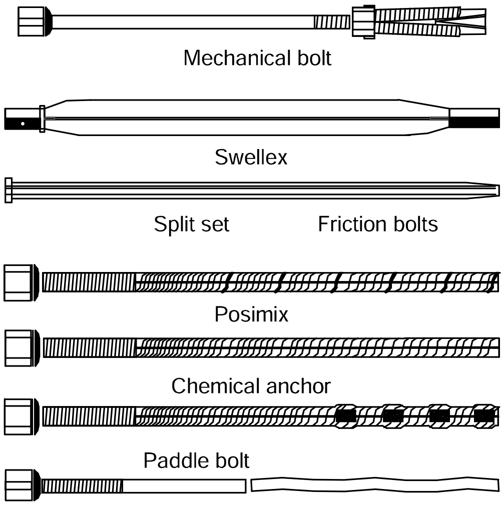

Figure 8.7 Rock bolts – some prevalent designs.[9b]

Resin grouted bolts:[4] Where high and quick strength is required, the resin bolts (fig. 8.6(e), 8.7) although costlier, find applications. In this practice a ribbed reinforcing rod is cemented into the drillhole by a polyester resin, which in few minutes changes into a thick liquid to a high strength solid by a process of catalysts-initiated polymerization.

In comparison to cement grouts, resin has the advantages of its quick setting and reaching the full strength quickly i.e. within 2–4 hours. The bond strength is much stronger than the cement grout. Complete grouting combined with tensioning can be achieved by inserting several slow setting cartridges behind the fast ones. If the resin starts to set before installation is complete, the bolt is left sticking out of the hole and is practically ineffective.

Wire mesh: This is also known as screen. This is available in different wire gauge thickness and mess apertures. Its main purpose is to support the rock between bolts, which is particularly necessary when the rock is closely jointed and the bolts are moderately to widely spaced. It can also serve as reinforcement for shotcrete.

In Figures 8.8(a) to (d), application of split set dome plates, split sets (rock bolts), sheet mesh and cables (strand Graford) at Mount Isa Mines for good and poor ground conditions, used for the long-term and short-term accesses, have been illustrated. These hard rocks mines have attained a depth up to 1.8 km.

8.3.2.4 *Concrete supports*

High compressive strength, easy to erect and manufacture, fire resistant, smooth finished surface and suitable under the adverse mining conditions including presence of abnormal make of water, are some of the advantages a concrete support commands over the steel and wooden supports. Low tensile strength, failure without warning and requirement of curing time for its setting are some of its limitations. Concrete finds its application for the following types of mine supports:

