undulation of surface does not hamper the effective operation of dragline.
8. During opening of opencast mine dragline is the most suitable excavator.
9. In the soft mineral deposit the dragline can operate efficiently than a shovel.
10. They can negotiate a gradient up to 12°.
11. In the presence of a hopper reloader dragline can load material into railway wagon, belt conveyor and other transport facilities.
12. Superior in wet pits.
13. Maintenance is cheap.

DEMERITS OF DRAGLINE:

1. If the blasted rocks are large lump the bucket field insufficiently and the bucket and dragrope wear rapidly.
2. Used for softer rock formation.
3. Production cost is more compare to shovel.
4. It is less efficient compare to power shovel.
5. Bucket fill factor is less.
6. Lesser spotting ability.
7. Less output than the powered shovel.

COMPARISON OF DRAGLINE AND SHOVEL:

1. Fill factor : Dragline bucket fill factor is less than that of shovel by about 12%
2. Cycle time : Dragline cycle time is more by about 10%.
3. Application area
   - i) multi-scam project : A shovel can do the job easily.
   - ii) bearing pressure : Operating bearing pressure on the ground is less in case of dragline. This is even less than what a man exerts on the ground. It is about one atmosphere while a shovel exerts about 3.5 atmospheres.
   - iii) Floor condition : Dragline requires a nearly horizontal floor. This is not the case with a shovel.
   - iv) Reach : Reach of a dragline is much more than that of a shovel because of its longer boom.
   - v) Digging below level : A dragline can cut much more below the floor level than is usually cut by shovel.
4. Maintenance aspects
   - i) Cost : Maintenance cost is less by about 18% in a dragline.
   - ii) Frequency of failure : Frequency of failure is more in the case of a shovel than a dragline.
5. Output : Output of a shovel is 20% more than a dragline.
6. Production cost : Unit cost of production that is Rs/m3 (bank measure) is less by about 10% in a shovel than a dragline.

DRAGLINE CAPACITY:

Hourly capacity of dragline is given by

$$Q = 60B\frac{f}{S_f}n\,u \text{ cu m/hour}$$

Where B = Bucket capacity in cu m.
f = Bucket fill factor
S_f = Swell factor.
n = No. of cycles per minute.
u = utilization factor.

There is another capacity i.e. technical capacity of dragline, where technical factor is taken into consideration is given by –

$$Q_t = \frac{60\ Bf \times u \times k_t}{S_f} \text{ Cu m / hr}$$

Where fill factor "f", swell factor or rock loosening factor "S_f" and technical factor "k_t".

BUCKET WHEEL EXCAVATOR:

Bucket wheel excavator digs and discharges continuously from bank to dumping point. Since series of bucket are attached to the periphery of a wheel which is rotated during cutting action hence it is known as bucket wheel excavator.

The machine can be classified into two types:

i. Rail-mounted bucket wheel excavator.
ii. Crawler-mounted bucket wheel excavator.

The crawler-mounted bucket wheel excavators are mostly used in mining projects as against rail-mounted bucket wheel excavators that are rarely used. A similar equipment known as the bucket-wheel reclaimer, which is mostly rail mounted, finds its application in material handling projects, ports and processing plants.

MAIN UNIT / COMPONENT OF BUCKET WHEEL EXCAVATOR:

The equipment consists of the following main components/system.

1) Bucket wheel,
2) Wheel boom and conveyor,
3) Transfer conveyor and boom,
4) Counter-weight and equalizer boom,
5) Se\wing or slewing system,
6) Luffing arrangement,
7) Travel mechanism,
8) Undercarriage unit,
9) Lubrication system.

SUITABILITY CONDITION:

1. Bucket wheel excavator which is used for long range stripping of soft over burden rock use for hard and tough, well fragment blasted rock
2. With no unless bulldozer.
3. It is suitable for uniform ground and bank condition
4. It is used for thin seam and selective mining.
5. It is used for land reclamation.
6. Excavator radius 40-90m.
7.

MACHINE AND ITS OPERATIONS:

1. Bucket wheel excavator has a wheel (2.5-17) m dia attached at the one end of the boom.
2. A wheel containing (6-18) m evenly spaced bucket attached around the periphery of the wheel.
3. Bucket capacity ranges from $0.4(m)^3 – 6.3(m)^3$
4. And it cuts bottom to top way manner.
5. The bucket are cellness semi cell type.

CAPACITY OF BUCKET WHEEL EXCAVATOR:

Let I = Inner volume of single bucket in liter
S = Number of buckets discharged / min
S_f = Swell factor
Z = Number of bucket in a wheel
N = rpm

Theoretical Capacity

$Qth = I \times 10^{-3} \times S \times 60$ cu m/h (loose measurea0

$$Qth = \frac{6\ IS \times 10^{-2}}{S_f} \text{ cu m/hours (solid or bank measure)}$$

No. of bucket discharge per minute

$S = N \times Z$

$$Qth = \frac{6l\,NZ}{S_f} \times 10^{-2} \text{ cu m flow (bank measure)}$$

Again Let

V_s = Slewing speed m/minute

h = cutting height in m

d = Depth of cut in m (average)

Qth = 60 h d V_s cu m/hour (bank measure) = 60 h d V_s X S_f cu m/hours (loose measure)

Annual production

Q_A = *service factor* X Qth X hours run per year cum / year Service factor depends on working condition and operational efficiency.

WHEELED SCRAPER:

This machine is diesel-operated with pneumatic tyred wheels and has at he centre a bowl fitted with a cutting blade at bottom. The blade is reversible and can be replaced when blunt. Its working may be compared to that of a lawn power. As a scraper is pushed forward by a dozer, its blade cuts a thin slice of earth usually between 75 mm and 225 mm thick over a distance of nearly 30 m. the earth is automatically collected in a central bowl shoes capacity ranges from 3 m^3 to 22 m^3 and it takes nearly one minute for loading. When the scraper is fully loaded its bottom opening is closed by the operator through manipulation of a cable (rope) and the loaded scraper, with the bowl lifted, travels to the dumping yard on its own power. At the dumping yard, as the scraper moves, the bottom opening of the bowl is opened and the contents are unloaded in a layer 150 mm to 250 mm thick, over a distance of 30 to 70 m. The bowl is always bottom discharging. Scrapers are unsuitable in soils with stumps, large boulders and hard rocks. When the ground is hard, it is necessary to rip the surface with the help of a ripper before loading by a scraper. Sandy soil is best for a scraper which has to be stopped during rains, if engaged in aluminum.

Scrapers are used in coal mines for cutting and transporting weathered sandstone as well as coal. The coal excavated by it is however smaller in size. A scraper may take 5 to 6 minutes for a complete cycle of loading and unloading if the total up and down distance of a trip is nearly 300 m. One way traffic of loaded and empty scrapers is desirable for good results. One dozer is normally sufficient for every two scrapers used.

The scraper manufactured by BEML has the following main specifications:

Flywheel H.P. of engine 332 at 2100 rpm;

Capacity : payload 23000 kg; struck 11.5 m3, heaped 16 m

Max. travel speed (forward 44 km/hr.

Overall dimensions mm : length 12600 ; width 3470 ; height 3890.

Net weight (no load) 26584 kg.

CLASSIFICATION:

This equipment can be classified in the following manner:

1. Self propelled scraper: The bowl hold the dirt. The bowl is hanged and tips forward to roll the material out. In others a tailgate, which is a wall in the rear bowl, is used to push the material out. The bowl is held in position by two lift cylinders. The cylinders are mounted on a frame.

   The apron may be straight or curved wall in the front of the bowl, which opens and closes to regulate the flow of earth in and out if the bowl. The movement of the apron is also controlled by a separate piston cylinder arrangement.

The cutting edge is lowered into the dirt to make a shallow cut.

2. Towed scraper: The scraper unit of this type is usually mounted on four wheels. These wheels carry only the load of the machine and materials. They are neither steered nor driven. This is attached at the rear side of the tractor. The individual motion os the scraper body are performed by two drum powered control units mounted on the back of the tractor. These consists of a pair of high speed winches driven by the power take off shaft and controlled separately by friction clutches and brakes.

