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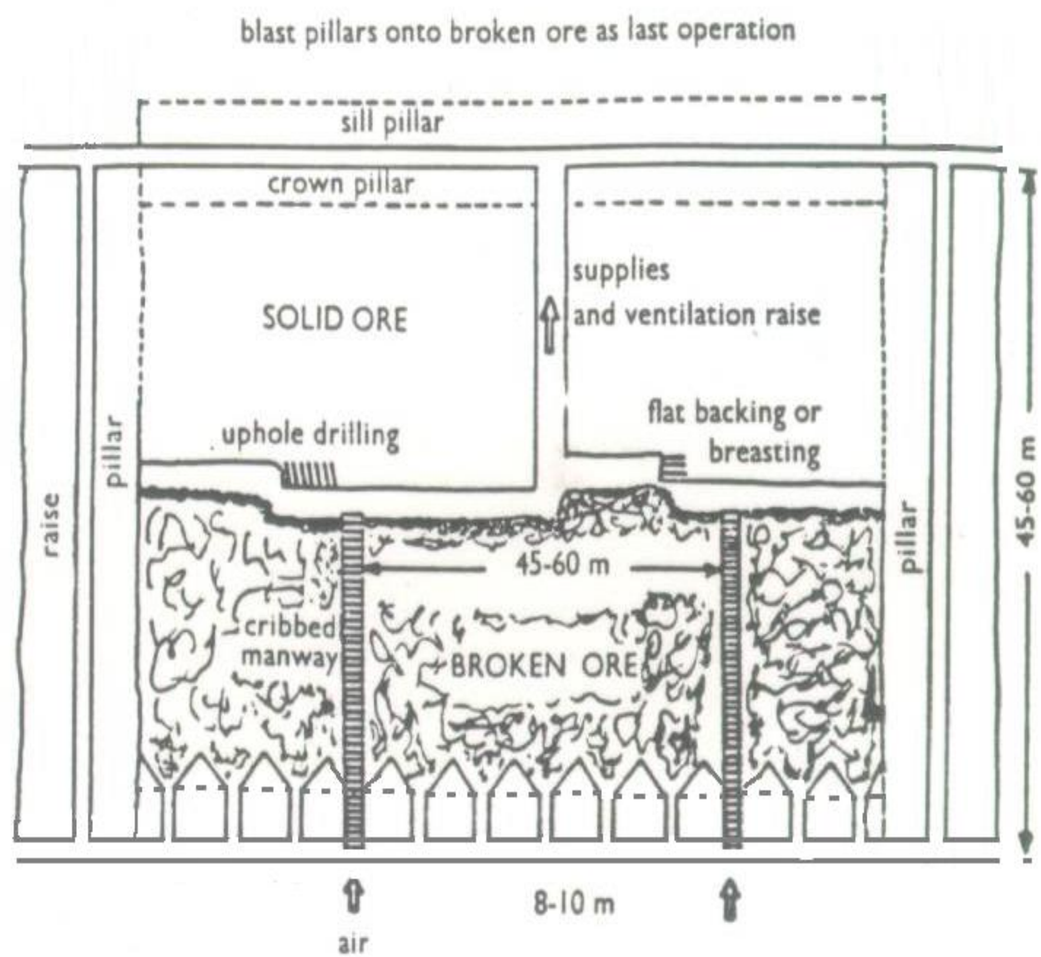
Shrinkage stoping

It is an overhand vertical stoping method in which the ore is mined in horizontal slices and remains in the in the stope as temporary support to the walls and to provide a platform for the miners. Since the ore swells in breakage, 30-40% of the broken ore in the stope must be drawn off ("shrunk") during mining to provide sufficient working space. The remainder of the broken ore is recovered when the entire stope block is stoped out. This holdback in production of 60-70% of the ore represents a significant tie-up of capital. On the other hand, it affords storage capacity and the opportunity for blending. The method is very simple but required skilled labour.



key design parameters in shrinkage stoping

The dimensions of the stope, largely governed by the shape and size of the deposit. In a relatively narrow ore body, the stopes are placed longitudinally; in a large or wide ore body, the stopes are placed transversely.

Stope widths vary from 1-30m, lengths from 45-90m and heights from 30-90m.

Although rock mechanics is a consideration in design, the stope opening itself is relatively small and not excessively stressed; and therefore the major concern is to maintain a manageable-sized stope that ensures a smooth flow of ore by gravity and effective draw control. Ores, especially those containing uranium minerals which exude radon gas, impose ventilation constraints on stope design. In most cases these problems can be minimized by limiting the size of stopes, by minimizing the duration of mining activity and by promptly drawing each stope empty following completion of mining.

Sequence of development

The nature of all vertical stoping methods is such that production operations are carried on all over a considerable vertical distance. Consequently several levels are required, the main or haulage production levels being spaced 60-180m. If the stope height is less than the level intervals, then sublevels may be constructed, connected by orepasses.

On each main level, a haulage drift is driven parallel to the strike of the ore body. It is connected to the shaft by a haulage crosscut. If the stopes are transverse or drawpoints offset, then a haulage lateral or loading crosscuts to the drawpoints are constructed. To provide multiple acess routes into each stope, but mainly to ensure through ventilation, raise manways are driven between levels, preferably at the ends of each stope.

The two main tasks of vertical stope preparation are to (1) construct a means of drawing ore in which muck flows by gravity to the bottom of the stope, and (2) undercut (horizontally) at the sill level or to slot (vertically) the stope, providing an opening into which the ore initially breaks and subsequently flows.

To construct the draw system and undercut, finger raises are driven at the desired spacing to connect the haulage level with the sill sublevel. The tops of the finger raises are connected by a small drift which runs the length of the stope; from it crosscuts are driven above the raises to the walls of the stope. To form bells, slabbing of the fingure raises begins at the top, diminishing toward the bottom of each raise, creating funnel-like openings. To form the undercut and provide working space for stoping (a desirable height of 1.8m), the pillars formed by the drift and crosscuts are slabbed off, the broken ore falling into the just-formed bells. Bells and finger raises terminate in chutes through which haulage conveyances are loaded directly. The spacing of chutes ranges from 5-15m.

Cycles of operation

Rock breakage is the principle activity in the stope itself. A bench face is established at the rib pillar and advanced across the face by horizontal drilling with airlegs or vertical drilling with stoppers. Using longhole drifter drills mounted on columns at the access raise, it is possible to drill out the entire stope length from one setup. After the holes are charged but prior to blasting, drawing of ore from the stope should occur. Before the cycle is repeated, any ground control is carried out, scaling followed by bolting (with wire mesh if necessary) or timbering.

Basic cycle of production operations

Drilling: pneumatic airleg, stopper, or drifter percussion drill, Dia – 29-33 mm, depth 2-2.5 m for stopper and 2-3.5m for drifter.

Blasting: ANFO or slurry, charging by hand (cartridge) or by pneumatic loader or pump (bulk), firing electrically or by detonating fuse.

Secondary blasting: pop shooting & plaster shooting, impact hammer

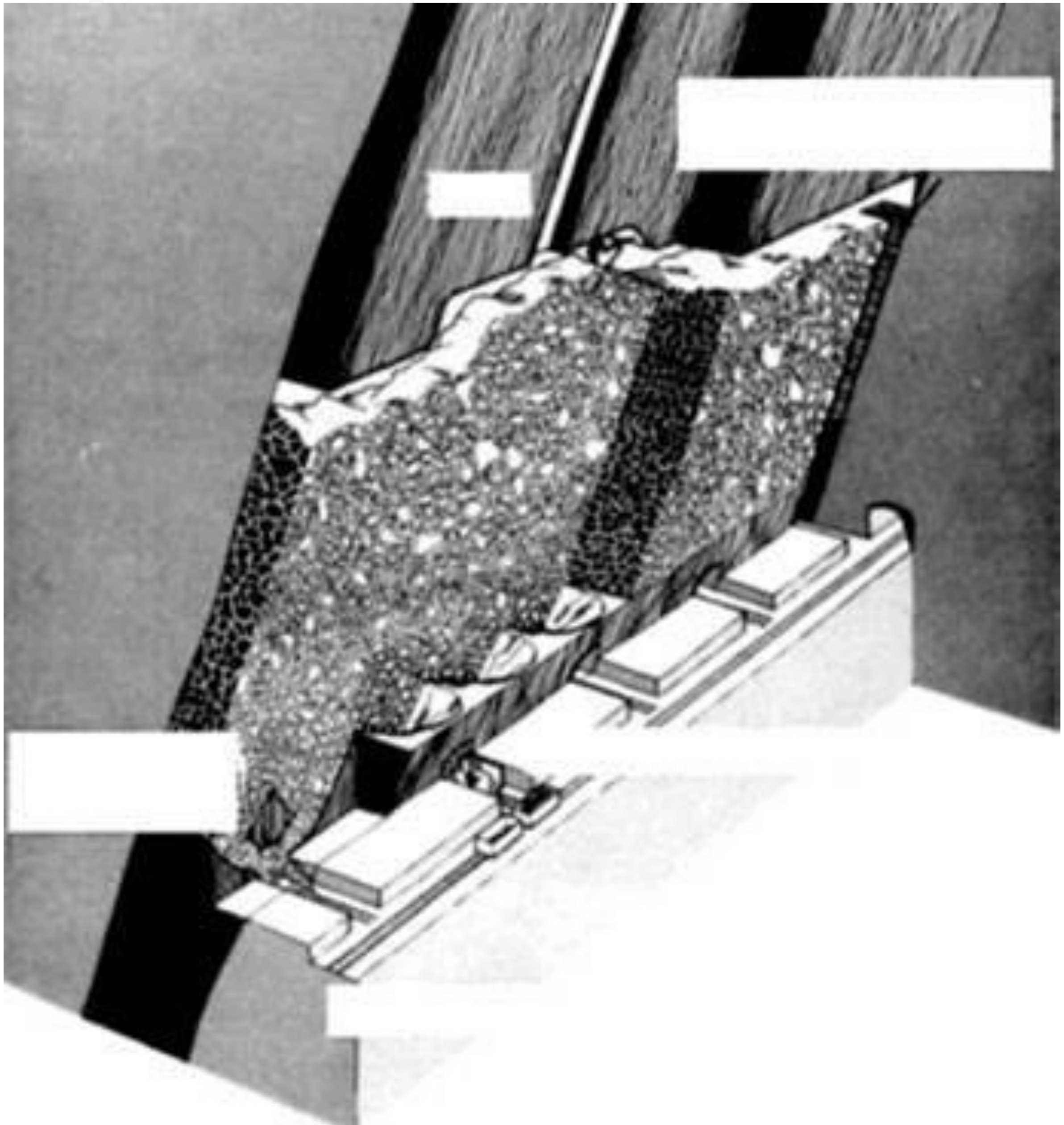
Loading: gravity flow (through stope), front-end loader, overhead loader, LHD, slusher, chute (under stope)

Haulage: truck, LHD, rail, belt conveyor.