- Shaft lining
- Mine roadways lining

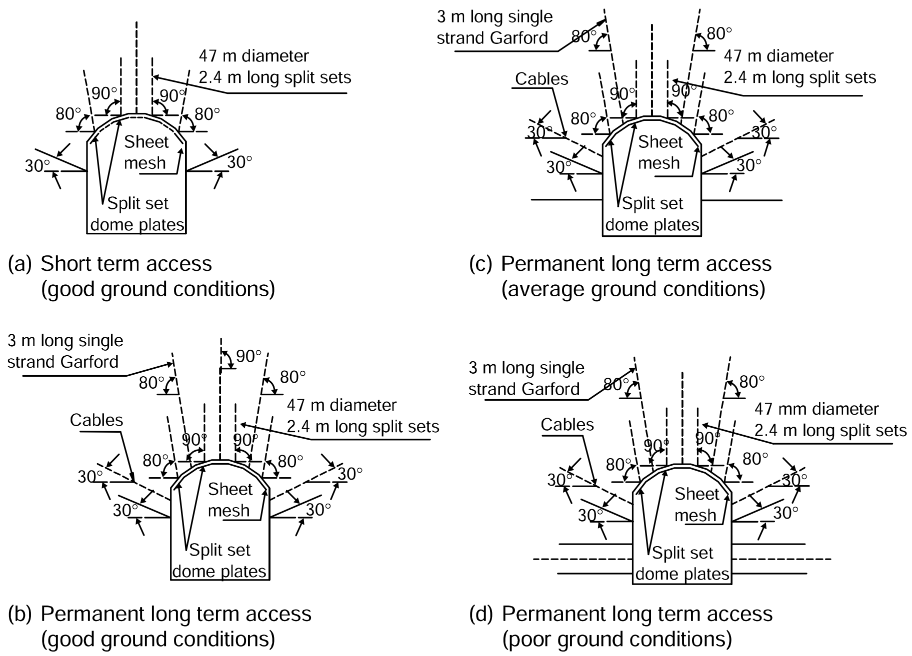

Figure 8.8 Application of split set dome plates, split set (rock bolts), sheet mesh and cables (strand Graford) at Mount Isa Mines for good and poor ground conditions prevalent for the long term and short term accesses Excavation size 4 m × 4.7 m. Max. bolt spacing = 1.2 m; Ring spacing = 1.4–1.5 m. Cable bolt ring spacing 2.5 m; Tension 3–5 tons. Sheet mesh = (100 mm × 100 mm × 5 mm). Courtesy: Mount Isa Mines, Australia.

- Concrete arches (fig. 9.20(c))
- As shotcrete or gunite.

Placement of concrete is achieved in the following manner:

- Monolithic concrete
- Concrete blocks
- Shotcreting or guniting.

Monolithic concrete (cast-in-place): A monolithic concrete is mass concrete instead the concrete with blocks. This involves placing a 40–60 cm (Cemal et al. 1983) wall around an opening's roof and sides. In order to cast it, first the shuttering/folds are built and then concrete mixture which could be in the ratio of 1:2:4 to 1:1:2 (cement:sand: coarse aggregates), depending upon the strength required. It is allowed to cure for 2–4 weeks. It can be further strengthened by incorporating steel tie rods, straps, angles etc., then it is known as reinforced concrete (R.C.C.). Monolithic concrete finds application is shaft lining, filling the gap between the steel arches and the ground (sides and back of mine roadways/openings). This concrete lining is very widely used in permanent workings such as shaft, chambers, and pit bottom openings including the shaft insets.

Shaft lining:[3,4] use of monolithic concrete is made to prepare the shaft lining. Mathematical relations used to determine the thickness (t) of concrete lining in the shafts have been worked out by some of the authors as outlined below:

According to Protodjaknov:

$$t = [(Pr)/\{(\sigma_b/F) - P\}] + \{150/(\sigma_b/F)\} \quad (8.9a)$$

$$t = \{0.007\sqrt{(2rH)}\} + 14 \quad (8.9b)$$

$$P = \text{Horizontal stress to the shaft lining}$$
$$P = (\gamma H)/(m-1) \quad (8.9c)$$

According to Brinkhaus:

$$t = (2r/10) + 12 \quad (8.10)$$

r = radius of shaft in cm.
According to Heber:

$$t = [\sqrt{\{(\sigma_b/F)/(\sigma_b/F) - 2p\}} - 1]\ r \qquad \text{if depth} < 400\ m \quad (8.11a)$$

$$t = [\sqrt{\{(\sigma_b/F)/(\sigma_b/F) - \sqrt{3}p\}} - 1]\ r \qquad \text{if depth} > 400\ m \quad (8.11b)$$

Whereas: t = thickness of lining in cm.
P = side pressure on lining in kg/cm^2.
σ_b = strength of concrete (after 28 days) in kg/cm^2.
F = factor of safety, usually 2.
H = depth of shaft in cm.
γ = rock density in kg/cm^3.
m = Poisson's ratio.

Using these relations,[3,4] given the following data, calculate the thickness of concrete lining.
Depth of shaft = 300 m; Radius of the shaft = 2.5 m; Factor of safety = 2;
Formation = sandstone; Poisson's ratio = 5; Density = 2.5 t/m^3.
Concrete details: compressive strength = 225 kg/m^2; density = 2.4 t/m^3

Concrete blocks: Sometimes using the concrete blocks arches or sets are built. In order to provide yieldability to them, the wooden pieces/blocks are inserted while carry out the masonry work to build these sets.

Shotcreting and guniting:[2,8] Mortar or concrete conveyed through a hose and pneumatically projected with high velocity on a surface, is known as shotcreting. Similar to this is guniting in which only cement mortar is applied and it contains no coarse aggregate. It has limited applications due to its higher cost and less effectiveness. There are two techniques that are used to apply shotcrete: 1 – Dry mix process – The mixture of cement and damp sand is conveyed through a delivery hose to a nozzle where the remainder of mixing water is added; 2 – Wet mix process: all the ingredients are mixed before they enter the delivery hose. A dry process equipment has components such as a gun, compressor, hoses, nozzles and water pump (in some designs), whereas in a wet mixer there is a mixing chamber to which compressed air is fed from the mains or through the separate supply.