DIFFERENT COMPONENT OF SCRAPER:

1. Bowl, apron, cutting edge.
2. Transmission system.
3. Steering system.
4. Braking system.
5. Control unit (cable or hydraulic)

BULLDOZER: A bulldozer is often referred to simply as a dozer. It is a tractor with a pusher blades attached to the front portion. The tractor is the diesel operated power unit equipped with either crawler chains or rubber tyred wheels for lifting. The pusher blade can be raised or lowered or tilted through small angles horizontally by rams operated through hydraulic pressure or by ropes. The dozer blade is used for pushing loose material of for digging in earth, sand and soft weathered rock. The machine is also engaged for leveling or spreading earth, for leveling of rock spoil in the dumping yard, grading and compacting temporary roads, pushing mineral into sub-ground level bunkers through grizzly, for towing dumpers, etc. It also serves the purpose of pushing boulders, pulling down trees, and is an essential equipment to push scrapers. A dozer equipped with a fork like attachment is known as ripper and operated like a plough to loosen moderately hard rock. The loosened rock may be loaded by a scraper. A dozer can dig 1.2 m to 1.5 m below ground in earth or weathered rock.

CLASSIFICATION OF DOZER: There are six major types of dozers:

1. Straight or bulldozers.
2. Angle dozers.
3. Tilt dozers.
4. Push dozers.
5. U-shaped dozers.
6. Brush or rock rakes.

COMPONENT OF DOZER: A dozer consists of the following component- arms, blade, undercarriage unit, transmission system and blade operation system.

DOZER BLADE CAPACITY:

The major dimensions of a dozer blade

h = height of blade

L = length of blade

Therefore theoretical load = $1/2h^2 L^¥$

Where $^¥$ = specific weight of the material.

On level or horizontal ground 20 per cent centre bulging was found.

Hence:

$$\text{Load} = 1.2 \frac{h^2 L}{\quad}{}^{¥} = 0.6\, h^2 L^{¥}$$

ROAD GRADER AND RIPPER:

Road Grader: This is a machine for leveling the road surface by smoothening out the ups and downs and for casting aside the boulders on the road. It is always pneumatic tyre mounted with only rearwheel drive and the front wheels are small. The grading blade is attached to a circle that is hung from the overhead frame and pulled by a drawbar fastened to the front of the frame. The blade is usually 3.5 to 4 long having replaceable edges on the sides and bottom. Steering is direct-connecting mechanical by a hand wheel though a hydraulic booster id fitted on some models.

CLASSIFICATION: Grader can be classified in the following manner :
(i) Self-power grader (ii) Towed grader.
Again, self powered graders are of two types :
(i) Mechanically controlled machine. (ii) hydraulically adided machine.

COMPONENT: The following are the components of a grader.
(i) Blade and circle: The blade is curved with a cutting edge at the bottom and has two corner pieces at the side. The width of the blade is less in comparison with that of a dozer blade. Circle is a machined ring with gear teeth on the inner surface.
(ii) Transmission: Power form the engine is transmitted through a wet multiple disc clutch and coupling to the gear box. Power form the gear box is taken in a differential through universal joints and a propeller shaft. The differential distributed power on the left and right hand side as the situation demands to the tandem drive sprockets.
(iii) Steering system: when the steering wheel is rotated, the shaft of the steering system is rotated either clockwise or counter – clockwise and the motion is transmitted through the gear box to the vertical shaft. An arm, mounted on the vertical shaft, operates a ball joint. The ball joint in turn helps in steering the machine by changing the direction of the wheel through two tie rods.
(iv) Blade control unit: The movement of the blade is of three types: (i) up an down, (ii) sideways, and (iii) rotational.

The above movement is done by operating a lever which transfers the effort through a control panel, universal joint, primary and secondary gear boxes to the arm or crank. The arm is thus rotated either clockwise or counter-clockwise, as a result of which the blade experience lateral and vertical shifting movements. The effort of the lever passes through a control panel, universal joint, primary gear box, universal joint and secondary gear box to the pinion. The pinion is in constant mese with the inner teeth of the circle. The rotation of the pinion provides rotational motion of the circle, which in turn rotates the blade.
(v) Brake system: Two types of brake are used. They are

(a) The parking brake: This is a mechanical brake applied on the output shaft of the transmission or gear box. It is an internal expanding type.

(b) The service brake: This is a hydraulically actuated brake applied on all the wheels. It is an internal expanding type. In the servo system compressed are is allowed to flow and push the piston of the master cylinder, which applies the brake through hydraulic fluid pressure.

FUNCTION OF GRADER: The following are the fuctions of a grader,
i) Bulldozing, ii) Side casting, iii) Planing, iv)Crowning Bulldozing.

Byldozing: A grader may be used as a bulldozer to a limited extent for spreading piles of loose material. This can be achived in many ways, either by extending the blade sideways or by driving the front wheel over the piles.
Side casting: The blade is set at an angle such that the material being pushed in front of it is thrown off to one side. When the blade is angled more sharply the throwing off of the material is done at a higher speed. The thrust acting on the blade due to this is usually compensated by steering the front wheel.

Planing: The blade is also used to plan off irregular surfaces. This is done by lowering the blade sufficiently below the original level of the material and the material is cut and stored in proper places, giving a neat surface finish.
Crowning: To give a surface finish to a haul road it is usually crowned so that water will flow off to the sides. The road material is bladed inward form the sides and the top of the crown is cut by adjusting the blade accordingly.

RIPPER: The main function of ripper is to loosen the ground. This is necessary for stiff clay, semi-consolidated sand and gravel, weathered rocks ground, etc, which has to by scrapers and/or dozers.
CLASSIFICATION: Rippers can be classified as two types:

i) Tractor mounted type, ii) Tractor pulled type.

The tractor pulled type may again be sub-divided into the following two types:

i) Cable control unit and ii) hydraulic control unit.

Tractor mounted type: This type of ripper is usually mounted on the rear end of the tractor as an extra attachment. The movement of the ripper body is controlled by two hydraulic by two hydraulic rams which are controlled form the tractor. These rippers are simple in construction and are used for medium to hard soils.

Similarly mountings are also provided at the rear and at front ends of a dozer. The sizes of the ripper shank and tooth are larger in dimension compared with others. They are used for heavy duty work.
Shanks: The shank's teeth in a ripper are attached at the bottom reverse of the dozer blade. Ripping action takes place when the dozer is moving in a reverse direction. This arrangement helps in executing ripping and dozing action by the just by revesing the motion. Sometimes three or four heavy shanks are fixed on to a frame, which is mounted on the rear end of a dozer.
Cable control unit: This comprises a frame and a draw tongue. The frame has provision for three or more shanks which are held in the sockets by single pints. The shanks carry detachable teeth at the bottom end. On either side of the draw tongue are two semi-circular bell cranks hinged to the frame. The top ends of the bell cranks carry a hoist pulley. When the hoist line is pulled form the tractor, the top ends of the bell cranks move forward, thus lifting the shank. Down pressure is given by the weight of the rippers frame which may be as heavy as 6 tones.
Hydraulic control: The ripper attachment is placed at the rear of the tractor as in the cable control unit. The control of teeth, that is, lowering and raising, is performed by a hydraulic ram, which in turn is controlled form the operator's seat.
USES OF RIPPER: Rippers are used for the following purposes,

i) Inspect teeth profiles and sharsen them, if necessary.
ii) Inspect the condition of the cable, pulleys, etc, for activating the shanks.
iii) Operate the hydraulic control unit to check its smooth functioning.

4.0 DRILLING: The process of drilling is done for the following purposes,
1. To know about the ides about the rock, nature, characteristic etc. Also to prove the existence of the mineral.
2. To get the core of the hole and from which we can get the gearing pressure of the ground on the mineral which is helpful in set up of weighted engine foundation.
3. To carry out electric cables, water pipes, signal wire from surface to underground.
4. To have tube wells for supply of water to colonies.
5. For ventilation purpose in U/G mines.
6. To extract gas & water from old working.

4.1 Explain different methods of exploratory drilling.

The methods are:

1. Percussive Drilling.
(a) Drilling by rigid rods. (b) Drilling by cable, rope (Churn drilling)
2. Rotary Drilling.
(a) By saw toothed cutter. (b) By tricone rock roller bit. (c) By diamond drill bit.
(d) By chilled shot.