Shrinkage stoping

Applicability Conditions

- *Ore strength:* strong (other characteristic: should not pack or stick together, i.e., free flowing, too much fine or clayey materials will hamper free flowing, oxidized or be subject to spontaneous combustion)
- *Rock strength:* strong to moderately strong
- *Deposit shape:* almost any shape but should have regular dip and boundaries
- *Deposit dip:* fairly steep (> 45-50^0 or angle of repose, preferably > 60^0), regularity along the dip is a pre-requisite of shrinkage stoping as there must be no serious obstruction to flow of ore.
- *Deposit size:* narrow to moderate width (1-30m) length minimum 15 m to fairly large extent
- *Ore grade:* moderate to fairly high
- *Ore uniformity:* uniform
- *Depth:* shallow to moderate (<750m)



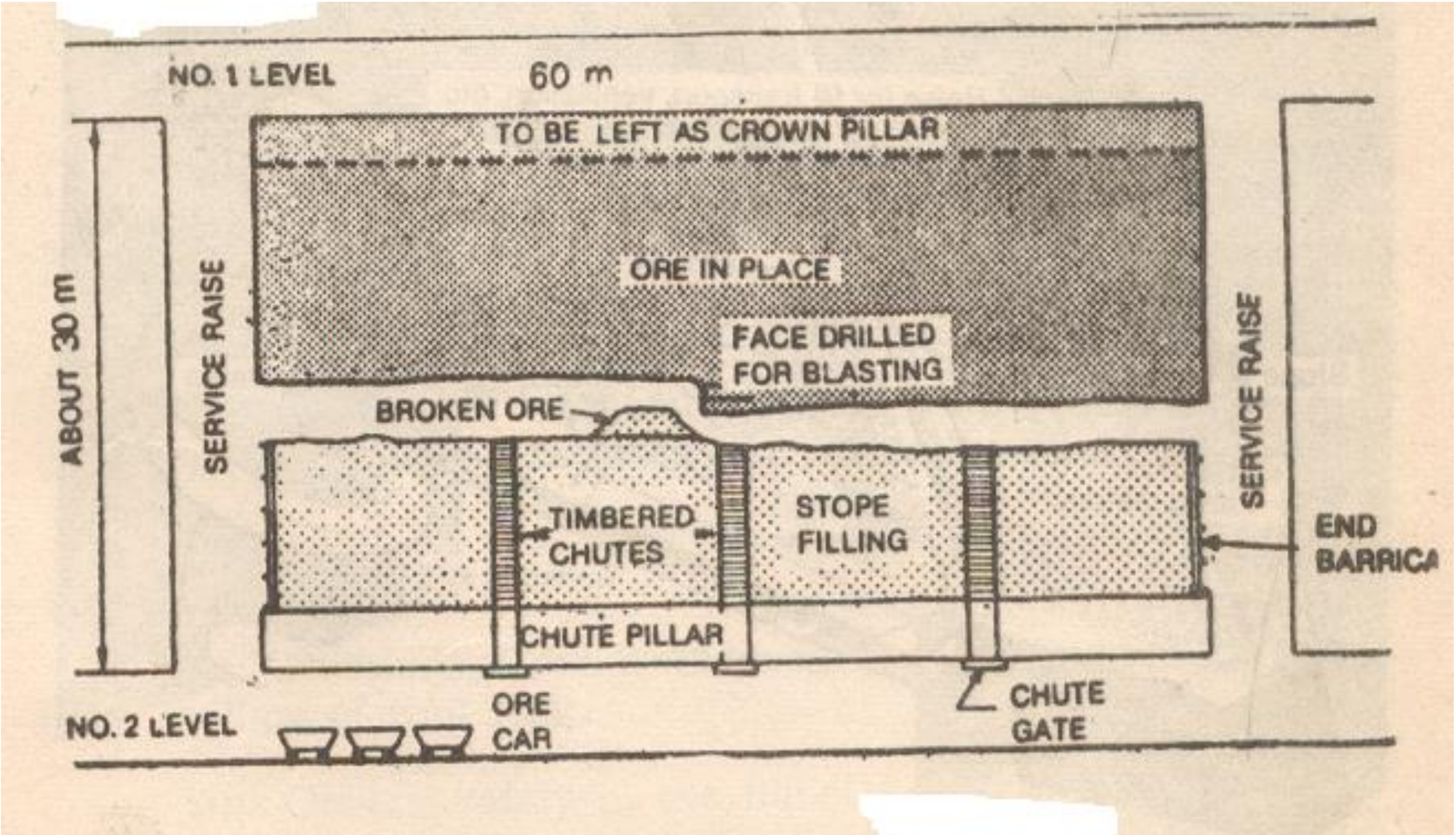
Advantages

- Ore drawn down in stope by gravity
- Method conceptually simple, can use for small mine
- Low capital investment, little equipment required for basic method, lend itself to some mechanization
- ground support in stope if any minimal .
- Stope development moderate.
- Fairly good recovery (75-85%), low dilution (<10%).
- Reasonable selectivity possible.

Disadvantages

- Low to moderate productivity (3-10 tons per employee-shift).
- Moderate to fairly high mining cost
- Labour- intensive, difficult to mechanize
- Rough footing, dangerous working conditions
- Majority of ore tie up in stope (> 60%) until completed
- Ore subject to oxidation, packing, and spontaneous combustion in stope.

Cut-and-fill stoping



Cut and fill stoping. Longitudinal section

The term *cut and fill stoping* - underground mining methods requiring support.

Support in stopes: local or short-term and general or long-term.

Local support is used for supporting the back of a stope, e.g., rock bolts or timber stulls. provides a safe working place for the stope miners.

General support for the walls of the stope - provided by backfill. preserves the access way to the stope and prevents large scale dilution of ore.

Cut and fill mining is primarily a cyclic stoping method. The cycle begins with the first round after backfilling on the previous cycle.

sub-cycles of the cut and fill cycle are:

- Drilling
- Blasting
- Mucking/slushing
- Ground support
- Cleanout
- Raising up
- Preparation for backfill
- Backfilling

Backfilling

Fill performs two unique functions:

- Ground support to the weak walls - compressibility is the most critical fill property, varying from 25% for mechanically placed dry fill to 5-10% for hydraulically or pneumatically placed fill.
- Working platform on which the next ore slice is drilled and blasted.

0.6 ton (approximately) of fill is required per ton of ore mined.

Deslimed mill tailings or waste sand from the surface is usually hydraulically slurried and piped underground for distribution in the stopes; sometimes, cement is added to produce a weak concrete fill, or natural-settling sulphide tailings are employed for greater fill strength. Hydraulic filling in cut – and - fill stoping requires special placement and drainage techniques since the slurry contains 30 - 40% water. All access openings into the stope from below have to be equipped with bulkheads to prevent flooding the haulage drift during filling. Sand barricades or fill fences - earthen dams or burlap-covered lagging on posts --- are erected in the stope to control the placement of fill. The tops of manways and orepasses must be extended above the fill floor to keep them open. To provide proper drainage of the fill while it sets, percolation drains are installed along the stope sill, and decantation towers are maintained through the fill; runoff water must be disposed of in the drainage system on the level below. Fresh fill withstands human traffic in few hours and vehicular traffic after 2 - 4 days. The timing of fill placement in cut and fill stoping is critical to the success of the method, since the fill must be place in time to assume some or all of the original superincumbent load on the ore in the stope.

Design parameters

- rock mechanics
- fill placement,
- mechanization factors (ease of access, maneuverability of equipment and production rate requirements).

Stope height -- 45-90m,

width -- 2-30m (limited mainly by rock mechanics and fill placement concerns).

Length -- 60-600m (function of mechanization).

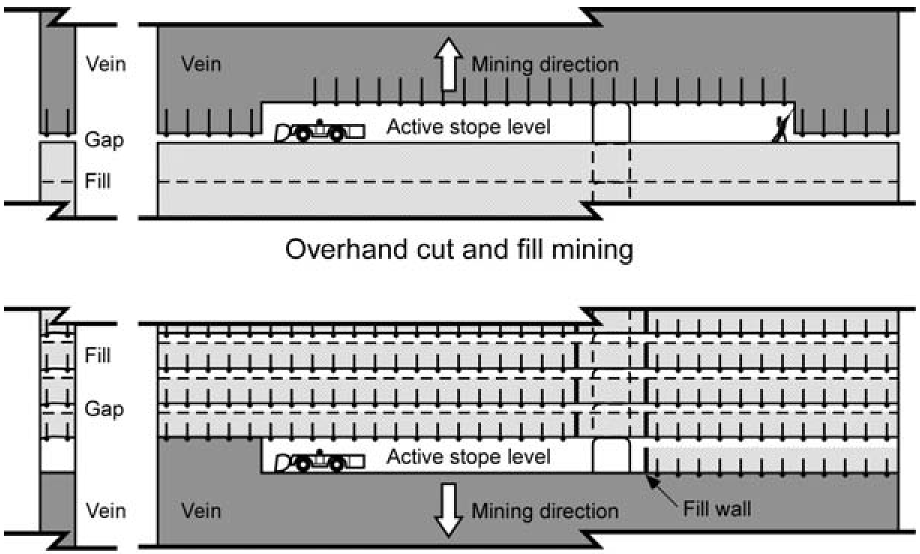
Thickness of each horizontal slice -- 2.4-3.6m (function of the drilling method used).

