# CHAPTER 3.
# LITERATURE SURVEY

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## 3.1 General

Roof strata control is one of the critical components of underground coal mining. Without proper support design, the roof strata may be destabilized, resulting in disastrous consequences to the health and safety of employees and significant financial loss due to production downtime. So, to overcome these issues, we have to have a proper support design in underground coal mining.

According to statistics, bolting is now one of the most dominant support methods in both mining and civil engineering (Agioutantis, et al., 2001).

It can be used in various geological conditions in an economical manner. It can be combined with other support measures such as shotcrete, steel props, or concrete lining to create a more competent structure. Rock bolts can be installed simply and quickly and in a fully mechanized manner.

## 3.2 Bord & Pillar Mining

In underground coal mining, there are, basically, two methods that are widely accepted globally are Bord & Pillar and Longwall method.

In the U.K, former USSR, France, China, and other European countries longwall method of coal mining is the main method of mining. In India, about 98% of the underground output of coal is obtained by Bord and Pillar method, and the rest are from longwall methods.

The geo-mining characteristics of seam in Indian conditionsare following.

- The occurrences of upper seams at many places are at a low depth of cover.
- The shape of deposit and presence of faults prevent planning for long panels.

- The extensive variation in the thickness of seams leaves chances of coal remaining non- extracted within a panel.

Longwall mining is not considered as a feasible method in the above geo-technical conditions.

However, Bord and Pillar (room and pillar) method may be considered in that case. It is commonly used for flat or gently dipping bedded coal seams. The coal is extracted from the seam in two stages, namely development and depillaring. Pillars are formed during the development stage by driving the galleries. Pillars are extracted in depillaring stage. Depillaring operation generally started from dip to rise of the seam.

Method of extraction can be divided into two categories; semi-mechanised using SDL/LHD operation and mechanised using continuous miner operation. In SDL/ LHD operation, blasting off solid is adopted at a face. It is used for loading and hauling the coal into the transfer point.

Depillaring can be done by stowing and caving method. In the depillaring with stowing method goaf is filled up with incombustible material such as sand to reduce the subsidence. Whereas, in the depillaring with caving method roof is allow to cave into the goaf area.

Now further, Bord & Pillar mining are classified into two parts based on mechanisation.

### 3.2.1 Conventional method

In a conventional approach or semi-mechanised, machines that is used to develop the panels is SDL/LHD. The panel would be made, and coal is being extracted by the development and depillaring stage. In the development stage, the pillar is formed by extracting the gallery. Depillaring started from dip to rise after the creation of dip side main dip heading. Blasting off solid has been suggested in a face, and coal would be put into pony belt conveyors by SDL/LHD.

### 3.2.2 Mass production technology using continuous miner

Five heading panel is generally being adopted considering optimum and maximum utilisation of the machine. The distribution of the workload in five heads panel would be as follows:

i. One heading for Continuous Miner operation
ii. One heading ready for cutting
iii. One heading under support
iv. One heading cleaning by LHD
v. One heading having ventilation and direction line extended

The continuous miner cuts for a specific distance without support called cut-off distance. The cut-off distance will be required to be fixed after conducting physio-mechanical tests in the specific seams. In general, the cut-off distance is usually between 7 to 10 m. after the completion of the cutting of the face, the cutter moves to the next supported face, and the bolter will move in for support.(Refer figure 3.1)