1. Percussive Drilling: The oldest type of drilling in which the or shot at small by striking or giving a thrust or shot at small interval of time and is slightly rotated. Thus the rock is chipped away. During drilling the chisel like tool is suspended over the ground by a rods or wire or ropes. The weight of the chisel like tool is used for giving striking force. Usually 38mm X 38mm of square cross sectional of 3m length steel rods are used. The rods used are generally Ni-Cr or carbon steel
(a) Drilling with rods: In this type of arrangement a power operated which is used to raise or lower the rods. The walking which is used to raise or lower the rods. The walking bean is operated by a crank which is geared through a engine, used to give 25 to 30 blows per minute and 225mm dia. strive. The walking beam is mounted on a steel spring which cause sudden recoil of drilling bit and thus prevent the jamming. The rods are attached to the ropes with a swivel and brace head attachment. The walking beam is connected by a rope which is slackened from time to time to make the rod in contact with the rock.

Water is supplied with hose pipe through the hollow rods. These waters ames out from the hole along with sludge's.
(b) Drilling by cable, rope (Churn drilling): In this method rods are replaced by the steel ropes of 18mm dia and about 300 metres length. The surface arrangement is same as compared to drilling by rope. In addition to drilling with rods, the walking beam is attached at the end of the temperature screw. The drilling cables are carried from through the clamps of the temper screw across the pulleys of the derrick.

A feed of up to 1.2m is fixed to the temper screw. When no more feed is possible, then the temper screw is run back and a additional 1.2m high feed is used.

There is no arrangement of device for the rotation of the steel pipe between successive blows as the lay of the steel pipe between successive blows as the lay of the steel pipe causes the tool to twist. The rope always have a lefthand lays which not only tend to twist the tope but also tend to tightened the joints between them.

2. Rotary Drilling: In rotary drilling hollow rods of aluminum and steel is used which are thread connected. At the end of the columns of drilling rod the drill bit is used. These rods also transmit the torque or pressure energy rotation of drill bit of gearing is driven by the prime mover at the earth surface. As the rod rotates the drill bits abrade the rock and is cleared up by the supply of pressured water or compressed air comes out from the out side of the rod through the clearance.

The aluminum rods weights halt as that of steel rod but offers 90% mechanical efficiency. The parts which are mostly exposed to wear is made of Ni-Cr steel. Aluminum rods are easy for handling, increased mechanical efficiency and faster rotation. The drilling tool used in this drilling process is chisel shaped.

4.3 Describe simple constructional features of churn drill, drills master, wagon drill and jack hammer.

CHURN DRILL: This type are used for drilling vertical holes of 225 mm diameter and 150 to 300 mm long drill holes. It is suitable for hard rock and is drilled at slower rate as compared to other percussive drilling and STH.

Churn drills are self propelled crawler mounted provided with hydraulic jack to level and lifted up, during drilling. They may be diesel or electrically operated.

The fragments of the rocks in side blast hole fall upon a distance of 0.6 – 1m inside the hole, the rocking lever goes down. The sheave pull down the rope and the drilling tool moves up, when the sheave moves up the tool falls freely with deepening the drill hole. The rope is cut from the hoist drum.

Water is supplied into the blast hole to form sludge's so that the drill bit will strive the fresh surface. Sometime the excessive of sludge's causes jamming in the hole thus ceases the process. Thus a sludge remover sludge of length 3.5 m is inserted into the hole by removing the drill bit. It consist of a non return valve, which will opened up when the sludge rests on the bottom end of the hole. Thus it allows the sludge to come in and is closed down when the slunder.

<u>DRILL MASTER</u>: This is used to drill 165mm dia, drill holes which is a diesel powered rotary air compressor having capacity 17 m^3/min. the traction is by crawler chan. The tramming speed is 3.2 km/h and the gradient negotiable is 15°. The rotary head of air motor gives rotational thrust to the drill rod and feed to the rod is by chain operated by 12.5 H.P. reversible piston by air motor.

The drill rods used are usually of 132mm dia and 7.6 in lengths. The assembly of down-hole drill and bit is 1.8m. So the rod and down-hole drill have a length of 9.4m. So a drill hole of 9m can be possible with a single rod. The total mast height is 12m above the crawler surface. Hydraulic jacks are provided, i.e. two in front and two in the rest to level the machine during drilling operation. A hydraulic pump is provided, driven by diesel engine for supplying power for:

1. Dust collector blower.
2. Hydraulic jacks.
3. Hydraulic wrench used for coupling and uncoupling.
4. Mast raising cylinder.

   The drill cuttings are collected by a dust collector blower which is a hydraulic motor.

They are arrived through a tube to the separate motor with a large hopper. The cuttings stored inside the hopper is used as steaming material.

<u>WAGON DRILL</u>: Compressed air drill mounted on a mobile frame is known as wagon drill. This can be tyred wheel or crawler chain mounted. This can drill ip to 3m to 15 with a drill hole of 50mm to 100mm diameter.

A drifter is mounted, is provided with a lever for reverse rotation and is kept at a steady rate by keeping the lever at neutral position. Another lever is provided for up and downward movement. This drill facilitates a drilling up to 140° inclined. This is mounted on a 3 weather frame of which two front wheels are larger the single rear wheel. The drill cuttings are removed to the surface by compressed air, which is done with a separate lever, to regulate the quantity of air.

The drill rod used in the wagon drilling is of either round or hexagoneal of 35 to 37 mm dia and in long the 1 to 4.5m. Normal air consumption of a wagon drill is $7m^3$

<u>JACK HAMMER DRILL</u>: The air is passed through the drill with the help of hose pipe (19mm dia bore) and curved metal tube with swivel connection. The speed of the drill is controlled by a throttle valve which regulates the amount of the air passes through the hose pipe. The air passage is past pawls on a riffle bar into an automatic valve which directs alternately when the piston is forced down, it's lower and known as stem, strikes the upper end of the drill steel through chuck. In the top of the piston a riffle nut having splines are there which are twisted.

The piston can be moved upward by passing the compressed air through the bottom of the cylinder through automatic valve located at the upper portion of the riffle bar, when the piston moves up, the riffle bar splines exerts a twisting force on the riffle bar and piston as well. The piston experience slight rotation during one complete cycle of up by ratchet mechanism. The extent of twisting depends upon the twist in the riffle bar and spacing between rethread teeth.

4.4 State D.H.D.

The DHD percussive drill operated by the help of compressed air of 5.7 kgf/cm^2. the hole diameter by the DHD percussive drill holes are of 100-226 mm. The hollow pipes are rotated by rotary head. The air motor is used to heighten the threaded ends and drill bits.

The drill bits are usually tipped with tungsten carbide but have 5 no of air cleaning holes, one of the centre and other at the 4 end of the cross bit. The whole unit is called carset bit.

To avoid confusion, the whole machine is known as down-hole-percussive drill but the assembly of hollow pipes and carset bit are known as down-hole-drill.

The DHD contains a piston and valve arrangement. The pressure produced by compressed air is transmitted to the piston and valve and it moves in a reciprocating motion. The piston blow of the piston is devided to the bit. The no of the blows varies from 500 to 2500 / min and rotary speed varies from 1800 r.p.n – 2000 r.p.m.

5.0 BLASTING

5.2 Describe charging & stemming of blast holes

CHARGING A SHOTHOLE / BLASTHOLE

A cartridge of stemming material is push at first to the blast to make the in bye end smooth. After that intermediate cartridges are pushed to the blast hole then after al last the Prime Cartridge will be inserted along with a detonator with its business end directed towards in bye end of the hole. This type of charging known as Direct initiation.

But in the inverse initiation process the cartridge along with detonator is placed after stemming. In this process the business of the detonator is directed towards the out bye of the blast hole.

In direct initiation the proper and of cient amount of shock wave passes from prime cartridges to the next cartridges. It presents the ignition of fire damp and yield of coal will be in case of interact initiations the chances of blown out shot is more. It has two shattering effect as compared to direct initiation process.

After charging the holes the leading wires should be projected out of the hole. The steaming material should be a mixture of sand and clay at a proportion of 3 : 1. The first 2 – 3 cartridges are tamped lightly by the stemming rod and then the rest should tamped hard.