Orepases -- 1.8-2.4m square and spaced at 15-45m apart, depending upon the materials-handling equipment in the stope.

Variations of Cut and Fill Methods

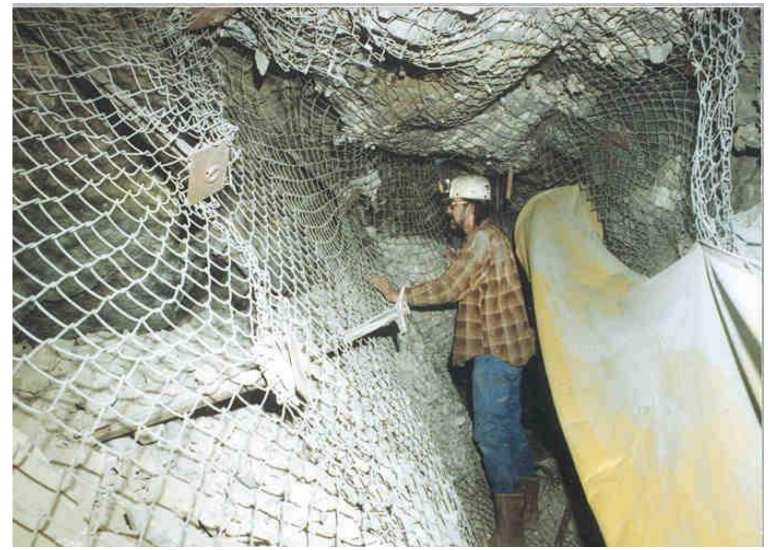
- OVERHAND CUT AND FILL STOPING:
- Stope advances upward - ore blocks are developed from the bottom.
- Horizontal cuts, 1.8 - 4.6 m in height.
- Mining advances away from an access point.
- Excavated ore falls and rests on the backfill placed during the previous cut and fill cycle. The back may be unsupported, rock bolted and timber stull back and rib.
- UNDER CUT AND FILL STOPING:
- Developed in Inco Ltd (Canada) in 1960s as a pillar recovery method to deal with abnormal grounds conditions. Ore strength can be weaker than in overhand cut-and-fill stoping. Ground is very weak or depth is so great that strong rocks have the potential to burst. Mining gold and silver veins at great depth or in very weak rock, operators need to minimize the amount of exposed ground.
- Mining is done in horizontal cuts (upto 4.5m thick slices).
- The direction of the stoping is downward. As each horizontal cut is mined, it is backfilled with cemented backfill (engineered mixture of waste rock, aggregates, and cement). Mining of the next cut proceeds beneath the cemented backfill placed against a timber fence along the sides and mat across the floor of the stope. Ore extracted underneath the fill can actually be made safer than the original host rock. Miners install supplemental support - rock bolts, wire mesh, shotcrete - to further protect themselves from the potential for falling ground.

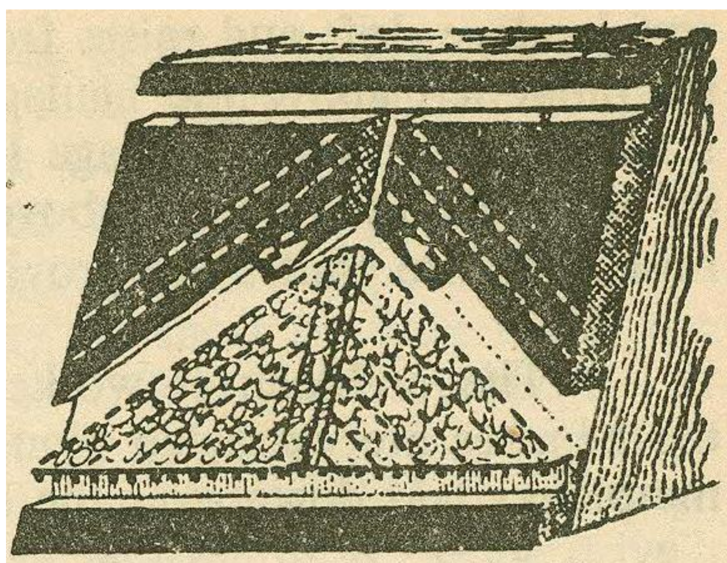
  Example: Lucky Friday silver mine in north Idaho
- Rill cut and fill stoping:
- Stope fill is inclined to facilitate the gravity flow of muck to the chutes.



Overhand cut and fill mining

Underhand cut and fill mining


Underhand cut-and-fill mining


Rill cut-and-fill mining

Excavation methods:

- *BACK STOPING (UPPERS).* When back stoping, enough open space is allowed between the backfill surface and the stope back to drill vertical or steeply inclined blastholes into the stope back in an upward direction. Back stoping can be done with rock-bolted or unsupported backs. Ribs should be competent to avoid dilution and safety problems.
- *BREASTING-DOWN (UP).* This excavation method employs a vertical working face and horizontal or slightly inclined blastholes. Backfill must be placed to nearly fill the void of the preceding cut, allowing only enough space (0.3 to 1 m) for expansion space for effective blasting. After each round is blasted, mucking and ground support activities are completed. Breasting is generally used in poor rock conditions often with timber support. In undercut and fill stoping, rounds are breasted upward instead of downward.
- *DRIFTING.* On the first cut or on any cut where breaking room is not provided, drift rounds are blasted using a burn cut or other type of drifting cut.
- *BENCHING.* Benching or the drilling of vertical holes from the top down can be done in undercut and fill stoping.

Development

One common method of developing narrow veins is to drive crosscuts from a lateral in the wall rock when poor ground conditions exist, or when the vein has poor continuity. The crosscuts are driven through the vein, and the raises are driven on the vein from the crosscuts. An initial drift is driven in the ore for the length of the stope, and it may be driven on the level or 6 - 8 m above the level. This serves as the undercut. Another common method is to drift on vein and not drive laterals.

- **Access**

  Access to the stopes is via raises or ramps in mines using LHD equipment. These raises are usually constructed of square or hexagonal wood crib, or of round steel liner plate. The manway and timberslide also serve as ventilation conduits. In mechanized stopes, ramps are driven in either the hanging wall or the footwall. The inclination of the ramps varies from 1 : 5 to 1 : 10. Short access inclines or declines are also driven into the stopes at regular intervals as the mining progresses.

- **Drilling**

  BREAST. non-mechanized - Jacklegs - 0.6-m starter changing to 1.2-m and 1.8-m to finish the hole. Mechanized - Rubber-tired or crawler-mounted two-boom jumbos - stopes.

  No. of holes - 8 for a narrow stope (1.07 m or less, in width) to 12 or more holes for wide stopes. Holes parallel to the walls in the vertical plane. Holes, except the top row parallel to the floor of the stopes in the horizontal plane The top row of holes should be inclined upward 2 to 3°.

  UPPERS. non-mechanized – stopers. Holes vertical to a depth of 3 to 10 ft (0.9 to 3.1 m). complete back of the stope is drilled out before blasting.

  Mechanized - Two- or three-boom rubber-tired-mounted jumbos or crawler-mounted drifters. holes may all be drilled vertical or at preset angle, and be 10 to 15 ft (3.1 to 4.6 m) in length. The important thing is to drill all of the holes parallel to each other, and to have them all end at the same height above the floor of the stope. If any ore is left on the periphery of the stope from the last cut mined, it is slabbed before the round is blasted.

- **Temporary ground support** - square sets, stull sets, stulls, and/or rock bolts.

Basic cycle of production operations

Mechanization of cut and fill stoping has led to the introduction of mobile drilling, loading, and haulage equipment in the stopes themselves, with corresponding improvement in production rate and productivity.

- *Continues mining:* roadheader in soft to medium hard rock (e.g., borate); in hard rock use conventional cycle.
- *Drilling:* pneumatic airleg, stopper, or drifter percussion drill, Dia – 29-33 mm, depth 2-2.5 m for stopper and 2-3.5m for drifter. Drill rig mounting pneumatically or hydraulic machines (percussion or rotary-percussion); hole size 51-76mm.
- *Blasting:* ANFO or slurry, charging by hand (cartridge) or by pneumatic loader or pump (bulk), firing electrically or by detonating fuse.
- *Secondary blasting:* (in stope) drill & blast, plaster shooting, impact hammer
- *Loading:* (in stope) LHD scraper to shute or orepass, gravity flow to drawpoints; (on level) LHD, front-end loader, shovel loader.
- *Haulage:* truck, LHD, rail.

Conditions

- *Ore strength:* moderate to strong, may be less competent than unsupported method.
- *Rock strength:* weak to fairly weak.
- *Deposit shape:* tabular, can be irregular, discontinuous.
- *Deposit dip:* moderate to fairly steep (> 45-50^0), can accommodate flatter deposit if orepasses are steeper than angle of repose.
- *Deposit size:* narrow to moderate width (2-30m), fairly large extent
- *Ore grade:* fairly high
- *Ore uniformity:* moderate, variable (can sort waste in stope).
- *Depth:* moderate to deep (<1.2 -2.4km).

Advantages

- moderate productivity (9-18 tonnes per employee-shift & 27-36 is max).
- permits good selectivity, sorting; can use waste as fill.
- moderate capital investment, adaptable to mechanization
- low development cost.
- versatile, flexible, and adaptable. excellent recovery if pillar mined (90-100%), low dilution (<10%).
- good safety record.

Disadvantages

- fairly high mining cost.
- Labour- intensive, requiring skilled miners and close supervision.
- filling complicates cycle, causing discontinuous production.
- compressibility of fill risks some ground settlement and instability.