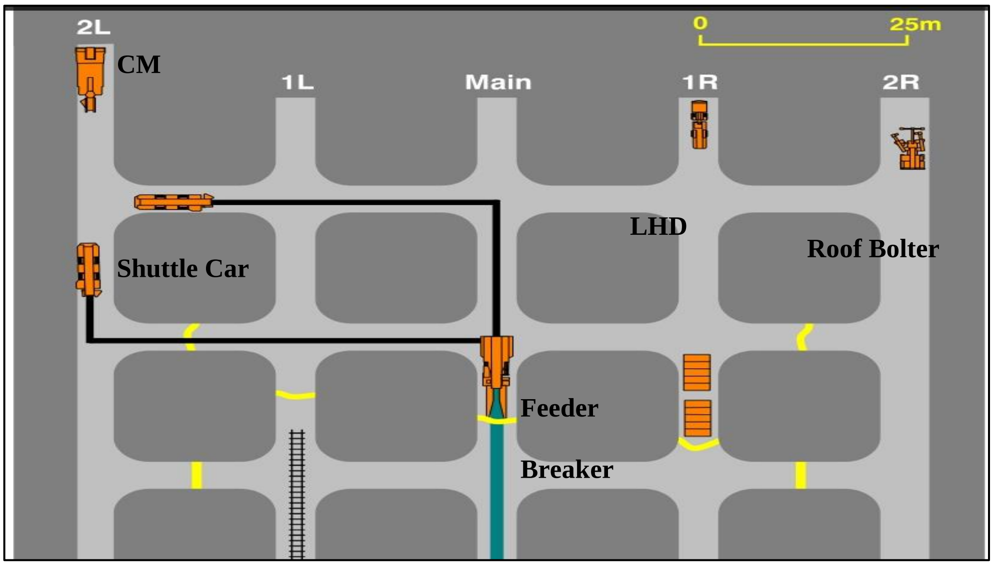


Figure 3.1 Typical Layout Development Panel of Continuous miner working

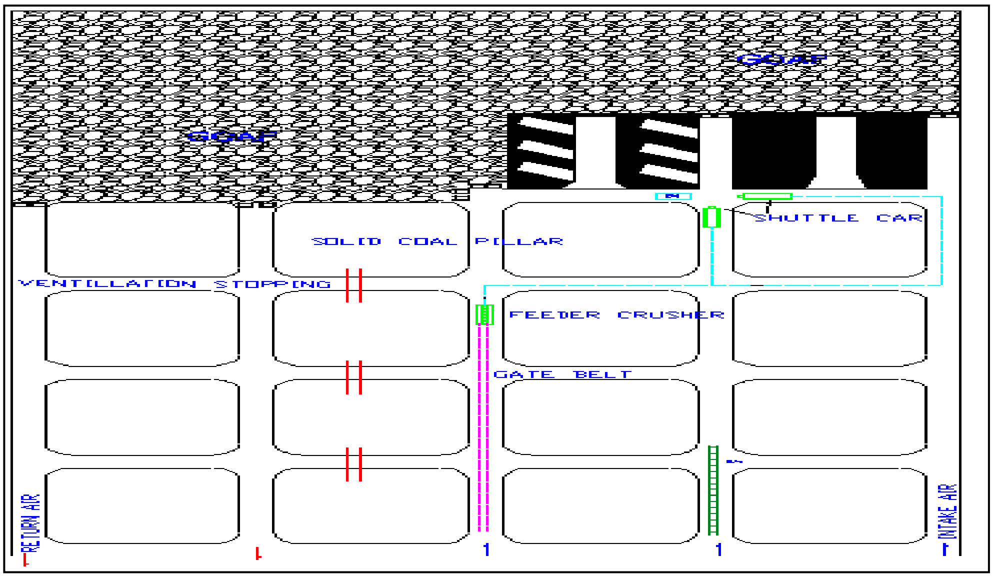


Figure 3.2 Typical Layout of Depillaring Panel of Continuous miner working

## 3.3 Mechanised Depillaring in Indian Coal Mines

A mechanised approach provides relatively faster extraction rate than conventional approach. It also improves safety and productivity during depillaring operation. Therefore, the coal mining industries of the country have introduced a number of MD operations in last 17 years. These depillaring operations in different coalfields encountered wide variation in site conditions ensuring mixed outputs (Mishra et al., 2013 and Oldroyd et al., 2006).

In February 2003, first MD operation (Leeming, 2003) in India was introduced at Anjan Hill mine South Eastern Coalfield Limited (SECL). Continuous miner technology has been adopted successfully in various Indian coalfields since 2003. Table 3.1 shows the mechanized depillaring approaches adopted by different mines having depth range varies from 60 m to 377 m. the nature of overlying strata is varied from easily cavable roof of Pinoura mines (SECL) to massive and strong strata of VK-7 mine of SCCL.

Table 3.1 Different mechanized depillaring faces using CM in India

| Name of mine | Geo-mining parameters of different mechanized depillaring faces | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | **Depth cover (m)** | **Pillar size (corner to corner) m** | **Bord width (m)** | **Height of extraction (m)** | **Overlaying strata** | **Manner of extraction** | **Cutting pattern adopted** | **Performance** |
| **Pinoura (SECL)** | 60 | 18.5x19.5 | 6.5 | 3.0 | Easy to moderate | Single pass | Fish- tail | Success |
| **Anjan Hill (SECL)** | 85 | 28.2x28.2 | 6.6 | 4.5 | Moderate to difficult | Splitting and slicing | Split and fender | Success |
| **Jhanjra (ECL)** | 125 | 26.0x26.0 | 6.0 | 4.2 | Moderate to difficult | Splitting and slicing | Split and Fender | Delayed caving |
| **VK7 (SCCL)** | 377 | 40.0x40.0 | 5.0 | 4.6 | Extremely difficult | Splitting and slicing | Split and Fender | Roof collapse |
| **Tandsi (WCL)** | 260 | 40.0x40.0 | 5.0 | 3.0 | Unstable roof strata | Splitting and slicing | Split and Fender | Partial extraction |
| **GDK 11 (SCCL)** | 325 | 48.0x46.0 | 6.0 | 4.6-6.0 | Difficult | Splitting and slicing | Split and Fender | Success |

## 3.4 Manner of Pillar Extraction

In India, three practiced MD methods are adopted, namely Fish - Tail (Mark and Zelanko, 2001), Pocket and fender (Singh et al.., 2011b and Leeming, 2003), and Remnant pillar (Oldroyd et al., 2006), as shown in figure 3.3. The Chrismas tree method (also known as the fish-tail method) is generally adapted from small to moderate size pillars of 18 to 24 m (Chase et al., 1997) under the shallow depth of cover. The pocket and fender method is adopted for relatively bigger size pillars (Mark et al., 2002), where the pillar is first split into fenders, and then each fender is extracted by slicing. The remanent pillar method is practiced to control excessive caving of the roof strata at a higher depth of cover (Oldroyd et al., 2006). Here at least one-quarter of every alternate pillar is left in - situ as a remnant to arrest the strata movement. This method is adopted by Tandsi mine, WCL, where immediate strata are weak, watery, and have high horizontal stress conditions.

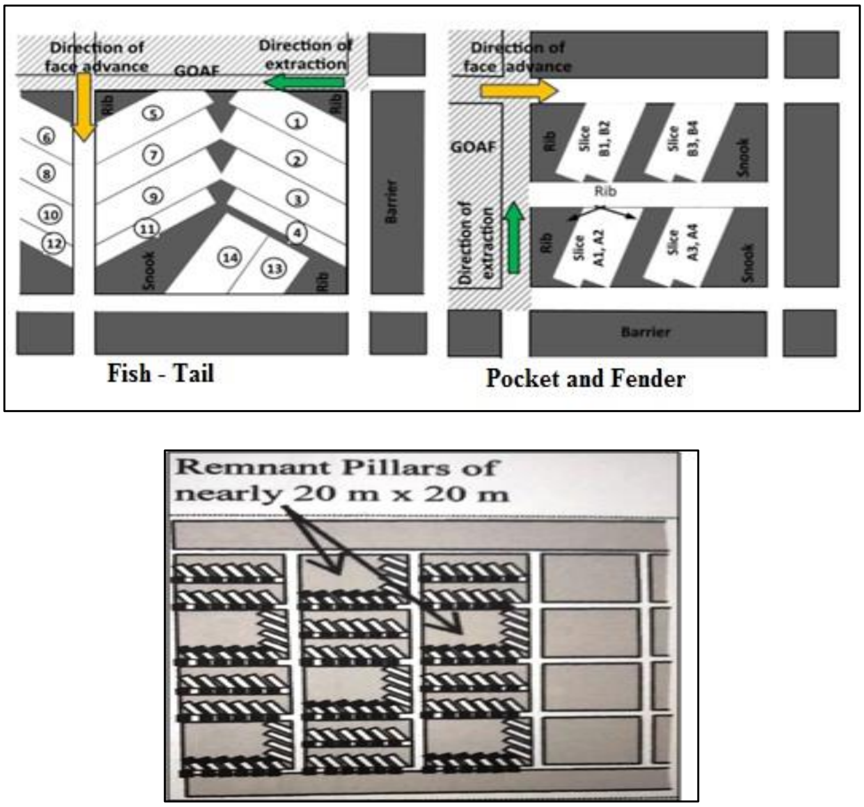


Figure 3.3 Manner of pillar extraction in MD operation in India

## 3.5 Factors affecting the integrity of mine structures

### 3.5.1 General

Rock structures are unique in that the strength of one essential component, the rock itself, can seldom be determined accurately. Similarly, the ground stresses are rarely well understood. Ground control engineers have had to develop novel techniques to compensate for these deficiencies. An assessment of the integrity of any mine structure must begin with an analysis of

- The structural integrity and strengthof the roof rock,
- The excavation geometry, and
- The forces applied to the mine roof

### 3.5.2 Strength of Rock

Rock strength is traditionally estimated from laboratory tests. The uniaxial compressive test is the most commonly used. Figure 3.4 shows the approximate range of compressive strengths observed in U.S. coal measure rock. Triaxial tests, where the rock is confined, more accurately simulate the three-dimensional stress that rock typically encounters underground. Shear tests of bedding planes can be very helpful in evaluating the likelihood of slip but are rarely performed in the United States. These three types of tests are shown in figure 3.5.