5.3 Determine amount of charge of explosives in blast holes.

Volume of rock blasted hole = length of hole X spacing
= h X s X bm^3

Where h= length of hole in mt.
s= Spacing of hole in mt.
b= burden of the hole in mt.

If the specific cheat of the rock blasted = m
Or bulk density = mt / m^3 of rock

Tonnage blasted hole = h X s X bm tones.

If mkg of the explosive is charged in the hole

power factor= tonnage .1 kg of explosive

$$= \frac{h \times s \times b \times m}{n} \text{ tones}$$

power factor for henaloite one = 3.5 te / kg

manganese = 4.00 te / kg
lime stone = 4.5
chal copy rate = 4.2
coal = 5.6

5.6 Use safety fuse, detonating relay ordinary & Electric detonators.

SAETY FUSE: A safety fuse which looks like a cord consists of a core of fine grained gunpowder wrapped with layers of a tape or textile yarn and waterproof wrapped with layers of a tape or textile yarn and waterproof coating. The burning speed is usually 100 to 120 sec/metre. ICL manufacturers a range of safety fuses to suit various conditions, e.g., Double Bull brand for dry conditions, Blue Sump for damp conditions, OCPS (orange coloured plastic sheathed) and Blue Plastic for wet and very rugged condition. IDL also manufactures safety fuse (yellow). When one end of the fuse is ignited, it carries the flame at a uniform rate to ignite gun powder or to detonate an ordinary detonator which is turn can detonate a high explosive.

These fuses are supplied in coils of different brands after testing after testing for its high quality and satisfactory performance test of each batch.

Type of fuse	Condition for use	Rate of burning of fuse	
Bull Brand	Dry condition	110 to 130 seconds/ metre.	The fuse burns due to direct application of fire.
Blue Brand	Damp condition	100 to 120 seconds/ metre	
W.C.P.S./OCPS	Wet and damp, rugged conditions.	100 to 120 seconds/ metre	The fire is carried up to detonator when fitted.
Blue Plastic fuse	-do-	100 to 120 seconds/ metre	

DETONATING RELAYS: In opencast working, detonating relays using detonating fuse for initiation provide a non electric delay firing system. This method avoids the electrical connections which are required when using delay detonators. A detonating relay is essentially an assembly an assembly of two open ended delay detonators coupled together with flexible neoprene tubing in an aluminum sleeve suitable for crimping into a detonating fuse.

Inside the detonating relay, the construction is symmetrical with the delay element at either end so that the detonation wave may pass in either direction. The delay interval for each detonating relay varies from 15-45 milliseconds. In sue, the main or branch line of detonating fuse is cut at the point where a delay is required, and the detonating relay is then crimped between the two cut ends of the line. By judicious selection of the points at which the detonating relays are inserted, any delay firing sequence can be arranged. Being non electric in nature, detonating relays are insensitive to stray current and static electric.

ELECTRIC DETONATOR: Electric detonator are same as that of plain detonator when the prime charge & base charge is concerned. But it is different from plain detonator by the process of initiation.

Electric detonator may be classified as

a) Low tension electric detonator, b) High tension electric detonator.

A electric detonator consists of a neoprene plug assembly at the upper part which consists

a) Electric fuse head, b) leading wire, c) Neoprene.
Plug through which the leading wire passes through.

A low tension electric detonator consist of two thin strips of brass which are separated from each other by a insulting board and are diagonally connected at the lower end with a fine bridge of nickel chromium alloy.

This lower end is immersed in a lead of ignition composition. The lead wire is soldered with the brass plate so that the whole will be a complete metallic circuit. The fuse head consists of lead mono-nitrate resorcinol (LMNR), potassium chlorate, nitrocellulose charcoal.

The leading wires are 1.8 mt long with plastic yellow coloured coating for water resistant. The scaling plug made up of neoprene is waterproof.

The internal resistance of a L/T detonator 1.3 ohm with 45 mt length of shot firing cable of 7 ohm resistance. The current required for ignition is 0.50 amp. So a detonator can be blasted with a minimum voltage of 3.5 volt.

5.7 <u>Narrate various types of explosives used in open cast mines – ANFO, LOX, Slurry Explosives, Emulsion Explosive.</u>

<u>ANFO (Ammonium Nitrate Fuel Oil)</u>: Ammonium nitrate, mixed with diesel oil, is used on a large scale for blasting in the quarries of coal and metal mines. The most effective and oxygen-balanced explosives mixture is one with 5 to 6% of diesel oil (by wt.). It has a Sp. Gr. of 0.8 to 1.0, wt. strength of 75-80 and velocity of detonation at 3500 m/sec. In the dry season, 7 litres diesel for 100 kg of AN suffice but in wet season, the quantity should be increased to 9 litres. Diesel oil in excess of 8% tends to lower the sensitivity of ANFO to initiation. The mixture causes irritation of the skin and the workers should, therefore, wear gum boots and rubber hand gloves. The mixing should be done with wooden shovels avoiding contact with iron. The mixture is safe to handle and without formation into cartridges can be mechanically loaded into blast-holes. Where ANFO consumption is heavy, stationary ANFO mixer similar to the concrete mixer may be placed at a centrally selected site. In case of a pneumatic ANFO loader, an electric detonator should not be used unless steps are taken to prevent premature initiation due to static electricity.

As the mixture cannot be initiated in the normal way by a detonator it is necessary to prime it with a small quantity of O.C.G. or booster. It is a good practice to use high explosives are difficult to sink in water due to low density of AN, and should preferably not be used in watery holes. If however, An-fuel oil mixture has to be used n watery holes it should be packed in polythene bags and forced down the hole with the weight of a high explosive and stemming above it. Holes of 62 mm dia. and above are considered economic for use of AN-FO explosives.

An increase in blast-hole dia. beyond 300 mm decreases the sensitivity to initiation of ANFO explosives. With above 4 % water an ANFO, the velocity of detonation decreases sharply and the mixture with 9% or more water cannot be detonated. When using ANFO, it is essential to have uniform mixing of ammonium nitrate and fuel oil. If the ammonium nitrate is not of adequate porous quality, it may separate from fuel oil resulting in inferior performance.

AN-FO explosive cannot be initiated direct by No. 6 detonator. It can, however, be blasted by a detonating fuse which needs no. 6 detonator for initiation. It may be initiated by no. 8 detonator which is not much used in mining practice.

<u>Slurry Explosives</u>: Slurry explosives are those explosives which consists of jelly consistency and are water gels. (water gel is mixture of oxidizing and are thickened with gums and gelled by cross linking agent. In permitted explosive a coolant is used in addition to reduce sensitivity.

The 1st commercial slurry explosive was developed by Dr. Melvile Cook in USA 1957. He used a composition of TNT, AN and H^2O at an proportion of TNT (20%). AN (65%), H^2O (15%). To this composition chemicals are added for gelling and cross linking to stabilize the homogeneity of the mixture. Some of the common ingredients are

Oxidizing agent	: ammonium, sodium or calcium nitrates.
Cross-linking agents	: potassium or sodium dichromates, antimony or boron compounds.
Gelling agents	: Starch
Fuel sensitisers	: TNT, PETN, pentolite – (all explosives). Aluminum, sugar, urea, paraffin, hexamine, ethylene glycol, woodpulp – (all non – explosives)

The slurry explosives has a sp. Gr. more than 1 and like ANFO can be poured directly into watery holes. they are also available in the form of cartridges with plastic or polythene wrapper and some (permitted type) can be used in underground coal mines. Such slurry explosives for use in underground coal mines have to be cap-sensitive and approved by the DGMS. The slurry explosives is highly water resistant. In the quarries holes of diameter 62 mm and above are economical for use of slurry, just as for ANFO, if it has to be poured into blast-hole.

The components required for ANFO and slurry explosives may be mixed at a plant away from the blasting site or at the blasting site itself. In the case of **PMS (plant mixed slurry)** system, the explosive is loaded into special tankers and from these tankers, the slurry is pumped directly into the blast-hole.

Unlikely cartridges slurries, pumpable slurries can be tailored to have the appropriate density depending on strata conditions. In the case of site **(mixed slurry system) SMS** only non explosive ingredients are stored at a warehouse and transported to the blasting site in a specially designed pump-truck.