Sublevel Stoping

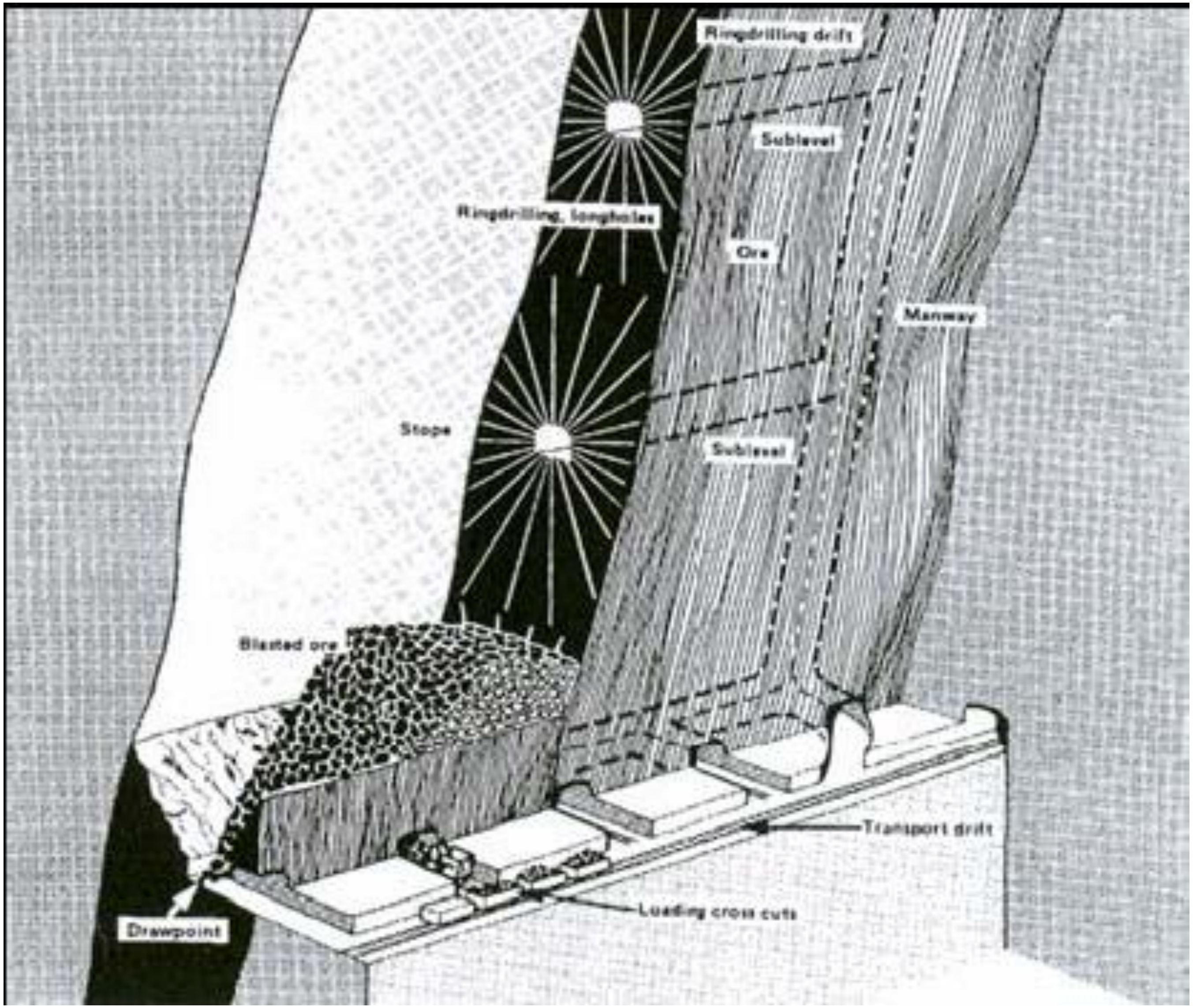


Figure – 1 Schematic illustration of ring drilling

- If geo-mechanics studies indicate that very high blocks (height exceeding the straight drilling length from one drill location) can be extracted using the same extraction level, then several drilling levels at various heights within the block must be created. Because of multiple drilling levels, or sublevels, this method is called sublevel stoping. It is the only patented mining method (more correctly, a modern version of the method, *vertical crater retreat*-VCR, is patented). Sublevel stoping, also known as blasthole, longhole, open or VCR stoping. It is an overhand, vertical stoping method utilizing longhole drilling and blasting carried out from sublevels to break the ore. It provides required geometrical form to create gravitational ore flow from end point of production sublevel drifts to drawpoints in open stopes.

- Method requires very less temporary support; in the sublevel, it is provided by rock bolts, wire mesh, cables, or shortcrete. Although the stopes are unsupported, pillars are usually left between stopes and occasionally within stopes. Method is very safe; no personnel are exposed in the stope: drill and blast crews work in the protective cover of sublevel drifts and crosscuts, while loading crews work under the security of the haulage drift below. Doesn’t have a limitation in depth and successful operation has been reported to a depth of approximately 900 m under the surface.

Stope layout

The orebody is divided vertically by driving crosscuts and haulage levels at every 45-120m. Stope width varies from 6-30m, maximum length 90m. Boundary pillars are located similarly to those in shrinkage stoping. Rock mechanics, blasting, and materials handling considerations are the major concerns in specifying dimensions. In addition, because of the unique reliance on longhole drilling and blasting, special attention in sublevel stoping must be paid to rock-breakage design: hole diameter and length, burden, explosive selection, powder factor, and so forth.

Sequence of development

The general sequence of development in sublevel stopping parallels that in shrinkage stoping and other vertical methods. A haulage drift, crosscut, and drawpoints or draw drifts and trench are driven for material handling, together with interlevel raises for access and ventilation. Either an undercut, or end slot is constructed to commence stoping operations. If an undercut, the sequence described in shrinkage stoping is employed. If a slot, sublevel crosscuts are driven across the stope from the sublevel drift and a raise driven at the boundary. These are then blasted in a manner to excavate a slot, starting at the bottom.

In the ring-drilling version of sublevel stoping, only sublevel drifts must be driven for longhole drilling. In parallel drilling and VCR methods, a horizontal slot must be opened across the orebody to provide room for the drill stations. This is constructed by driving a sublevel crosscut the width of the stope and then advancing it the length of the sublevel drift as mining progresses.

Pillars (sill, crown, and rib) may be delineated at the stope boundaries and left permanently or recovered in a retreat operation, often after back filling.

Basic cycle of production operations

Rock breakage and materials handling are carried out in separate sections of sublevel stopes. Drilling and blasting are conducted in the sublevel drifts, while the loading and haulage take place underneath the stope in the drawpoints or draw drifts. Coordination is necessary, of course, but the two major groups of unit operations of the production cycle are carried out largely independently of one another. The cycle of operations follows the basic production cycle.

Drilling: (1) longhole pneumatic percussion drill (small-hole) with coupled steel, drifter or fandrill-mounted. (2) downhole pneumatic percussion drill (large-hole) on drill platform or rig; or (3) roller-bit rotary drill (large hole) on drill platform or rig.

Blasting: ANFO or slurry, charging by hand (cartridge) or by pneumatic loader or pump (bulk), firing electrically or by detonating fuse; blasting by special longhole bench rounds for overhand large scale mining or by spherical charge in VCR method.

Secondary blasting: (in stope) drill & blast, plaster shooting, impact hammer

Loading: gravity flow to drawpoints; LHD, front-end loader, shovel loader, slusher.

Haulage: truck, LHD, rail.

Drilling systems

The main influential operation step to define production rate and economical result in a period of time could be associated to select type of drilling system.

Ring drilling and parallel drilling are two main drilling systems in sublevel stoping. Both employ longhole drilling, the ring pattern with small holes and the parallel drilling with large holes. In ring drilling, a vertical slot is opened; whereas either a slot or an undercut is used with parallel drilling (the VCR method requires parallel holes and an undercut).

In figure 1 a schematic illustration of ring drilling pattern has been demonstrated in an open stope. In this style of production drilling, blast holes are drilled on a ring pattern in ore body from the endpoint of each production sublevel drift to around the drift radially. Mechanized hydraulic Ring drill rig is the most fitting drilling equipment in this regard. Common diameter of blast holes in ring drilling system are between 50 – 75 mm with lengths up to 25 m. Longholes don’t generally exceed 30 m because hole deviation and manage turn into big problems. The performance of the drilling system in this respect is between 120 – 180 m in a shift. Also the production range of drilling and blasting in this case would be between 1.5 – 2.5 cubic meters ore per drilled meter (Gertsch and Bullock 1998). In each blasting 3 or 4 rows are blasted generally. Blast hole spacing is unlike in collars and ends but burden is regular.

Drilling systems

Parallel drilling system is the most recent developed drilling method in sublevel stoping which is possible to perform by mechanized airtrack drill rig with DTH hammer and high pressure. Extending of the endpoint of a production drift is the first stage to implement parallel drilling system. Production drift's sides are excavated right upto the walls, i.e., the width of the ore body. Blast holes diameter in parallel drilling is between 105 – 165 mm with lengths up to 90 m. The performance of the drilling system in this respect is about 50 m in a shift. Also ore production range of drilling and blasting is between 8–18 m^3/m hole length (figure 2). In this case blast holes are drilled in bottom of the production drifts downward to drawpoints. In general the inclination of blast holes equals the maximum dip of the ore body. Production drifts distance in a vertical alignment in order to implement this system is over 50 meters commonly. Excavation of one production drift at the top of the open stope is a typical design. In this case length of the blast holes is defined as the distance of bottom of a production drift to undercutting space.

Furthermore there are some other type of long hole drilling pattern which have created of combined parallel and ring drilling properties as underhand fan drilling by DTH jumbo-drill rigs. As a case in El Soldado mine underhand fan pattern has been implemented with blast holes' diameter 165 mm and length 80 m by DTH system (Contador and Glavic 2001). High pressure DTH hammers in parallel drilling system have the highest rate of drilling's accuracy. Inaccuracy of this equipment is less than 2% up to 120m hole length of the blast hole in general (Haycocks and Aelick 1992).