It requires a skilled operator. It is applied in a limited space; as such the working atmosphere becomes tedious and requires good ventilation and illumination. For its application the surface should be clean and free from dripping water or the running water. Safety types of couplings secured with chains should be used to avoid any accident in the event of improper fastening of couplings.

1. General use: It finds its application in shafts, adits, haulage-ways and service chambers to acts as:
   - Primary support
   - Final lining

- Protective covering for excavated surfaces that are altered when exposed to air.
- Protective covering for steel or wooden supports, rock bolt plates etc.
- As lagging material in place of timber, steel or concrete in between steel or wooden supports.

2. Use as rock sealant: This application of shotcrete can prevent slacking of shale or other rocks and thereby their weathering can be prevented.
3. Use as safety measure: When an opening is required to be rock bolted, the shotcreting can be applied before and after the rock bolts are installed. When blocky ground tends to produce fragments, it may be applied with wire-mess. The shotcreting is usually applied after mesh is bolted to the surface.
4. Use as structural support: 1 – Non-reinforced with the thickness in the range of 2–4″ (52–104 mm.) 2 – Reinforced with fiberglass, wires etc. to the same thickness.
5. For repairing stonework, brickwork and concrete lining. In making the bulkheads airtight. Some times as a temporary support during shaft sinking.

8.3.2.5 *Support by filling*[7]

In effectiveness next to natural pillars is the use of back-fills as support. It has almost 100% ability to support the superincumbent load without yielding. The concept of filling, as described in section 16.3.2, is to pack a worked out area with a fill, which could be waste rock, mill tailings, sand etc. The method has got applications while mining all types of ores. It is a reliable means of support but costlier than most of other means but allows almost a full recovery of ores, decrease in the use of other type of supports such as timber etc., and, improve fire safety and ventilation. Degree of packing depends upon the type of fill used. There are two main classes of fill – *Dry & Wet*.

The wet fill is mainly referred to a *hydraulic fill* that is mixture of filling material with water. After mixing slurry is obtained which is transported through pipes to the void to be filled. In *hardening type of hydraulic fill* together with water, sand or mill tailings some hardening ingredient is mixed which allows cementing of the filled massif. In pneumatic *fill* dry filling material is moved via pipes by compressed air. *The ores suitable to work as support temporarily should not cake, ignite or oxidize in its loose state.* Given below is a comparison made to cover the important features of these back-filling practices.

8.4 SELECTION OF SUPPORT

Support is very vital for the mines to exploit the ores safely and for civil tunnel to drive and maintain them safely for their day-to-day use. Its improper selection may not only jeopardize the safety of the mine, but its overall productivity, economy and recoveries. Experience and skill of the mineworkers and supervisors play an important role to judiciously select it. Unwanted supports increase costs and inefficiency, and inadequate support network can cause ground stability problems, thereby, creating unsafe conditions. Supports are needed at all the stages of mining i.e. during development, stoping and post stoping (in some cases). Influence of supports on the drivage costs of different kinds of development entries has been illustrated in the figure 16.34(a).[9b] Costliest are the stoping practices, as shown in figure 16.34(b),[9b] which require use of artificial supports in the form of timber sets or backfills.

Proper stability in a mine can be achieved by following some of the guidelines outlined below. These measures can save the support costs considerably.

8.4.1 MEASURES TO PRESERVE THE STABILITY OF THE STOPED OUT WORKINGS OR TO MINIMIZE PROBLEMS OF GROUND STABILITY[1,12]

- *Limiting exposure size (i.e. volume or span)*: An excessive exposure of roof and wall leads to partial or mass caving of the roof and walls, development of excessive rock pressure, destruction/damage of supports and need for their restore, ore losses and its dilution, unsafe working conditions and accidents. Therefore, keeping a proper span of the worked out area is the primary measure to retain the stability of the workings. The permissible exposure is achieved by allowing hanging wall to cave in some stoping practices. Arching of the roof of the room (the drive or working area) increases the stability of the working area.
- *Reduction in duration of stoping/exposure time of the excavated area*: The rock strength decreases with time; pressure from the surrounding solid makes the rock crack; and previously latent crack opens and penetrates into the rock. In addition, the exposed rock when weathered it gets weaken. Low advance rate/intensity of stoping operations increases expenses on support, leads to ore losses and dilution and reduction in the safety and productivity.
- *Direction of stoping*: The direction of joints, planes of weakening or lamination in the ore should be taken into account while selecting the stoping direction to facilitate breaking and ensure stability of the roof. If exposed surface of the rock is parallel to joints or laminations, the roof will exfoliate readily; cave, and high strength support will be needed. Thus, roof stability can be improved by changing the direction of stoping.
- *Reduction in seismic effects due to blasting*: Larger the blasting charges (explosive), higher is its seismic effect on the surrounding solid rock mass. This causes cracking of the roof and wall rocks. This is the reason that blasthole's diameter and length are reduced at deeper levels.