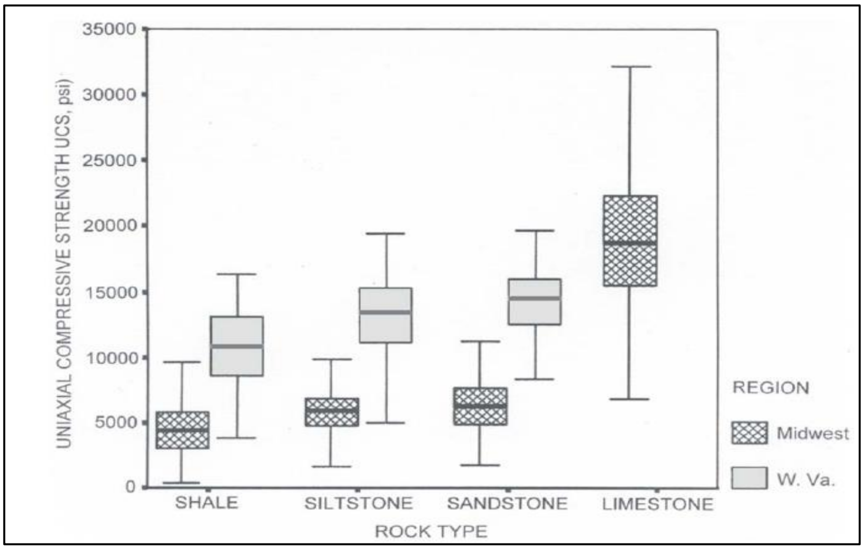


Figure 3.4 Range of compressive strength for U.S. coal measure rocks (Rusnak and Mark 2000)

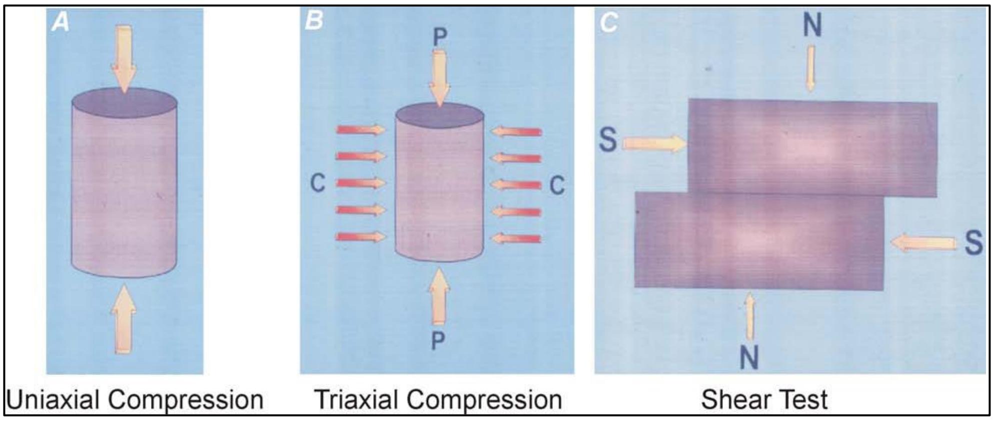


Figure 3.5 Three types of laboratory strength tests. uniaxial compressive strength test, triaxial compressive strength test, bedding plane shear test

Rock tests are severely limited in that they are conducted on small samples of intact rock. The strength of the rock mass in the mine roof is, however, determined largely by the presence of cracks, bedding planes, and other natural discontinuities. Rock mass classification systems were developed to help quantify their effects.

The Coal Mine Roof Rating (CMRR) focuses on the specific features that commonly occur in coal measure rock. It weighs the individual geo-technical factors that determine roof competence, including

- The uniaxial compressive strength of the intact rock;
- The spacing and persistence of discontinuities like bedding planes and slickensides;
- The cohesion and roughness of the discontinuities; and
- The presence of ground water and the moisture sensitivity of the rock.
- Simple index tests and observations are used to rate each of these parameters, which are then combined into a single rating on a scale from 0 to 100.
- The CMRR can be calculated from underground exposures like roof falls and overcasts (Molinda and Mark 1994) or from exploratory drill core (Mark and Molinda 1996).

### 3.5.3 Excavation Geometry

In underground coal mining, the excavation geometry does not vary much, but the span can be very important. The basic principle that governs the relationship between stability and the span was first formulated by Austrian tunnelling engineers (Bieniawski 1989).

- For a given rock mass, a tunnel's stand up time decreases as the roof span becomes wider; and

- For a given roof span, a tunnel's stand up time decreases as the rock mass quality becomes poorer.
- The greatest spans in coal mines are encountered in intersections. While entries are normally limited to 6 m, the diagonal spans of intersections are generally in the 7.5-12 m range. Approximately 70% of all roof falls occur in intersections, although intersections only account for about 20% to 25% of all drivage. Roof falls are therefore 8 to 10 times more likely to occur in intersections than in an equivalent length of entry (Molinda et al. 1991).

A study by Mark (1998) looked at standup time during extended (deep) cut mining, where the continuous miner advances the face more than 6 m beyond the last row of permanent supports. At 36 mines, it was found that when the CMRR was >55, the roof was stable in nearly every case. When the CMRR was <37, the roof collapsed before the cut could be completed. When the CMRR was between 38 and 55, extended cuts were feasible sometimes, but not others (figure 3.6). The data also show that extended cuts are less likely to be stable if either the entry span or the depth of cover is increased.

Many studies have documented the effect of roof span on stability. The longwall study cited earlier found a strong correlation between entry width and CMRR (figure 3.7) (Mark and Chase 1994). The relationship between intersection span and the incidence of roof falls at six mines was documented by (Mark et al., 1994) (figure 3.8). A similar correlation is reported by (Molinda et al., 2000).