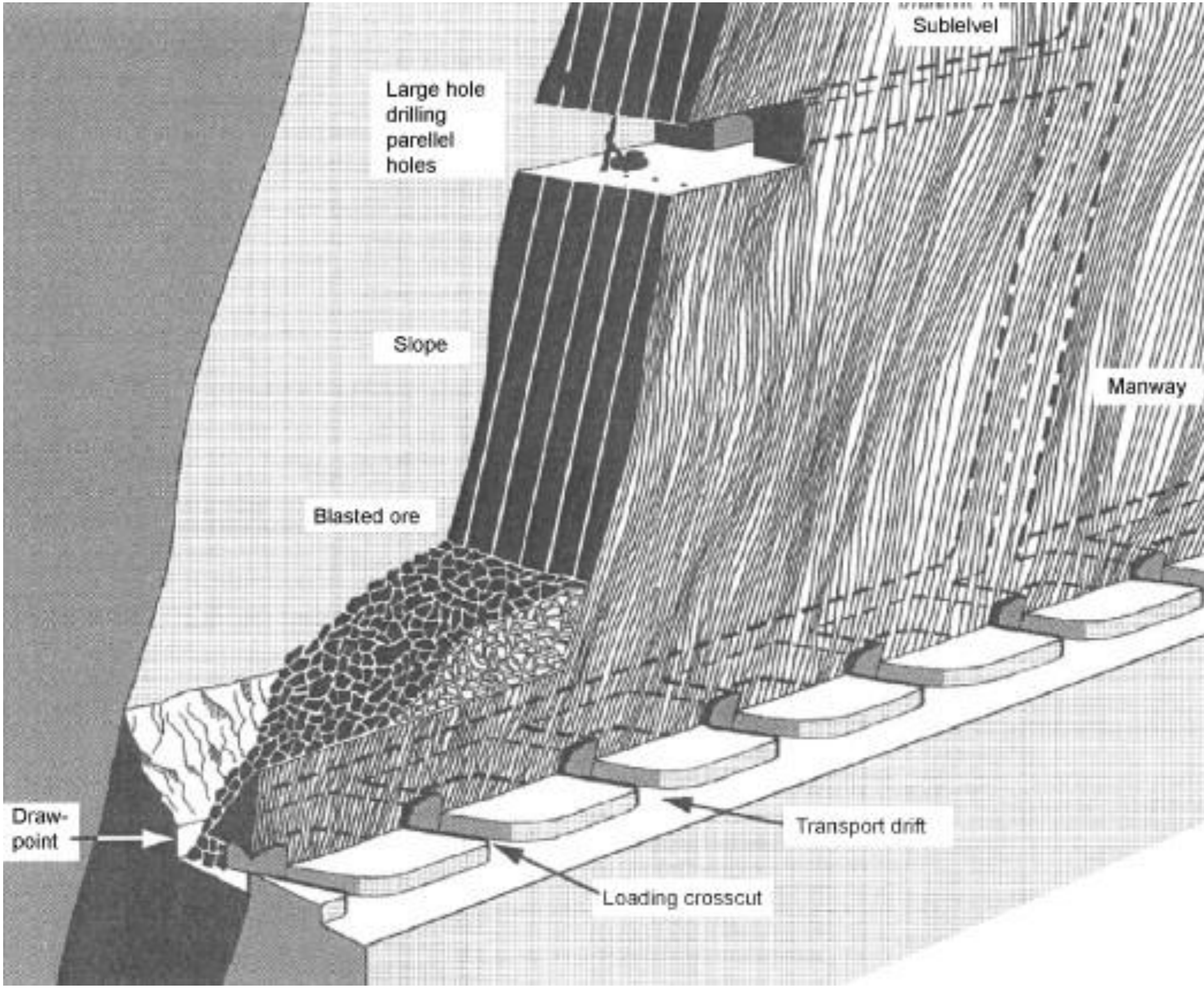


Figure-2 schematic illustration of parallel drilling

Conditions of applicability

- *Ore strength:* moderate to strong. ore has to be in a stable situation
- *Rock strength:* fairly strong to strong. competent hanging and foot wall rock. Lowest rate of essential compressive strength of the rock walls to apply sublevel stoping is 55 MPa normally.
- *Deposit shape:* tabular or lenticular, regular dip and boundaries.
- *Deposit dip:* fairly steep (> 45-50^0), preferably 60-90^0.
- *Deposit size:* fairly thick to moderate width (6-30m), fairly large extent.
- *Ore grade:* moderate.
- *Ore uniformity:* fairly uniform to uniform.
- *Depth:* moderate to deep (<1.2 -2.4km).

Advantages

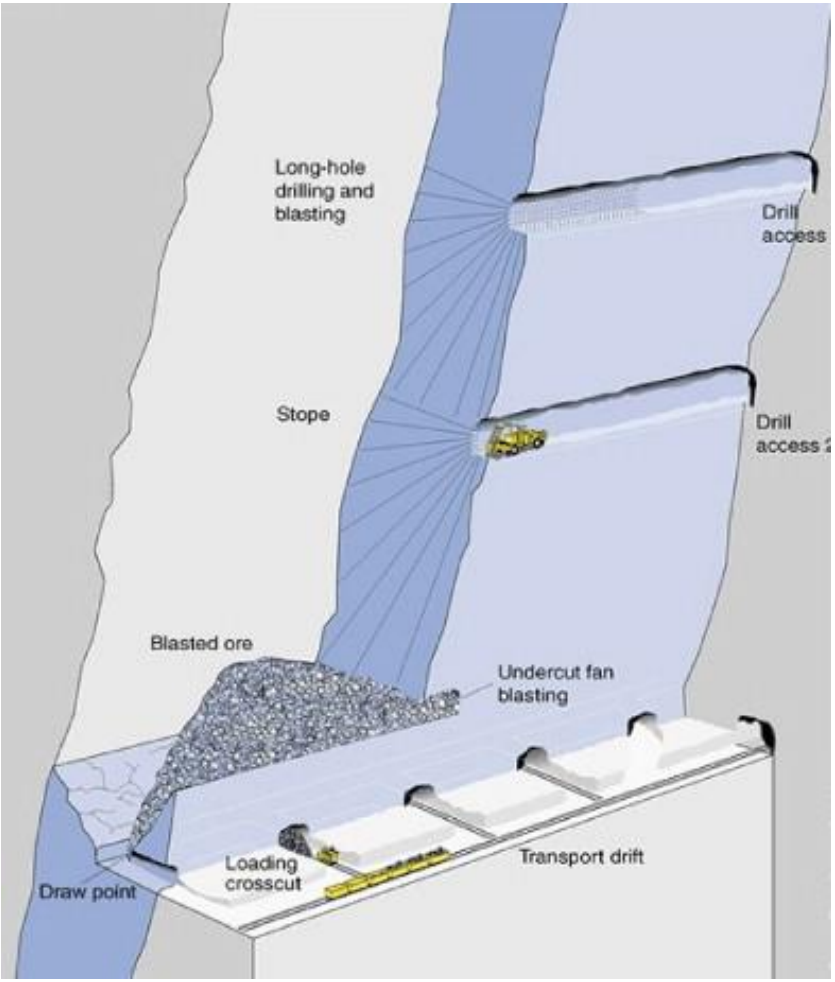
- moderate to high productivity (14-27 tonnes per employee-shift and 27-36 is maximum).
- moderate mining cost.
- moderate to high production rate, individual stope outputs running as high as 22700t/month.
- lends itself to mechanization; not labour-intensive.
- low breakage cost; fairly low handling cost.
- little exposure to unsafe conditions; easy to ventilate.
- unit operations can be carried on simultaneously.
- fair recovery (75%); can be in excess of 90% when good pillar recovery is possible.
- dilution is generally low and can be contained below 20%.

Disadvantages

- slow and complicated development, very capital intensive and high development cost.
- inflexible and nonselective.
- Longhole drilling requires careful alignment (<2% deviation).
- large blasts may cause excessive vibration, air blast, and structural damage.
- secondary blasting fumes may leak back into the stope if excessive secondary blasting is necessary.

Blasthole stoping

In blasthole stoping, from the drilling level at the top of the block, rows of radial blast holes are drilled down to the top of the extraction trough. A raise is driven at one end of the block and slashed to full stoping width to form a slot. The rows of blast holes are now blasted as one row or as several rows at a time toward the open slot. Hole diameter vary widely, but typically lie in the range of 76 -165 mm for wide blocks, 165mm in diameter holes are often used. Hole straightness is an important design consideration that affects fragmentation, ore loss, and dilution. In general, one would select the largest hole diameter possible for the stope geometry since straight hole length is strongly dependent on hole diameter. The specific development (amount of development required to exploit a certain volume of ore) is inversely proportional to block height. Since the cost of development is significantly higher than costs for stoping, one wants to have the highest possible number of extraction blocks associated with given extraction and a given drilling level.