Advantages of slurry explosives:

1. Slurry explosives don't explode on rough handing or it dropped, from shovel impact even set to fire.
2. It doesn't contain toxic fumes & headache gases.
3. Exudation from water get is harmless as in maximum cases, it is nothing but water containing some dissolve inorganic salts.

Emulsion Explosives: An emulsion is an intimate mixture of two liquids that do not dissolve in each other. In more technical terms, an emulsion is described as a two-phase system in which an inner or dispersed phase is years, contributed to our daily lives in such products as insecticides, photographic films and papers and cosmetics,

The unique feature of an emulsion explosive is that both the oxidizer and the fuel are liquids. The unique properties of emulsion explosives are due to the minute size of the nitrate solution droplets and their tight compaction within the continuous fuel phase.

Emulsion explosives depends entirely on the presence of voids for initiation and propagation. A change in the amount of voids effects a change in density. It is convenient and useful to relate properties to density and to consider voids and density adjustors.

Emulsion explosives are highly water resistant. They are more fluid than slurry explosives and therefore create problems when loading a blast hole with fissures or cracks. They lack the strong cross-linked gel that characterizes the TNT-sanitized slurry explosives.

6.0 CONTROLLED BLASTING TECHNIQUES

- Increasing the use of technique to control damage the remaining rock is being practiced in surface mining.
- In pre-splitting holes in the last row is detonator (blasted) in the beginning.

➢ Thus creating a cut between holes in cushion blasting a simple row of holes along the next excavation line the hole is loaded with light chord explosive with completely stemming and fired after the main excavation is removed.

6.1 <u>State and describe pre-splitting, cushion blasting, muffled blasting, coyote hole blasting, chambered whole blasting, directional blasting.</u>

<u>PRE-SPLITTING</u>: It creates an artificial discontinuity along with the periphery of the designed and expected limi of the main blast fragmentation to isolated to the blasted zone from the remaining rock mash.

It helps the prevent over break and creat smooth and shable high wall and help for reflection of the shock wave from the main blast.

Pre-splitting can be done in two way (i) By in hole delay arrangement, (ii) By air decking method.

<u>CUSHION BLASTING</u>: When the aim of basting is to obtain lumpy coal or minimum formation of dust the blasting practice is modified to cause heaving acting even though high explosive is used. For such reason low density high explosive is used. In such explosive addition percentage of combustible material is mixed in high explosive such as woodmeal. The large volume of explosive will have the same blasting effect as the original volume of high explosive. The explosive strength of the explosive is distributed to large length of hole so such explosives will cause the heaving effect in place of its original shattering effect. Same heaving effect can also be obtained in case of cushion blasting. In this case of blasting the hole is first charged with explosive followed by priming cartridge. The priming cartridge is further followed by one or two plastic air bags. The explosive charge and the air bags are slowly pushed up to the back of the hole. The remaining length of the hole is stemmed with stemming material. Initially the stemming material is very highly stemmed. At final stages the hole is tightly stemmed. In such blasting of hole the effect of blasting is heaving. The air bags work as the cushion. Such blasting with air gap is known as cushion blasting.

<u>MUFFLED BLASTING</u>: Sometimes the opencast mine working is made very near a populated area. Or the working of the opencast mine are near the busy roadways. In such cases of working there are danger due to blasting. The rock pieces are shattered due to blasting. These pieces may hit to a person who is long distance away from the place of blasting. To overcome this type of accidents muffled blasting is carried. In this case the hole is charged with high explosive as in case of general blasting. The grid of steel rods of dia 2.5 cm to 5.0 is prepared by placing the rods at 15 cm to 25 cm distances. And the grid is covered with cement bags filled up with sand.

After covering all holes by grids and sand bags the holes are blasted. The grid with the sand bags fly vertically above the hole forming the cover above the blasted material above the hole. Thus the rock pieces ejected due to blasting are prevented to fly to a long distance. But these pieces fly in air with the grid and sand bags vertically above the hole. They again pull down. As such, the pieces will not flow to long distance. The blasting can be safely conducted near the populated areas and roadways without any danger.

<u>COYOTE HOLE BLASTING</u>: In this system a large number quantity of explosive charge nearly few hundred tones is packed.

In the large chamber inside the underground excavation (tunnel drifts raise)

Reach chamber are made by driving (tunnel drifts raise) and are called coyote chamber.

After packing explosive in the coyote chamber the connected tunnels strip raises are been filled tightly which the muck.

Excavated when forming the chamber and the charge is normally blasted with the help of electro noting cord.

<u>CHAMBER BLASTING</u>: When high benches are blasted in the opencast working the explosive charged in the hole occupies the large length of the hole. While small length of the hole remains for stemming of the hole. Unless the proper stemming of sufficient length of hole is made, there will be blown out hole without performing any useful work. In case of blasting the chamber blasting practices are used. In this blasting, after the drilling of hole is complete in the bench, the hole is cleaned. One small priming cartridge is placed in such hole at its bottom. One to two kilogram of dry sand is poured in the hole to cover the priming cartridge in the hole. Such a charge of explosive is connected to the blasting cable. The small priming cartridge is blasted. Due to such blasting the bottom of the hole gets blasted. The material blasted at the bottom of the hole g and the dry sand put in the hole are thrown out of the hole due to blasting. Now the space formed at the bottom of the hole will be large. The charge of explosive when placed in the hole for blasting of bench the length of hole occupied by the explosive is small as large amount of explosive get filled in the chamber. The large length of the hole is available for the stemming of hole. The large amount of explosive is concentrated at the bottom of the hole. So the material will not remain un-blasted near the foot of the bench to cause imperfect blasting and more of dressing of hole. But the material at the bottom of hole is thrown away from the side of the bench at the bottom and the blasting is perfect.

<u>DIRECTIONAL BALSTING</u>: Over burden casting by the directional blasting offer a no of advantages.

Saving in time and capital and operating cost in re-handling the blated material which is known as transported.

Every cycle of operation increases the over ore production. Production will increases with a development mining equable the directional involves a very complex inter acting factor. Which include drilling and explosive accept, costing factor, air drag, surface topography of the blasting side and spoils dump - the distance between the blasting side and spoil dump.

7.0 <u>SECONDARY BLASTING</u>

In opencast mining the high benches are formed. During the blasting the high benches are broken. Many times the required fragmentation is not formed but the benches are blasted to large side of blocks. These large blocks are not handy and these blocks create loading problems by machines. To break such big blocks into handy fragmentation the secondary blasting is carried in such block. There are three methods of secondary blasting: (1) Pop shooting, (2) Plaster shooting and, (3) Snake blasting.

(1) <u>Pop Shooting</u>: A small hole of 30 mm dia and approximately 0.3 to 0.4 m length is made in the block to be blasted. These holes are charged with small priming cartridges. The electric detonators are used for priming. These holes are blasted by taking proper shelter and giving the warning to the workers to take shelter. Due to this blasting there is shattering effect. The block is broken to pieces. The blasted material is scattered and small pieces fly to long distances. The separate arrangement is required to be made for drilling small dia hole. The blasting gelatine is used of different strength for such blasting. Works are stopped during blasting. Separate arrangement of drill, compressor and workers to operate the machine is

required to be made. There is no much sound and explosive consume is lost as compared to other methods of secondary blasting.

(2) Plaster Shooting: In this case of secondary blasting the surface of the block is dressed slightly on the top of the block to make a small area plane. The plaster gelatine slabs are available of high explosive of size 83cm X 64cm X 13cm and weight 125 gms or power blast of size 80 X 60 X 25 cm size and 250 gms weight are used for plaster shooting. The slab of explosive is placed on plane area of block on its top. The slab of explosives is covered with thick mud on its top and side after priming when such a charge of explosive is blasted it makes very high sound. But the explosive being not confined to hole the material blasted is not thrown or scattered. But due to the shattering effect of the explosive the cracks are developed in the block in its own place. The block can then be spitted in small and handy block along the plane of cracks developed in the block. In this case of blasting there is no necessity of drilling of any hole and there is no scatting of blasted material.

There is no plying pieces of rock thrown by this blasting. High sound is made due to which this blasting may be objected when blasting may be carried near the populated area. The consumption of explosive in this case of blasting is more as compared to pop shooting.