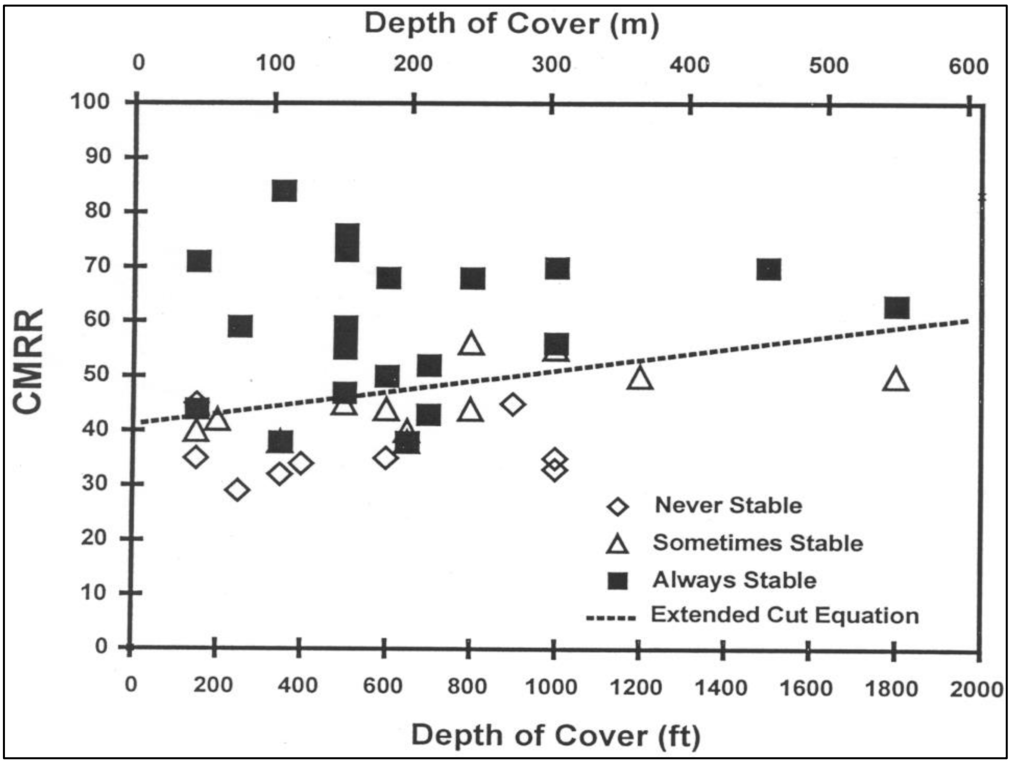


Figure 3.6 Relationship between CMRR, depth of cover, and the stability of extended cuts (Mark, 1999a)

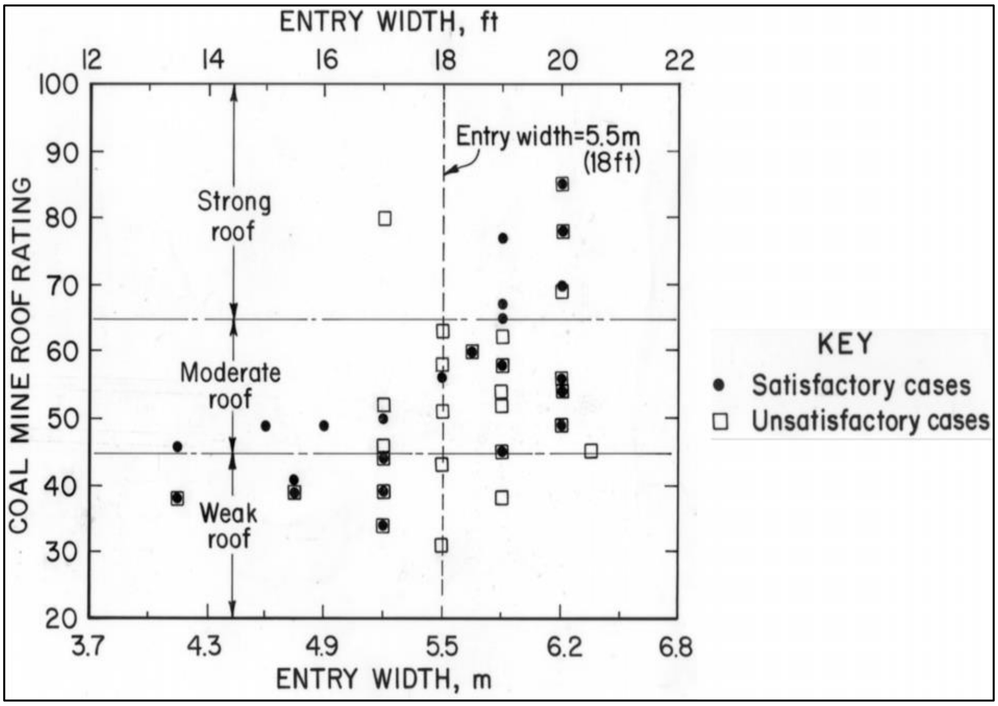


Figure 3.7 Entry widths and CMRR in U.S. longwall mines (Mark and Chase 1994)

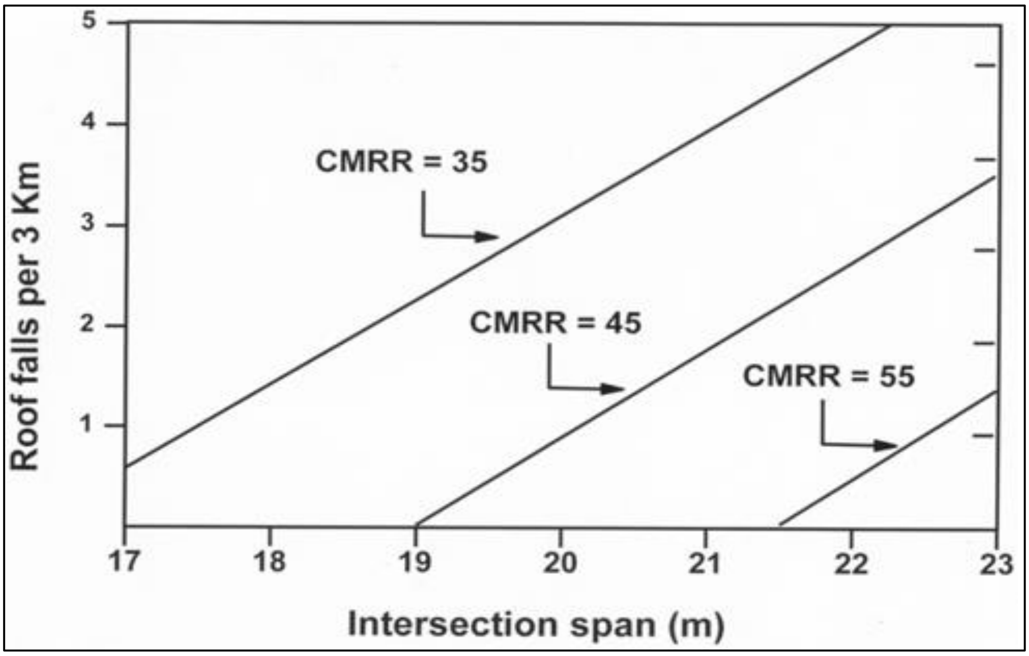


Figure 3.8 Relationship between CMRR, intersection span, and roof fall rate at six U.S. mines (Mark et al., 1994)