Blasthole stoping

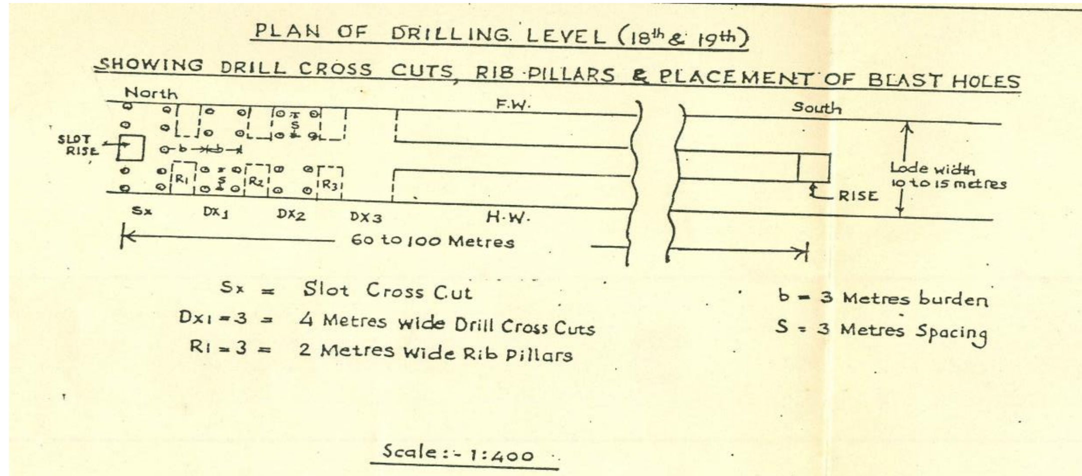
Large diameter blasthole stoping at Hutti Gold Mines

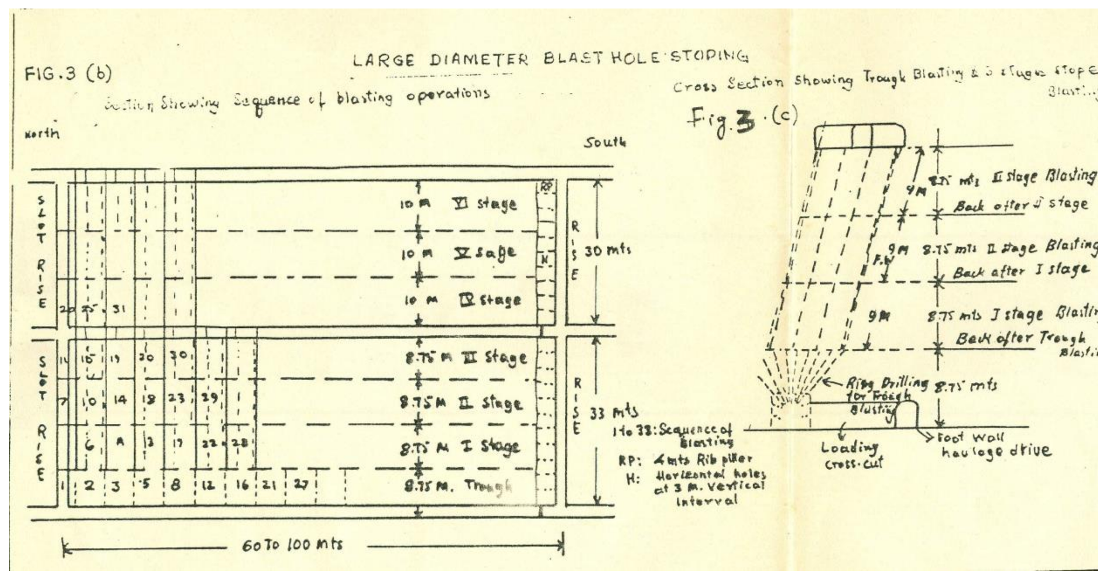
Blasthole diameter 152-165 mm. Cylindrical explosive charges are used.

Drilling is done from the main level or a sub-level 9 -10 m below the main level.

A 3 m wide slot is created along the full width and height of the stope. Drill cross-cuts are developed parallel to the slot and blasted into the opening one by one. Vertical slices of ore between the levels are blasted in stages into the voids created by previous blast.

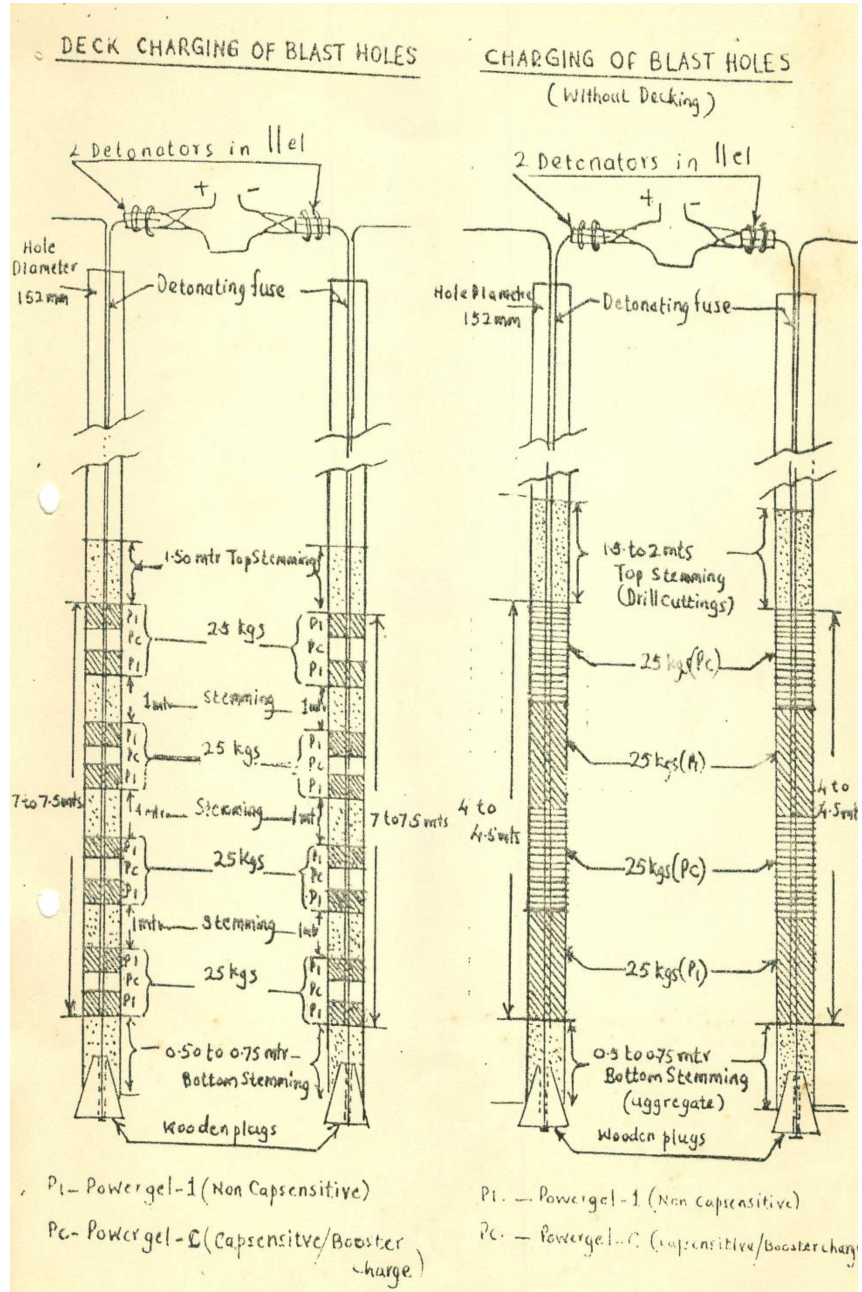
Blasted material is collected in a trough developed in the extraction level and drawn.





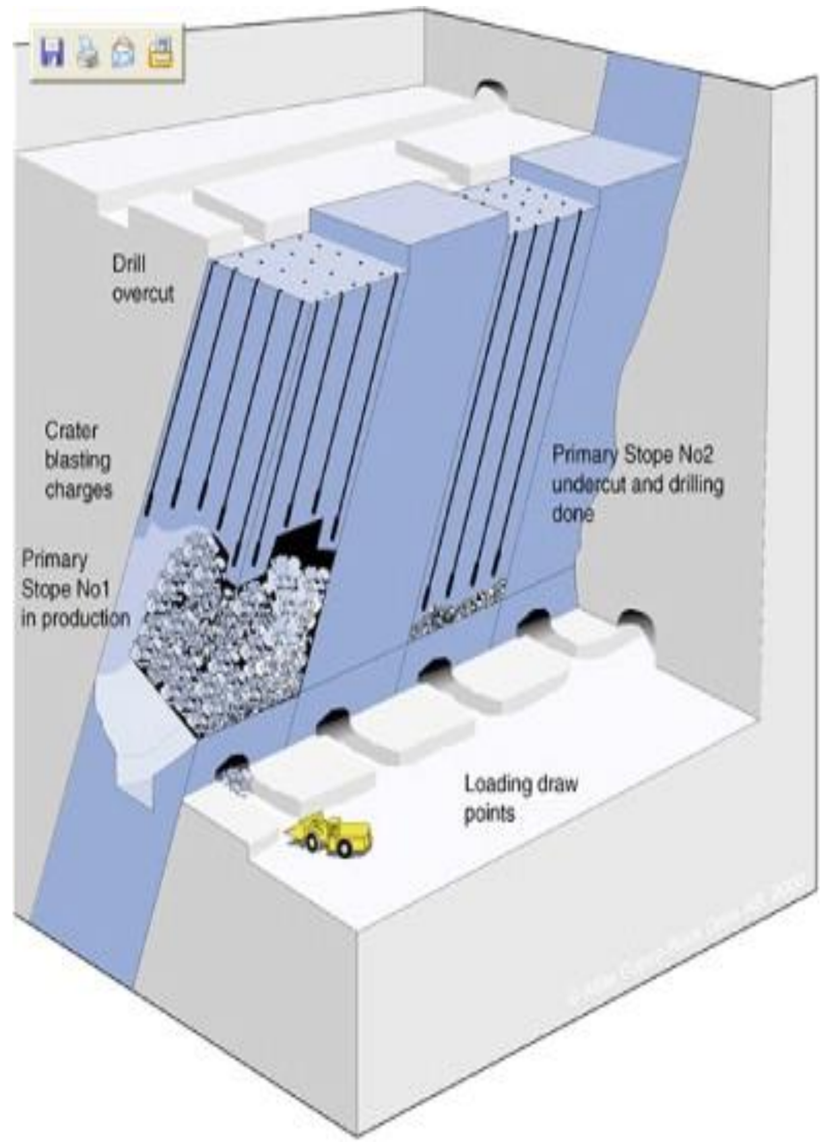
Design and operation details of LDBH stope

Strike length of the stope block – 50m
Height of the block – 60m
Dip of the orebody – 65 deg.
Average width of the stope block – 9m.
Footwall haulage drive, at the extraction level at a distance of 7m from the orebody.
Loading cross-cuts from the haulage drive at an interval of 10m.
A traugh drive and a traugh above in the orebody for the entire length of the block.
Drill cross-cuts of size 4m widths and 4m height in the drilling levels, leaving a pillar of 2m thick between the drill cross-cuts.
Drop raise at one end of the block from the drilling levels which will be developed into a slot.