8.5 EFFECT OF ORE EXTRACTION UPON DISPLACEMENT OF COUNTRY ROCK AND SURFACE[1]

Magnitude and nature of pressure depends upon number of parameters that can be termed as model parameters and design parameters. The *model parameters* include depth of working; deposit size, shape and orientation (dip); physical and mechanical properties of the enclosing rocks and ores to be mined. These parameters can be determined based on their natural occurrence. The *Design parameters* include: method of support, face advance rate, method of rock fragmentation (dislodging or breaking), duration of working a particular block etc.

As the mine deepens, the stoping methods and ore winning procedures need change. Deeper levels warrant minimum hanging wall exposure, discontinuation of stoping methods (that were applied at shallow depth) and large sized pillars.

But beyond certain depth, say, 700–800 m,[1] the support by pillars becomes impossible due to problems of rock bursts. This is due to the fact that at these pillars due to continuous working in and around them, a heavy concentration of stresses is usually noticed/experienced. The outburst of these pillars is accompanied by ejection of ore

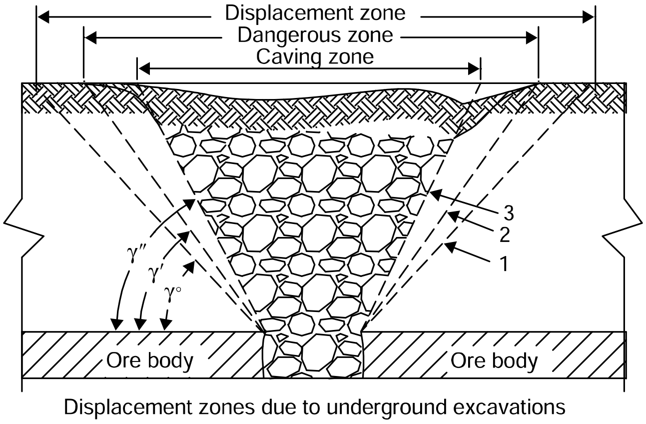

Figure 8.9 Ground disturbance above the excavation area influencing the above lying strata right up to the surface datum.

lumps from face and walls of the workings. In strong rock the pillars' failure may be sudden and severe resulting into the shock waves.

Rock displacement:[1] The void created by extraction of ore is get filled with caving rock in due course of time resulting deformation and subsidence of the overlying strata. This is called displacement of the rock. Displacement causes a slow and smooth subsidence of earth's surface without rapture, or abrupt subsidence with considerable movements, caving and collapses.

The rock displacement zone includes a caving zone, as shown in figure 8.9. The earth's surface, which experience displacement, is called a *'trough'*. This trough area covers subsidence over 10 mm.[1] Rock displaces along a curvilinear surface, but for graphical representation (fig. 8.9) they are assumed to be planes forming with horizontal Boundary angle γ_0, Displacement angle γ' and Caving (fault) angle γ''. The boundary angle defines the entire area of rock displacement. The caving (fault) plane extends through the extreme outer cracks on the earth's surface; the displacement angle determines the zone of dangerous displacement for the surface and underground engineering structures. The value of displacement angle[1] depends upon the physical and mechanical properties of rock, nature of rock, water permeability, angle of dip of the deposit and mining depth. For massive rocks it ranges from 45 to 70°, whereas for laminated rocks they are between 30 to 65°. The surface structures to be protected should be positioned outside the displacement zone else a protective pillar below them should be left.

Safety factor:[1] The mining depth at which stoping of ore does not cause earth's surface displacement is called the safe mining depth. The ratio of minimum safe depth to deposit thickness is called safety factor. The safety factor depends upon the physical and mechanical properties of the rock and it is about 200 for mining without filling, 80 for complete dry filling and 30 for mining with wet filling.

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