### 3.5.4 Forces applied to the coal mine roof

Stress is everywhere in theunderground (figure 3.9). Usually, the external forces applied to rock are all compressive, but they are not equal in all directions. The in situ stresses are normally resolved into three components: (1) vertical stress, (2) the maximum horizontal stress, and (3) the minimum horizontal stress.

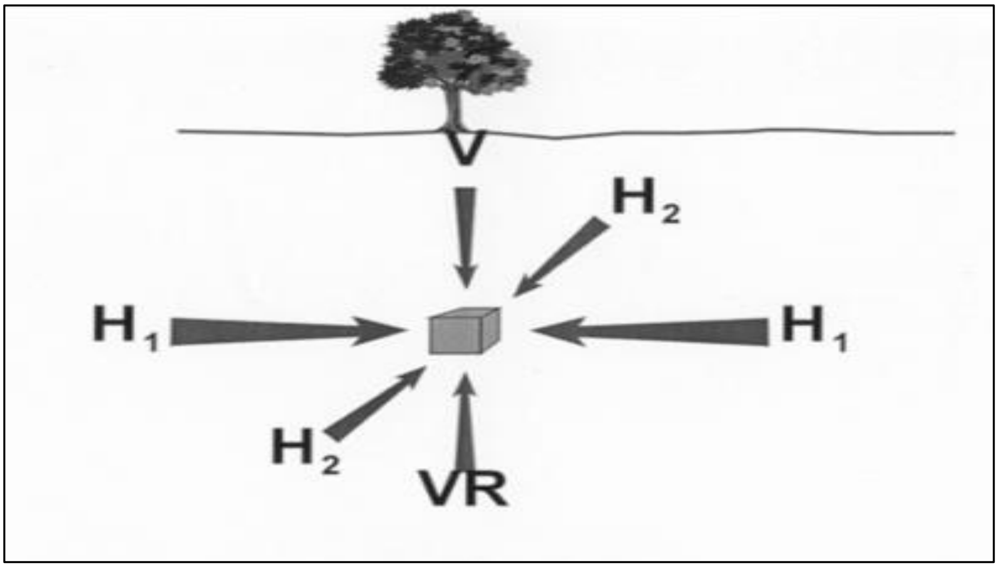


Figure 3.9 Stress on a typical element of mine roof

### i. Vertical Load

The most obvious source of loads on mine structures is the weight of the rock itself. It is convenient to analyse two types of vertical loads (figure 3.10):

- The roof load, which is due to the weight of the immediate roof strata as they sag into the mine opening; and
- The pillar loads, which are applied by the weight of theoverburden.

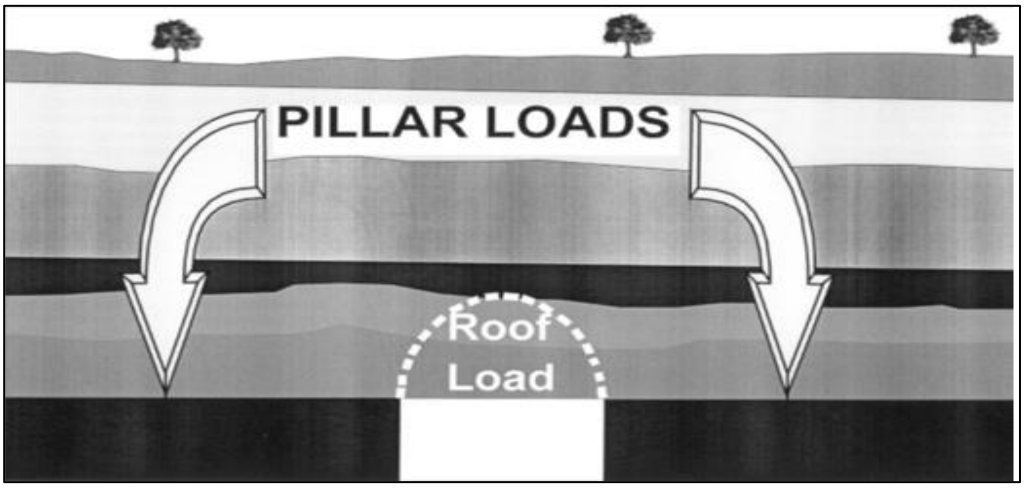


Figure 3.10 Vertical loads in underground coal mines

The roof load is the vertical force that most directly applies to roof support. Various "dead weight" design methods are based on estimating the volume of immediate roof rock that has separated from the more stable overlying rock mass and consequently must be supported (Mark et al., 2000).

The overburden load, on the other hand, is primarily carried by the pillars, but it can affect the immediate roof stability (and thus support loading) by

- Causing sloughing of the pillars, thereby increasing the roof span in the mine entry;
- Excessively loading or yielding the pillars or the mine floor, resulting in differential movements that can damage the immediate roof rock;

- Stressing the pillars, which causes them to squeeze out and apply a horizontal force to the immediate roof rock above the mine entry.

**ii. Horizontal Stress**

The horizontal stresses are normally more important to roof stability than vertical stresses. The reason is that most vertical stress is applied to the pillar, whereas the roof must bear the full brunt of the horizontal stress. Moreover, the magnitude of the horizontal stress is usually greater than the vertical stress. The effects of horizontal stress are:-

- Compressive type roof failures (commonly called cutter roof, guttering, shear, snap top, and pressure cutting). In a thinly bedded roof, the failure develops as the progressive layer–by–layer crushing of the individual beds.
- Directional effects, because roof damage is generally much greater in entries oriented perpendicular to the maximum horizontal stress than in entries driven parallel with it.

During the past 15 years, horizontal stress has become central to an understanding of coal mine ground control. An important breakthrough was the recognition that the stresses observed in coal mines are caused by global plate tectonic forces (Mark 1991).