VCR Stopping

One of the most recent methods to be adopted in underground metal mining the vertical crater retreat (VCR) mining which is now being employed in over a dozen of large and medium scale underground mines in different countries. The application of this new and revolutionary mining method has been possible only after down-the-hole (DTH) drills were introduced to underground mining operations around 1973. The method employs large diameter long holes of 152, 165 or 200 mm diameter and is based on the spherical charge technology (also known as crater-blast technology) which is used to produce a series of craters in a horizontal plane, as result of blasting.



Vertical Crater Retreat Method

Mechanics of creater formation

The method of **VCR** mining utilises concentrated or spherical charges. A charge is considered to be spherical if its length-to-diameter ratio does not exceed 6:1. Thus for a hole of 165mm diameter, a slurry package of 165mm diameter and 990mm length would form a spherical charges. The geometrical configuration of spherical charges limits its weight to approximately 35 kg in a 165 mm hole (Monahan, 1982). These spherical charges are placed in vertical or near-vertical parallel blast-holes at an optimum distance from the bottom of the hole (Fig. 1). The optimum distance (also called the depth of burial) is defined as the distance from the free surface to the centre of gravity of the charges and is so chosen that the maximum volume of rock is broken to an excellent fragmentation size. When the charges is detonated, it produces a crater (surface cavity) in the surround rock. As gravity works with the explosives breakage process and as the explosive energy of spherical charges is used at optimum confinement conditions, the resultant crater depths normally exceed the top of the explosives charges location and the muck produced is very well fragmented for an efficient handling.

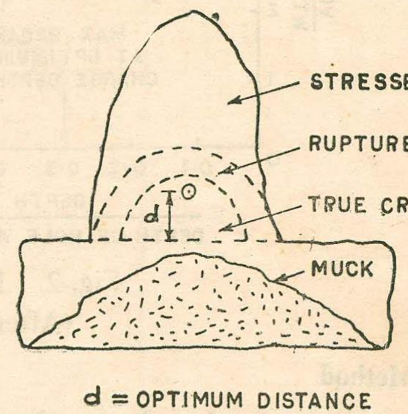


Fig. 1. Crater Formation in VCR Method (After Lang, 1982)

Crater Blasting

The concept of catering and its development may be attributed to C.W. Livingston whose of long and devoted work resulted in an excellent tool for studying and understanding the blasting phenomenon, and finally finding a practical application for it in under – ground mining.

A crater blast is a blast when a spherical charge is detonated beneath a surface that extends laterally in all direction beyond the point where the surrounding material would be affected by the blast.

In analyzing crater blasts, it has been found that there is a definite relation between the energy of the explosive and the volume of material that is affected by the blast, and this relationship is significantly affected by the placement of the charge.

Livingstone determined that a strain – energy relation exists, expressed by an empirical equation: $N = EW^{1/3}$

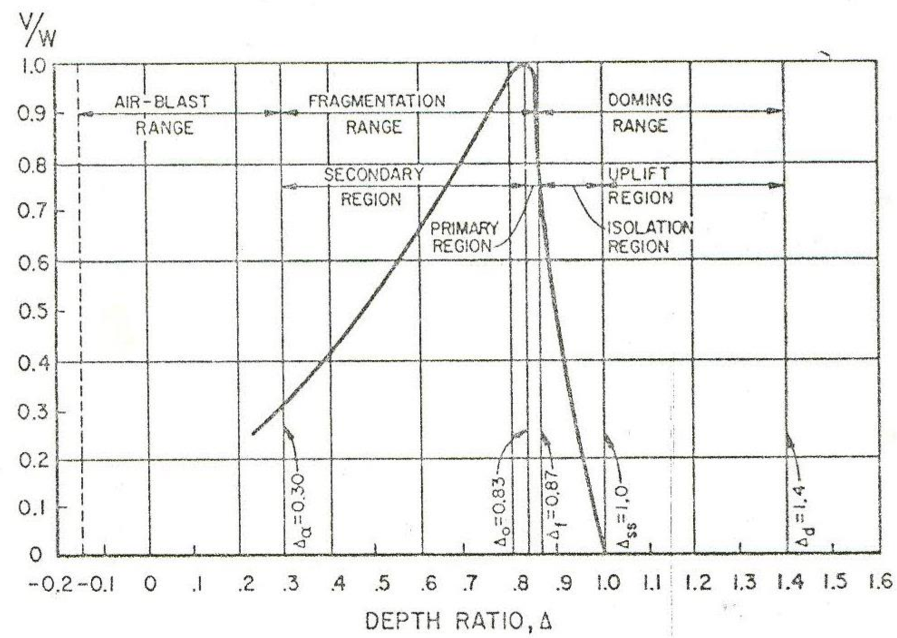
Where *N* is the critical distance at which breakage of the surface above the spherical charges does not exceed a specified limit, *E* is the stain energy factor, a constant for one given explosive- rock combination and *W* is the weight of the explosive charge.

The same equation may written in the form relation $d_b = \Delta\, EW^{1/3}$

Where d_b is the distance from the surface to the centre of gravity of the charge i.e. depth of burial and Δ equals d_b/N which is a dimensionless number expressing the ratio of any depth of burial compared to the critical distance. When d_b is such that the maximum volume of rock is broken to an excellent fragment size, this burial is called optimum distance (d_o)

Cratering Experiments

An actual cratering experiment consists of a number of cratering-type shots in which the type of explosive, its charge weight and the rock are constant and only the distance of burial varies. Critical distance is obtained by observation. All craters are excavated and their volume determined, Figure shows the plot of data obtained from a cratering test in which the depth ratios Δ are plotted against energy levels (volume/charges weight: V/W). This idealized curve of Livingston also shows the important transition limits from the isolation range to the other extreme which is the air-blast range. Once having this curve established, production – scale blasts can be designed



The explosive's contribution in blasting is to provide pressure. The forces generated by pressure acting over a borehole's surface area accomplish the necessary work to cause stress condition within the surrounding mass for fracture and displacement.

The explosion produces two distinct and separate pressures. The first is the detonation pressure developed as the detonation front passes through the explosive charge. The explosive's detonation velocity directly affects the magnitude of this pressure. The value of the detonation pressure is approximately proportional to the explosive's density and its detonation velocity squared. This pressure is applied at only a very shorts period of time against the surrounding mass at a given section of charge length. For a cylindrical explosive charge in a typical blasthole the detonation pressure would have only a minimal influence on the rock surrounding the borehole, and the detonation pressure would have its greatest effect only at the charge and opposite the point of initiation. One can conclude that the detonation pressure in a cylindrical borehole is not very effective to fragmentation.

The second pressure that quickly follows the first is the borehole pressure produced by the high temperature gases formed by the chemical reaction. The entire surface area of the borehole where the explosive is contained will be exposed to a sustained loading condition. It would be, therefore, the borehole pressure which dominates in the process of breaking the rock.

Dynamic loading by borehole pressure in a cylindrical hole is predominantly directed laterally or radially outward from the borehole axis, which little or no force being directed towards the charge ends.

The Breakage mechanism of a spherical charge is quite different. The forces produced by a spherical charge are directed radially outward from the centre in a uniform spherically diverging action in all planes passing through the centre. It follows that the entire surface area of the cavity confining the spherical charge receives all the detonation pressure and the borehole pressure.

While Livingston was using true spherical charges, it has been found that as long as the deviation from the true spherical (diameter =length) is not greater than 1:6 diameter to length ratio, the breakage mechanism and the result are practically the same as that of a true spherical charge.

Detonate spherical charges towards a free face such as the back of a mine opening is entirely new concept of cratering forming the basis of a new blasting technology leading a new underground mining method. Here crater is formed in a downward direction and are not affected by the adverse effects of gravity and friction. To the contrary gravity enlarges the crater dimensions by removing the entire rupture zone.

Once the equilibrium of the mass is disturbed by the excavation of an underground opening, a stressed zone of elliptical shape is formed above this opening. Depending on the stability of the rock the material within the stress zone caves in sooner or later unless properly supported. When a crater is blasted in the back of an underground opening this stress zone is enlarged, the material within suffers further damage and caves in following the detonation. Depending on rock properties and structural geology the total height of this cavity may exceed the optimum distance of the spherical charge from the back many time .

However, cratering characteristics are unique for each ore and crater design must be based on actual tested. Fig. 2 (Crocker, 1979) shows the result of a set of crater tested for a particular rock and explosive combination, depicting the occurrence of maximum break with good fragmentation at a particular optimum charges depth. Further, the drill pattern should be so designed that the blasted craters may overlap slightly in the horizontal plane.

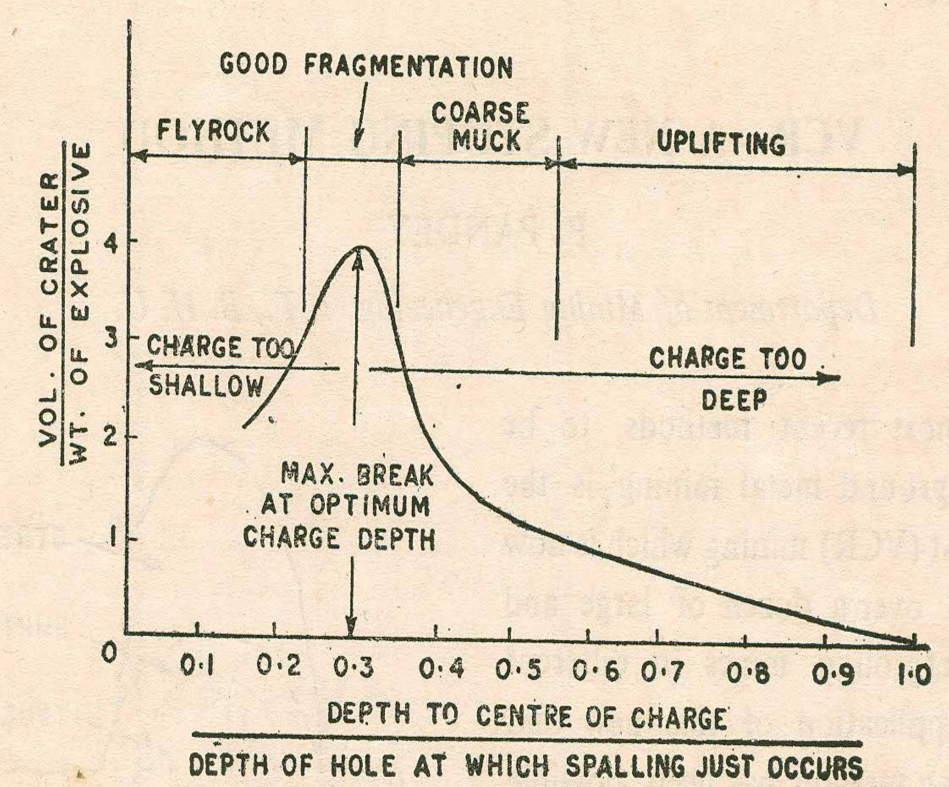


Fig. 2. Result of carter tests
(After Crocker, 1979)