(3) Snake Blasting: In this case of blasting the block is blocked by the travel of the explosive waves through the block to be broken. The explosive is not placed on the block top of the block. The explosive charged is placed at this space below the block such the explosive charge is almost touching the block. This charge of explosive is covered with thick mud as far as possible including the possible side. When such a charge of explosives is blasted under the big block the detonation waves of explosive charge travel in all direction including the open side. The detonation waves traveling in the open space (side) are lost without any useful work. While the waves traveling in the block cause breaks in the block. The block is splitted along the breaks and cracks cause in the block by explosive. Thus big block is broken to small block. This makes the block handy so as to load by mechanically.

In this case there is no flying of broken pieces. It is sae blasting. It can be used near populated areas and roadways. No drilling is required. There is no requirement of separate drilling machine, compressor and workers required by their operation, so the cost will be reduced. Much of the blasting emerging of explosive is lost through the open gap where it is not possible to cover the explosive by mud. It is difficult to cover explosive on all sides. The explosives consumed is much more as compared to pop shooting. In this case the explosive consumed is 3-4 times compared to pop shooting. The cost of explosive consumption is high. It makes high sound during blasting.

8.0 DESIGN OF OPEN PIT.

8.1 Determine height, width and slope of a bench.

HEIGHT OF BENCH: If the height of the bench are increased the no.of bench will be less. An it will helps efficient leading and transportation the following factors are govern during the section of height of bench.

- ⇨ Thickness of deposit depth.
- ⇨ Capacity of drill machine.
- ⇨ Maximum digging depth of the dragline.
- ⇨ Nature of rock.
- ⇨ Transportation system of the machinery.
- ⇨ Stability of rock.
- ⇨ Slope of the bench.
- ⇨ Plane of weakness.

- ⇨ Variation of ore grade.
- ⇨ Physical nature of ore and mineral.
- ⇨ Extent of over burden.

Advantages of larger height of bench.

- ⇨ Efficiency of the shovel is increased of the bench height match with the machine.
- ⇨ Minimize the amount of ripping and leveling out of the beam.
- ⇨ There are fewer benches in the opencast mining transportation system improved.

Disadvantages of larger height of bench.

- ⇨ Super vision of the benches are difficult.
- ⇨ Deep hole blasting may yield oversize rock fragmentation which required secondary blasting
- ⇨ If the eight of bench is increased probability of slope failure.

WIDTH OF BENCH: It depend on height of bench manner of blasting and transportation system. Bench width for rail transport system is greater then that of the truck haulage. During the determination of bench width, the width of the muck pile, clearance between the muck pile and the width for truck / drumper movement width of either single lane traffic / double lane trafoic. Clearance for the safety of truck from the ede of the benches. Etc are to be considered. Optimim bench width is essential otherwise if the bench width is more then it will adversely effection economy since this will delay production and lock up capital for a longer period. After reaching to the limit of pit the width of the bench is generally reduced forgiving a shape of overall required pit slope.

SLOPE OF BENCH / SLOPE ANGLE: The angle of slope of bench should be either equal or less than the angle of repose of the bench rock. In moderately loose and friable rock the dressing in the bank slope is necessary upto the angle of repose of the rock to prevent over hanging and subsequent falling of rock when the materials get loosened by the action of weather or vibration of the beavy earth moving machineries. In competent rock angle of slope of the bench veries beween 70° and 85°. This slope may be obtained by verying the inclination of the blast holes. However, this slope angles also depends upon the plane of weakness, orientation of bedding plane, etc. Slope failure is due to mainly by a slow process of rational shear, mechanical properties of rock like cohesion, angleof friction, etc, pore water pressure seepage forces, tension crakes etc.

8.2 Calculate explosive charge for a given production.

Volume of rock blasted per hole = (s X b X h) m^3

Where s= spacing in mtr.
b= burden in mtr.
h= depth of hole in mtr.

If the specific gravity of the blasted rock = w,
Then tonnage blasted / hole = s X b X h X w tone
Let nkg of explosive is charged per hole then
Power factor = (s X b X h X w) / n
Power factor haematite ore = 3.5 m
Manganese = 4 m
Limestone = 4.5 m
Coal = 5.6 m.

8.3 Design haul road.

The design of the haul road should be such that with minimum cost it should have high load bearing capacity and tally the designed traffic loas. If the surface topography is gentle, then heavy traffic is selected. Compare to the mines at hills since they have to laid on several severe grades, water seepage during rainy season also causes severe land sliding problems. Adequate regular water sprinkling arrangement or salt zoning (spreading cells or NaCl on the road surface and after then spraying of water with help to consolidate dust on the crust). System has to introduce for efficient suppression of dust and to stability the road surface. The design of haul road should tune up with the following.

- ⇨ It should offer efficient transporting operation of mineral / ore / overburden.
- ⇨ Selection of the site of the road should be such they it enables minimum cost of transportation of ore / coal overburden and there is no weak ground or slope stability problems. The site of the road should be safe and has ease access to the mine working site.
- ⇨ It should be sufficiently wide to avoid traffic congestion. No real shall be of a width less than 3 times the width of the largest turn outs and waiting pants are provided. One way traffic is the best preposition for any surface mining operation.
- ⇨ Very safe and less maintenance cost, road are to be maintained in good working condition.
- ⇨ It should nicely balance the linkage among the crussing and screening plant, verification plant, stock yards, waste dumps etc.
- ⇨ For better safely the optimum gradient of a haul road is around 5° however the gradient may go up to 9°. When severe gradient comes to a question then for every 500m, a patch of 50m at a gradient of less than 1° is to be provided for the purpose of safely after taking due to consideration of drainage. Unless specially permitted by the chief inspector of DGMS shall have a gradient steeper than 1 in 14 of any place. A gradient up to 1 in 10 may be permitted for designing the ramps.
- ⇨ In the haul road vertical and horizontal curve should be laid in such a manner that it will provide enough safe sight distance (not less than 30m along the road) sight and stopping distance are very important in both the curves. If it is not possible to ensure a visibility for a distance of 30m, then the two roads. (one for up and the other for down traffic shall be provided)
- ⇨ Road may be single lane one way trffic number of land depends upon the traffic density, normal safe distance between the trucks, truck speed, constraints of space etc.
- ⇨ Safe distance between the vehicle which depends upon many factors e.g. gradient, speed of the vehicles, coefficient of odhesion, sensation of driver etc.
- ⇨ In the curves additional allowance in width is to be added with both the curve and tangent to the curve, besides the Stanford of width (at least 3times the width of the largest vehicle) already mentioned.
- ⇨ Super elevation is take provided on curves to educe the side friction between the typrs and the road surface, super elevation should be driven at slow speed in super elevation.
- ⇨ Culverts and drains are to be provided all along the haul roads the factors which leads the design of the drainage system depends upon the degree of rain fall, ground topography, catchments area, depth of the road base etc.

9.0 DRAINAGE IN OPEN PIT

Source of water in rush into the surface mining.

Source of water:

1. Surface water due to rain, flood, snow and also the ground water are the source of potential hazard in surface mining system particularly during rainy season.
2. Water may also in rush into the surface mining system from the rsea by the tidal wake during typhoon from the river during flood.
3. From the isolated water bucket targeting the ground.
4. Depending upon geological surface topography permeability strata area, hydraulic gradient, porousity of rocks, presence of joints fissure in the rock and ground water table.

Control major:

1. The pit floor should be sloped around towards the extraction side to drain out rain water.
2. When the floor rock is impermeable rock in nature pit is always above the water table.
3. Slope is stpered quantity of rain falling.
4. surface water body like river, lake, nala may diverted or drain off other constituting sufficient amount of drain or trend.
5. Pump should be install in the sump which is preset lowest level of surface mining.

Mechanized opencast mining : In this method open cast mining heavy earth moving machinery like draglines, power shovel, dumper, hole drills etc are used. The blast hole are 6m to 18m deep and 125mm to 250mm diameter. The rock is blasted by explosive like liquid oxygen, open cast genginite or other high explosive. Mineral are ver burden are transported by locomotives belt conveyors and large truck known as dumper.