In-situ testing is only been done in few coal mine. Therefore, enough data bank of in-situ stress is not available. However, these measured field data of Indian coalfield shows good agreement with the theoretically estimated values of the in-situ stresses (Sheorey, 1994). The theoretical model assumes that the horizontal in-situ stresses are dependent upon the elastic modulus (E), Poisson's ratio (υ), and co-officiant of the linear rock mass. Based on this concept, mean horizontal stress ($\sigma_h$) is estimated by using equation 3.1 (Sheorey, 1994).

$$\sigma_h = \sigma_v \left(\frac{\upsilon}{1-\upsilon}\right) + \frac{\beta EG}{1-\upsilon}\,(H + 1000)\ \text{MPa} \tag{3.1}$$

where, $\sigma_v$ is vertical stress in MPa, H is depth in m, g is acceleration due to gravity in m/s$^2$, $\sigma_h$is horizontal stress in MPa, $\upsilon$ is Poisson's ratio, $\beta$ is the coefficient of thermal expansion in /°C, E is young's modulus in MPa, G is the thermal gradient °C/m. among the available stress data for our coalfields, the latest three measurements (Sheorey et al., 2001) are of considerable importance and observed that the horizontal stress field is not highly anisotropic.

Two other factors also determine the degree to which horizontal stress will affect ground control:

- Roof type: Weak roof is more likely to suffer damage than strong rock, and laminations or thin bedding (as in shales or stackrock sandstones) greatly reduce the ability of rock to resist horizontal stress.
- Surface topography: Stream valleys can concentrate horizontal stresses and have often been associated with particularly difficult horizontal stress conditions. Stream valleys can also reorient the maximum horizontal stress away from the regional direction (Molinda et al., 1991).

## 3.6 Roof Bolt application

### 3.6.1 General

Roofbolting in coal mines was initiated on a significant level only after World War II. At that time, the technological factor that drove the change from timber support to roof bolts was the increased mechanization in the mining industry, while the introduction of carbide-tipped drill bits made the drilling of bolt holes on a production basis feasible (Thomas 1950, 1954).

Roof bolting technology is widely and successfully accepted for support design in an underground coal mine. It is the only way to efficiently support the roof, particularly in the case of mechanised bord and pillar operation by continuous miner technology. It also reduces the hindrance for the smooth operation of machinery and manpower in underground working as compared to other support systems used in mine. With this background, roof bolt technology is proved to be an effective support system at a number of mechanised operations in Indian coalfields.

### 3.6.2 Type of Bolt

The type of bolt can be important for roof control because each support has characteristics that determine how the bolt will support the roof. The main characteristics that differentiate supports are whether the bolt is pre-tensioned and whether the anchorage length is full or point contact. How the bolt will interact with the site-specific rock mass properties and stress conditions and stabilize the roof must be considered when selecting support (Mark et al., 2000, Deere et al., 1970, Scott et al., 1989). The design of the support system must also consider other support properties, including anchorage capacity, anchorage load distribution, axial stiffness and toughness, shear resistance, shear stiffness, and shear toughness (Karabin et al., 1980).

The five main types of roof bolts used in coal mines can be classified by two criteria: anchorage length and whether the support is installed with pretension (Scott 1989; Peng 1998). Both criteria affect how the support actually functions in supporting the rock. From an anchorage standpoint, the bolts are either point- or full-contact anchors. For a point-anchor system, the anchorage lengths are usually 0.6 m and include the mechanical anchor and resin-assisted mechanical anchor bolt. With the full-contact supports like the fully grouted rebar system, most, if not all, of the support is in contact with the rock. Besides anchorage, the full-contact provides reinforcement through resistance to rock movement. The full-contact support includes the fully grouted and torque tension bolts. Although the combination bolts are only partially grouted, the anchor section can be considered a full contact support.

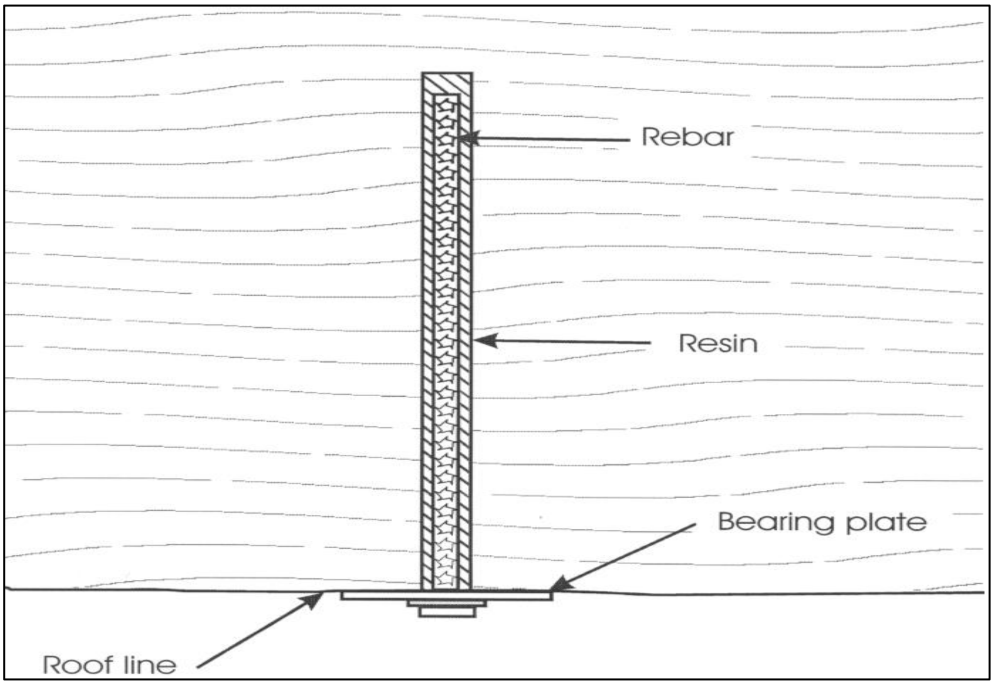


Figure 3.11 Fully grouted resin rebar installed in roof

The fully grouted bolt is now the main support used in coal mines, with about 80% of the market. The system consists of a headed rebar anchored with a full-length column of resin obtained from a cartridge (figure 3.11). The system is usually considered non-tensioned, although plate loads of several thousand pounds may develop during installation (Karabin et al., 1976). There are also special techniques that will allow for even higher installed loads (Tadolini et al., 1991).