VCR Stoping Method

The **VCR** method required large diameter holes usually of 165 mm diameter, to be drilled in a parallel pattern from a top drilling drive in the ore (called an overcut) down to an undercut on the level below .When the drill pattern has been completed over the whole stoping block, the bottom of each hole is blocked off and charged with ‘spherical’ slurry bags are placed in the hole at an optimum depth of burial. Horizontal slices of ore up to about 5 m thick are then blasted into the undercut (Anon, 1981). The ‘swell’ of broken ore is then drawn off (as in shrinkage stoping) from draw points by LHD equipment, prior to the next blast being taken. After each blast has been drawn off, the space between the top of the broken ore and the face of the stope is measured which forms the basis for determining the thickness of the next slice to be blasted.

Repeating this loading and basting procedure, mining of the stone or pillar retreats in the form of horizontal slices in a vertical upwards direction until the entire block is crate blasted. The VCR method necessitates the use of water jel or aluminized slurry explosive having high densities, high detonation velocities and high bulk strengths. ANFO, because of its low density has not been used in VCR blasting, despite its attractive cost and safety characteristics.

Several patterns of millisecond delays for blasting the slices of the ore body are used but the preferred method is to first blast a burn cut out of the centre of the pattern while the remaining holes are then blasted concentrically around the burn. This method gives each hole two free faces into which it can break, laterally into the burn and downwards into the horizontal stop back.

Diameter, Length and Inclination of Blast holes

- Most VCR mining has been done with hole diameters in the 152 or 165mm diameter range. Blast holes in the 200mm range have been successfully used and can give good production rates, although vibration considerations and the inability to execute larger drill patterns in narrow widths can be the limiting factors. The blast holes are usually about 50m long and the efficiency of the stoping method is largely dependent upon the drilling accuracy, since a poor configuration of holes produces a substandard blast. In this regard, the method works best where the ore body dips at angles greater than 70^0 since the static pressure of the drill string in an inclined hole is greatly reduced (Mitchell, 1980).
- Although some blast holes in excess of 90m length have been used, experience suggests that a reduction in depth to 75m or even 60m can result in lower overall mining costs because the higher development costs are then offset by improved results arising from greater drilling accuracy.

Advantages

VCR method has gained popularity both as a stooping method and for pillar extraction, in conditions where suitable ore blocks are available and the rock mechanics aspects are favourable. The VCR stopes have been used both as sublevel and shrinkage stopes. The method has also been used in drop raising. The main advantages of this method include :

- Higher tonnage per day and lower stoping cost.
- Lower development cost since it eliminates raise boring and slot cutting.
- Increased safety of operations because drilling and blasting are carried out from above and there is no need for the miner to enter the actual stope.
- Improvement in fragmentation (the method yields lowest powder factor).
- Reduced labour requirements and drilling and charging time.
- Elimination of up-hole drilling and up-hole loading of explosives.

Problems in Deep Underground Mines

Deep mining problem may be categorized into three groups:

- Problems related to the stability of structure in high stress environment at great depth.
- Problems due to excessive rock temperature
- Other associated problems in various mining activities due to large depth.

Problems at great depth due to high rock pressure

From geo-mechanics stand-point, great depth refers to distances below surface where creation of an opening results in an overstressed condition in the surrounding rock, resulting in failure or yield. Thus the depth at which a mine may be considered deep depends on the strength of the rock mass. In very strong, brittle rocks such as metaquartzites or granites; mines are not generally considered deep until their workings extend 1800m or more below surface. By contrast, coal and salt rock mines can be considered deep when they extend more than 450m below surface.

Geomechanics behaviour in deep mines: openings in “deep” mines in weak, ductile (or pseudo-ductile) rocks, such as salt rock, shale, or bituminous coal, are characterized by viscoplastic (or pseudo-viscoplastic) deformation of the surrounding rock. This behaviour serves to mitigate the effects of the high stresses. In strong, brittle rocks, however, strain energy resulting from excavation is not dissipated through viscous flow, but is stored in the rock until a limit is reached at which failure occurs in an explosive manner. Such explosive failures, which are called rockbursts, are small-scale seismic events – micro-earthquakes accompanied by emission of acoustic energy.

Control measures to avert high rock pressure

High rock pressure situation may be averted by proper planning and practicing right extraction methods that are as follows:

- Stop Planning: Planning of the stopping methods should be such that it can withstand the high rock pressure without endangering the stability and workmen.
- Speed of Stopping: Maintaining the extraction rate as fast as possible is a strategic measure for controlling rockbursts.
- Ground Control: Limited number of working faces, adoption of flat back stoping method with short strike lengths, orientation of mining faces at an angle of 20^0-35^0 to the major trend of rock fracture and avoidance of mining the faces towards each other are some of the strategic measures for ground control.
- Geological Conditions: Geological discontinuities are hindrance to the transfer of stress concentrations. Whenever the orebody is intersected by dykes and faults or other geological structures, the stoping should be commenced from the dykes/fault position and moved away from them instead of advancing towards them.
- Shaft and level linings: In high pressure environment movement between the hanging wall and the foot wall and rock displacement in general affect both the shafts and levels. Shafts may loose their verticality hindering faster hoisting operation.

Problem of excessive rock temperature

Environmental control in deep mines can constitute as much as 25% - 30% of the total mine working cost. High environmental temperatures coupled with heat and humidity can produce on workers many adverse physiological effects such as heat cramps, heat stroke, heat exhaustion and collapse. In order to create acceptable working conditions for men, heat must be removed from workings.

It has been found that in shallow mines about 1.6 m^3/s ventilation air per 1000t of rock broken per month and for deep mines, about 4 times this quantity is desirable. However, since the power required to circulate air increases with the cube of the quantity of ventilation air, there is an economic limit to ventilation, in the region of 4.2 m^3/s per 1000t rock broken per month. The economical depth limit is found to be around 2 km depth and the normal ventilation by circulation is found to be sufficient for rock temperatures up to 30^0C. When the rock temperature is about 40^0C, about 60% of the cooling requirements can still be provided by the normal ventilation air, the balance being provided by refrigeration (cooling service water). As rock temperature increases to about 45^0C, the ventilation air can provide only 30% of the cooling requirements, while 25% is provided by cooled service water and the balanced 45% by cooled ventilation air. At greater depths where rock temperatures are of the order of 55^0C, all cooling requirements must be provided by refrigeration. Of this, only 20% can be distributed by cooling the service water with a further 40% by cooling the ventilation air. The remaining 40% of cooling is provided by the successive re-cooling of the ventilation air as it passes through the mine.

Also it is essential to ensure that maximum production is achieved from minimum number of working areas to keep ventilation costs to a minimum. More recently, use of ice for cooling the underground workings, rather than the conventional chilled water circulation has been considered in South African deep gold mines.

Measures to reduce temperature

Chilled Water Cooling System: Chilled water at any temperature between 0^0C to 10^0C is produced by refrigeration plants situated on the surface and/or underground, and then distributed via an extensive pipe network to various parts of the mine for cooling purposes. A 300 – 400 mm dia insulated shaft line delivers chilled water down the shaft.

Underground spot coolers and cold water chillers: Spot coolers are self-contained small spot refrigeration units and are generally used for localized or secondary cooling on jobs where temperatures are exceptionally high. Spot coolers reduced the temperature of air locally by allowing hot air to evaporate refrigerant directly.

Use of ice for cooling the underground workings: Small ice crystals (60-70 mm square and 8-10mm thick) in slurry are conveyed pneumatically for long distances in underground. The air used in pneumatic conveying is cooled to 1^0C to minimize melting of the ice during transport to shaft bottom. It is estimated that for cooling a very deep mine, about 2 t of ice/t of rock hoisted, will be necessary.

Backfill and recirculation: Installation of backfill gives rise to a substantial reduction in the rock surface area which emits heat. The environment is further improved by better ventilation flow, since the presence of backfill prevents the unwanted passage of air through back areas where it picks up heat.

Recirculation of a portion of the ventilating air in the underground workings can provide an alternative to increased flow rates of downcast intake air. Advantages include smaller mine fans, saving in airways (including reduction in the number of additional ventilation shafts), reduced air leakage and more effective refrigeration system.

Other associated problems due to great depth

Hoisting problems: Hoisting from great depths is often complex, time consuming, difficult and therefore expensive on the whole. Maintenance of deep shafts with expensive hoisting equipments itself an important, parameter for mine safety. Great depths require a system of primary, secondary and tertiary drops.

Surveying: Great depths makes surveying operations very difficult, time consuming and complex. High precision instruments are necessary to do the normal traverses. Correlation survey is tedious and problematic.

Dust control: This is another problem at great depths. One cannot simply spray water to control dust as it causes enormous humidity due to high rock temperature thereby making the workers life difficult. Ice blocks are at times used to cool, the water at the face which at times is used for drilling.

It may be mentioned that combating all these problems of excessive rock temperature makes the over all mining operation expensive. Hence such precautions can only be practiced when the mineral to be extracted is either a priority item or can pay for its high cost of production. However, the national policies of any country dictate the terms for exploiting such deposits that involves high investment in capital equipment to overcome the problems of high rock pressure, rock temperature and hoisting from great depths.