Favourable condition for mechanized open cast mining : For mechanized quarries employing heavy earth moving machinery. The DGMS makes by law covering bench sizes, roads etc.
Mechanized opencast mining is preferred when there is a thick mineral bed of mild indicates at a low depth and the reserve are plenty bully.
The overburden may be removed by a combination of dozer and scrapers if the rocks are hoar blast hole are derived by wagon drills and blasted with explosive.
The blasted rock is loaded dipper shovel or tractor shovel.

Choice for mechanized opencast mining :

- Thick mineral bed.
- Mild indication.
- At a low depth.
- It should not in pockets.
- Mineral bed be continuous.
- It should have a longer life.
- Beches.

Stripping ratio : It is the amount of over burden (m^3) required to get 1 tonne of mineral. Unit = m^3 / tonne.

Over burden : It is the excess amount of mineral soil able deposit or mineral.

Break evens Stripping ratio : It is the point at which the Stripping ratio becomes un-economically.

Cutobb Stripping ratio : Break even Stripping ratio is same.

Limitation of open cast mining :

1. Mineral bed a line below 400 mt, mechanized mine are not applicable (depth).
2. Stripping ratio : KOIAR gold mine – 3200mt depth bench of beam are same meaning Stripping ratio - m^3 / tonne.

Bench Parameters :

Parameters for height of bench :

1. Thickness of mineral deposit.
2. Depth of mineral deposit.
3. Transport system.
4. Slope of the bench.
5. Nature of rock.
6. Stability of rock.
7. Maximum digging capacity of dragline. If the height of the bench is increased.

Advantages :

1. Lathering cost is reduced.
2. Formation of number of benches are less.
3. Efficient transport system can be achieved.

4. If the height at the bench matches with the sawed height, it excavation is easier.

Disadvantages :

1. Because of vomiting, stabbing etc. probability of slope failure is increased.
2. Due to slope failure score damage of machinery and serious accident of workers occurs.
3. Supervision of benches is difficult.
4. Deep hole blasting may yield fragmentation in ore which may require secondary.
5. complexity may arise mining.

Width of bench : Width of the bench depends upon following face.

1. Height of the bench.
2. Manner of blasting.
3. Transportation system.
4. Width of the bench w= largest dumper moving or W = largest machinery.

Bench width for rait transport system greater then that of the truck haulage system. The inter system has greater movement flexibility in operation.

Slope angle : The angle of slope of bench should be equal or less than the angle of repose of the in case of weak and friable rocks the angle slope of bench is equal to the angle of the bench slope in competent rocks any slope of bench varies between 70° and 85°.

In order to maintain stability range of the bench may be due to heat moving machinery playing on it and due to adverse effect of the climate and may be the same extend of blasting.

Soft clay = 25° - 35°.
Hard stone, sand stone, lime stone = 40° - 50°.
Weathed igneous rock = 50° - 60°.
Metamorphic rock = 60° - 70°.

Length of Bench :

Let an opencast mines has to produce Y million tonne ROM (run of mines) per annual, let the height of the bench, burden and spacing of the holes are H, b and respectively.

Yeild per hole = h X b X s X y

Production per week = --------------------- X 6 tonnes.

Where 6 is the number of working day in week number of net working days per annual = 250.

So number of holes required per week.

= ______________________ = ________________________

Length of face required to accumulate above number of holes.

= ---X spacing

= --in meter

Or production of shovel face another equivalent face length is also required for drilling

Total face length=

Commended face length for grade control another length will be required in mine

Total number of face = --------------------------------------- tonne

Length of each bench is fixed to Z meter in number of benches required

=--------------------------------

<u>Type of drill used in opencast mines</u> :

1. Rotary drilling : (a) Auger bit drilling.
   (b) Diamond bit drilling.
   (c) Roller bit drilling.

2. Percussive drilling : (a) Churn drilling.
   (b) Air operated hammer drill.

3. Rotary percussive drilling : (a) Down the hole drilling.
   (b) with drag chisel bit.
   (c) with roller bit.

4. Thernnal drilling.

1. <u>Rotary drilling</u> : It is one type of drilling which exhaust by tensile force.

(a) <u>Auger bit drilling</u> : This is used for vertical and inclines drill hole of dia 125-160 mm and upto a depth of 25m. they are used in coal, soft type stone etc. They are capable to giving an output of 15-106 m/shift.

(b) <u>Roller bit drilling</u> : They are used for drill holes of 190-320 and depth is 35m in rocks having Rol = 5-10. they are capable of giving an output of 20-150 m/shift.

2. <u>Percussive drilling</u> :

(a) <u>Churn drilling</u> : It is used hetrogene out rock with Rol = 10-20 and is used for drilling exploratory hole 150-350 mm dia upto 50 m depth.

(b) <u>air operated hammer drill</u> : This is used for drilling blast holes of dia 32-40 mm in igneous and metamorphic rocks.

3. <u>Rotary percussive drilling</u> : Rotary percussive drilling means drilling which air hammer tools. This type of drilling is applied too drill holes at 50-200 mm dia. Depth of 30 mt in rock having Rol= 5-10. they are suitable for highly of difficult to drill kinals of rocks. Its output verify 10-30 m/shift.

<u>Down the hole :</u> they are suitable for drilling large holes 65-230 mm dia. Vertical holes in hard strata upto 9.

<u>Types of drill bit used in opencast drilling</u> :

1. <u>Drag bit</u> : Drag bit is for small diameter hole ranging from 45-75 mm generally have two cut wings, coated by tangsten carbide.
   For soft rocks the coal wings are long narrow but for hard rocks like shale sandstones. They are short and has a smaller gap between the wing.
2. <u>Roller bit</u> : The strength at the beat increases the dia of the hole increases. These are suitable for drilling relatively larg diameter holes in rocks like shale,

sanat stone. Now usually canial rollers are used. The cut ofbit is coated with tangstem carbide.

3. Auger bit : Noe flushing auger bits are for drilling large diameter holes (upto 900 mm) soft rocks. The bit which is for soft rock consists of the wings with b cetral point screw. For hard the wings are coated with tangstem carbide.

Blasting :

Spacing : Spacing is the distance between the advent holes is a rock. Spacing together with burden is an impar poue as it determines the total volume of rock to be broken. Spacing is usually refeated between one and two types the burcen as the break out angle is about 90° when dealing with the hard deposits and where good fragmentation is far mechanical loading. The spacing is usually selected to two times the burden.

Burden : Burden is the distance to the nearest force have from blast. The optimum burden is generally found to range from 20-25 times the blast hole diameter. An insufficient burden causes, excessive air blast and fly rock. Burden is one of important factor which effects the air blast. If the burden is less than optimum, the excess explosive gases vent from the burden and increases the intensity of air blast.

Stemming : The main purpose of stemming is to confine high pressure detonation gases that are refused immediately at detonation, sand is considered to be the best stemming material for convenience, a mixture of 6 parts at sand and part at clay is recommended. Too short stemming length can give rise to excessive air blast and fly rock. Too large stemming length can create excessive boulders in the upper part of the mines.

Deck charging :
The charging of explosive in a hole at more than one places is known as deck charging.

Necessity : In opencast working of mine, where the high bench has hard and soft bands, the blasting effect on the bench is not proper and uniform. The complete change at the hole than blasting is carried of such benches. The rock of the bench the hard band remains unaffected due to such blasting and this causes over hanging.

In order to avoid such over hanging of rocks the complete explosive change of the hole is placed of moment than one places in the hole and this techniques is known as deck blasting.

Process : In deck blasting some change of explosive is placed of the bottom of the hole and hole is stemmed upto hard blast first band position, the second charge of the hole is placed at the 2nd band position.

The charge of explosive placed of the bottom of the hole is 2/3 cm v_2 of the total charge of the hole while the rest of the explosive charge placed at hard band is 1/3 at second position or incase remaining ½ charge is placed at 2nd and 3rd placed against the hard band in the hole. Such charging at the hole will blast the bench without causing over hanging as hard bands are standard.

Single raw blasting : In single raw blasting all the shots are fired simultaneously, simultaneous firing makes blasting easy and quick as all the holes can be connected to a single detonator through detonating fuse. This type of shot firing often preferred but simultaneous firing at a large of shots in large dia drill holes cause ground station which may damage building and structural so here we chosse short delay blasting.