## 3.7 Guidelines for the selection of the rockbolt system

The design of roof bolt systems can be performed by

- Analytical Approach
- Empirical Approach
- Numerical modelling Approach

### 3.7.1 Analytical Approach

In this approach, three basicmechanisms are classified as per the condition of immediate roof rock in underground working such as suspension, beam building, and keying. The details are explained below (Luo. J, 1999).

- Suspension: Whenever an underground opening is made, the strata directly tend to sag. If not properly and adequately supported in time, the immediate laminated roof could separate from the main roof and fall out. Roof bolts, in such situations, anchor the immediate roof to the self-supporting main roof by the tension applied to the bolts.
- Beam Building: The name itself explains the concept of mechanism. Applicability of roof bolt is to combine all the layers to form a beam, which helps to prevent the separation of the individual layer. This theory is used when the strong and self-supporting main roof is beyond the reasonable distance that ordinary roof bolts cannot reach to anchor for suspension.
- Keying: This mechanism has been used when the roof strata are highly fractured and blocky, or the immediate roof contains one or several sets of joints with different orientations to the roofline. Roof bolting provides significant frictional forces along with fractures, cracks, and weak planes. Sliding and/or separation along the interface is thus prevented or reduced.

This approach is not widely used because of the following reason. The first one is the variability in mechanical parameters of rock which causes a large overall inaccuracy in calculations. Another reason is the complicated condition of natural rock structures, which form an obstacle to the application of simple and readily calculated design. Results obtained using the analytical methods should therefore be used as a guide only and in combination with other design approaches.

### 3.7.2 Empirical Approach

The empirical assessments are based on the rock mass classification, case histories, and the experience of the engineers. The following three empirical assessment systems are widely and successfully used for rock bolt design.

- Empirical design recommendations in Indian coal mine.
- Design chart for rock bolt reinforcement based on the Q system
- Empirical assessments based on the rock mass classification by Bieniawski, 1984.
- **Empirical Design Recommendations**

  The empirical assessments are based on field experience, and the number of case studies has been done in different mine. Based on these experiences number of important parameters has been observed to play a vital role for designing support requirement in the underground coal mine. It includes the following parameters such as uniaxial compressive strength of intact rock, rock quality designation (RQD), spacing of joints, condition of joints, groundwater condition, and orientation of discontinuity. These parameters assign weightage based on quantification. The sum of all the parameters gives the Rock Mass Rating (RMR) of the rock. RMR has given by many researchers named as (Terzagi, 1946, Bieniawski, Z.T. 1984, 1988, 1989, 1990 and Barton et al., 1974). Two empirical approaches based on RMR in Indian and US coal mine is discussed in the following paragraph below relating the rock load height.

➢ **CMRI-ISM RMR Classification (Indian Coal mine)**

  Rock mass rating determined by CMRI-ISM (CMRI Report, 1987, V. Venkateswarlu et al., 1989; Bieniawski 1984, 1988; Singh et al., 2005; Paul et al., 2009) is modified the geomechanics Classification of Beiniawski, 1984 for estimating roof condition and support in the Indian coal mine. In this classification number of cases of different mine has been accessed. After an analysis number of key parameters have been observed, depending on their importance, weightage has been assigned on the scale of 0 - 100. The parameters have been quantifying in the range of 0 - 100 in the form of rock mass rating. Weightage of different parameters are (a) layer thickness, 0-30, (b) structural feature, 0 - 25, (c) rock weatherability, 0 - 20, (d) Strength of roof rock, 0 - 15 and (e) groundwater seepage, 0 - 10. Rock Mass Rating (RMR) is estimated to add all five parameters. It is to ensure that the RMR of each individual bed is estimated up to the height, equal to the gallery width. The weighted average of RMR of each individual bed is to be taken for further calculation.

The parameter values for the classification should be determined individually for all the rock types in the roof, up to a height of at least 2m.

- **Existing Support Design Methodology**

Rock Mass Rating (RMR) is the sum of the five parameter ratings. If there is more than one rock type in the roof, RMR is evaluated separately for each rock type, and the combined RMR is obtained as:

$$\text{Combined RMR} = \frac{\sum(\text{RMR of each bed} \times \text{bed thickness})}{\sum \text{thickness of each bed}} \tag{3.2}$$

The RMR so obtained may be adjusted, if necessary, to account for some special situations in the mine like great depth, stresses and method of work.

Table 3.2 CMRI-ISM RMR roof classification

| **Parameter** | **Range of values** | | | | | |
|---|---|---|---|---|---|---|
| 1. Layer thickness | (cm)<br>Rating | <2.5<br>0-5 | 2.5-7.5<br>12-Jun | 7.5 – 20<br>13-20 | 20- 50<br>21 -26 | >50<br>27-30 |
| 2. Structural features | (index)<br>Rating | > 14<br>0-4 | 14-11<br>10-May | 11-7<br>16-Nov | 4-Jul<br>17-21 | 4-10<br>22-25 |
| 3. Slake durability Index | (%)<br>Rating | <60<br>0-3 | 60-80<br>8-Apr | 85 – 97<br>13-Sep | 97-99<br>14- 17 | >99<br>18-20 |
| 4. Strength of the rock | (kg/sq.cm)<br>Rating | < 100<br>0- 1 | 100-300<br>6-Mar | 300-600<br>10-Jul | 600 – 900<br>13-Nov | >900<br>14- 15 |
| 5. Ground water seepage rate | (ml/m in)<br>Rating | >2000<br>0- 1 | 2000-200<br>4-Feb | 200-20<br>7-May | 20-0<br>9-Aug | Dry<br>10 |
| **RMR** | | **0-20** | **20 – 40** | **40-60** | **60-80** | **80-100** |
| CLASS | | V | IV | III | II | 1 |
| **DESCRIPTION** | | **VERY POOR** | **POOR** | **FAIR** | **GOOD** | **VERY GOOD** |

An empirical relation obtained between RMR and rock loads is:

**Rock load at gallery (tonnes/Sq.m)**

$$RLD = B \times \gamma \times (1.7 - 0.037 \times RMR + 0.0002 \times RMR^2) \quad (3.3)$$

**Rock load at Junction (tonnes/Sq.m)**

$$RLD = 5B \times \gamma \times \left(1 - \left(\frac{RMR}{100}\right)^2\right) \quad (3.4)$$

where. $\gamma$ is the unit weight of rock in t/m$^3$, B is the roadway width, m and RMR is the average rockmass rating of the immediate roof after adjustment.