Multi raw blasting : Multi raw blasting is less commonly used mines and rarely lore than two raw. This type blasting gives better rock fragmentation than single. It is used where a large production from a is desired. But this blasting heed higher specific explosive.

Deep hole blasting : Before charging the hole is clear its depth is measured. The burden is check excessive work. Then the cartridge are insert and each pressed home. After loading the stemming is inserted when all the hole have charged and stemmed the cardox from each by a suitable point to the main line of fuse which is initiated electrically or by plain detonator. The holes are generally fired in stage in blasting programmed, such as drilling and firing are planned and carried out. So as to ensure the maximum safety and to maximum production.

Controlled blasting techniques :

Vibrations are setup in the ground where are acting is done at underground level. When the explosive are very heavy, the vibration can be done enough to damage building. The controlled blasting techniques has been evolved with a view to caresting flying rock fragments and to control the ground action produced due to the blasting operation.

Before the application of controlled blasting intrance a number of blasts are to be made to determine the specific consumption of explosive for particular site and for different rock formation of charging has been preferred in controlled blasting. So as to take advantage of various rock along the length of blast hole.

System of chargin in holes for overburden bench.

1. Rock formation.
   Shape / medium hard sand stone.
2. Pattern of drilling.
   For 6.3 m to 18.8 m depth of less.
   Depth X spacing X burden.
   18.8 to 6.3 m X 5.4 m X 5.4 m.

LOX (Liquid Oxygen Explosive):

It consists of a combustible in gradient which is socked in liquid oxygen (oxygen gas liquefies at 183° C). The absorbent catridge contains a special composition called Loxite. Used to reduce the temperature of the gaseous product after the blastin. Then the absorbent catridge wrapped in paper covered by cloth when it subjected to reaction of combustion. The relation takes place to release very large volume of gases at high temperature,

Salient features :

1. It doesn't require booster.
2. It characteristics are not constant since it depends on the time gap between bining, the life of lox cartridge in open is about one hour.
3. Large blasts are not suitable for use it since time needed to complete charging and firing is more. This leads to the evaporation of the oxygen.
4. Highest standard of safety to be observed.
5. It can be fired with or without the aid of detonator. It can be fired by safety fuse. Holes dipper then 3 m are fired by detonating.
6. The lox cartridge should be used in the field without delay to prevent loss of absorbed liquid oxygen.
7. Grease and oil should not come in contact with cartridge.
8. It can not be used if Iron pyrite is present in the hole.

9. When it used in watery hole, the cartridge packed in polythene bag.
10. It has no headache producing ingredients.
11. A lox cartridge ready for blasting is prepared at the depth by soaking on absorbent cartridge in liquid oxygen.
12. Lox cartridge are inflammable and the flow of gases oxygen eliminating from a cartridge will causes surrounding materials, glowing coal and lig slubs burst into flames. Lox should therefore be kept away from such burning material.

Different methods of quality control in opencast mines :

In cases of open cast mines, quality can be control by following way.

1. flexible mining method.
    (a) By working in number of benches simultaneous.
    (b) By working at variable depth.
2. By selective mining method.
    (a) By adopting highly mechanized mines.
    (b) Movability of used machine.
    (c) Continious minor.
3. By blending : It means mining high grade one with low grade ore to obtain cutoff grade of container.
    (a) Traffic control : In this system, ore are transported from mine in a regular manner from one bench then other.
    (b) Different type breaker.
    (c) Transportation system.

Sources of water in open cast mines :

Surface water due to rain, flood, snow and also the ground water are the sources of potential hazards in surface mining system particularly during rainy season. Besides above water way also rush into a surface mining system by the sea by the tidal wave during typhoon, from river during flood.

Problem of water in mines :

1. Slope stability of pitwall clearances.
2. Slope stability of dumper decreases.
3. Machine can't move in a watery ground that way production decreases.
4. Maintenance cost and operating cost increases.
5. Caves down the blast holes.
6. Pumping of water causes extra expenditure.
7. Supply of electric power extra expenditure.
8. The haulage system will not work properly.
9. Wet ore, coal stick onto the belt conveyor, cost of transporting and processing of ore increases.
10. Heavy water may decrease mine life.
11. Acid water seepage may destroy the surrounding cultivating land.
12. Water decreases the efficiency of labour and machine and cost of production becomes higher.

Precautions against water in mine :

1. Pit floor must be inclined, approximately taints the working bench.
2. Embankment must be erected around the water boaring sources.
3. Sump and adequate pumping arrangement must be provided.
4. Precautionary bore hole pumping must be done.
5. Mine entry must be lie above HLF / bank.

Layout of opencast working :

Describe the opening of the deposit box cut opening up of open pits is done by an opening cut for the development of first working bench. The opening cut is called the box cut and suitable gradient. Sleepest is advisable both technical and economical points of view for transport, holding space and mining the cost of excavation for deep pits is necessary.
The box cut may be internal located on berits on one side of the pit, saves cost of excavation suitable for deep pits since number of benches in this space is more and excavation by external means is extremely large and costly and external suitable for shallow pits.
Bore cut is excavated initially down to the floor level of the first bench from the surface. Then a level bench for opening is extended from its opening cut to from the first bench the opening bench is narrow keeping due regards of the turning of the machineries used for excavations and extends along or across the quarible limit depending on the type of the deposit. When the first bench is sufficiently advanced, the box cur is riented and extended to the nest lower bench keeping due regards of the sufficient amount of rooms for the approach road to the top (1^{st}) bench and for opening bench for the way is number of working benches are developed and the width of the box cut should be sufficient enough to diversity the approach road to all the benches. If the number of benches are developed from one opening cut, the cut should be started enough away from the pit limit. So that bottom bench can be reached at the desired slope of the pit.
This type of opening cut may be very long and may be curved depending upon the shape extend of the deposit it or opening upon hilly deposit, a central bench cut is given across the top level for the first bench of from one side in the same contour level, forming a length of face which will give the required production rate

<u>Layout for loop layout</u> :
<u>Description</u> : In this type of layout, have distance is longer as the have road runs along the periphery of pit. In this layout mining has to start from the pit.
<u>Loop layout</u>
1. Pit entrance is outside the mines .

<u>Applicability condition</u> :
1. this method is practiced where pit entrance is outside the mines boundary.
2. Mineral bed is deep sheated.
3. The extent of mineral bed is more.

<u>Description</u> : In such cases the mineral and overburden may have to be worked in a number of benches. When such pits not too deep can be laid out with separate transport roads or tracks to each benches shown in figure. The track on each bench makes a complete so that a train need not back out of the pit.

<u>Advantages</u> :

1. Very first transportation.
2. Different haulage for different mineral and overburden bench.

Disadvantages :

Initial development work or haulage roadway is high.

Pit entrance is inside the mines :

This type of layout is practiced where the pit entrance is inside the mine boundary this layout is only possible for truck haulage.

Advantages :

1. Initial development is law as compared to pit entrance outside the mine.
2. Transportation can done from the lowest bench directly.

Disadvantages :

Roadways are at very high gradient.

Switch back layout :

In case of still dipper pit. It is not possible to provide a separate track or road to each bench from the main pit approach such condition, the switch back system of track layout may be adopted. For the whole train has to shunt back and forth on each bench in order to get down the bottom most bench along the path. Shown by the arrows.

Applicability condition :

1. This type of layout is practiced for truck or train haulage.
2. Where it is very difficult to maintain loop layout.
3. When mineral bed is still dipper carthed.

Advantages :

This involves much time consuming shuting operation in addition to requiring the provision of sufficient length of tail track for accommodation the whole train.

Disadvantages :

1. Continuous production from a bench is not possible because unidirectional transportation system is adopted.
2. The ramp roadways are at higher gradient.

Spiral layout :

Applicability Condition :

1. This type of layout is practiced where shape of mine is approximately circular.
2. When the depsit occur at a greater depth.

This type of layout for truck haulage. The haul distance is longer in this case though the speed of the haulage is faster owing to the absence of switch back or sharp curves. Since the haul road runs along the periphery of the pit, mining has to start from the pit boundary and proceed downwards. The bench at a time. This may be no economic disadvantages if all benches are mined in ore, but its ore body has a thick cover of overburden it involves heavy investment in stripping in the early life mine with no provision.