Support load density can be calculated by using below expression.

$$Support\ Load = \left(\frac{No.\ of\ Bolt \times Bearing\ Strength\ of\ a\ Bolt}{Row\ Spacing \times Gallery\ Width}\right) \quad (3.5)$$

Using the above expression factor of safety is calculated using expression (3.6).

$$Factor\ of\ Safety = \frac{Support\ Load}{Rock\ Load} \tag{3.6}$$

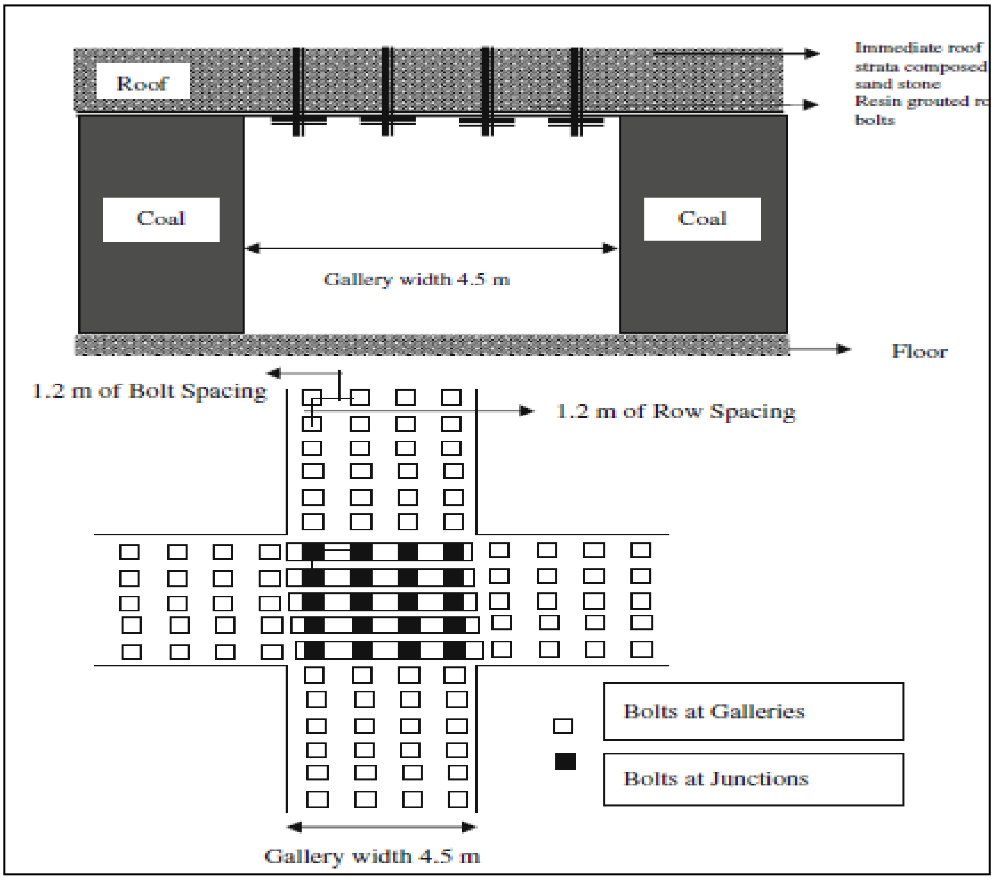


Figure 3.12 Support pattern installed in gallery and junction

➢ **USBM classification**

The coal mine roof rating (CMRR) system was developed by the US Bureau of Mines (Barton N, 1974). The range of CMRR is 0 - 100. This system has been derived for studies of 59 coal mines in various US coalfields and considers the parameters cohesion/roughness of weakness planes (CMRR = 0 - 35), joint spacing, and persistence (CMRR = 0 - 35), and compressive strength (CMRR = 0 - 30). These three principal factors, which make up the range 0 - 100, are first determined as per standardized procedures for each bed, which comes within the bolted interval or bolted height. The equation for support load density is written below:

$$Pr = 5.7\, log_{10} H - 0.35\, CMRR \quad (3.6)$$

where, $H$ is depth of cover (ft).

## 3.8 Numerical Modelling Approach

Various models have been used in the field of engineering for the purpose of optimization and refinement of criteria for better design of engineering structures and understanding of mechanics for improved understanding of an engineering or natural system. Numerical modelling application in mining engineering aims to provide a better understanding to the mining and rock mechanics engineers for solving problems related to design of mine layout and roof support system to enable the consistent and techno-economic viable performance of mining structures throughout their planned life of the operation. Most of the numerical modelling techniques used in rock engineering are based on the Lagrangian principle and hence can be broadly categorized into differential and integral methods. This type of categorization helps in analyzing the suitability and effectiveness of an approach for solving a specific problem.

### 3.8.1 Boundary element models

In this method, the only boundary of the domain area is discretized, and therefore, it needs less computational memory to store the elemental information. However, the accuracy of the results within the solid region is relatively less, and therefore, this method has limited applicability. It is applied for solving three-dimensional problems due to its advantage of the reduction in model dimension (Kuriama et al., 1995).it is used for general deformation analysis due to underground excavations (Pan et al., 1998; Jaiswal et al., 2004; Li et al., 2009), soil-structure interaction, groundwater flow, and fracturing processes. It is more suitable to solve problems of fracturing in inhomogeneous and linearly elastic bodies.

### 3.8.2 Finite Element Model

In this method, the total area is divided into a finite number of elements/meshes. Material properties and necessary boundary conditions, stresses, strains, displacement are applied. The method experienced considerable enhancement in computational techniques and development of different software packages, which attracted many geo–technical engineers (Oraee and Hossaeni, 2007; Jaiswal and Shrivastva, 2009; Aksoy et al., 2010). It uses an implicit function scheme (Loui and Sheorey, 2001), in which all the elemental information is stored at the same time, and the solution is obtained simultaneously for all the elements. Therefore, the application of this method to solve a problem of bigger size requires a fast computer with large memory. Further, this method demands a more regular domain (in shape and size) in comparison to other numerical modeling methods, and therefore, complicated geometry requires special effort in this method.

### 3.8.3 Finite difference method (FDM)

In this method, discretization of the domain area is done in a similar way as it is done in FEM. The method also requires regular-shaped elements. Thus it is difficult to use this approach for the simulation of complicated geometry. It uses an explicit solution scheme (Yasitli and Unver, 2003; Reddish et al., 2005; Mandal et al., 2008). The formation and solution of the equations are localized, which is found to be more suitable for computers memory and storage. Therefore, even a bigger problem can be simulated with finer mesh size without much time consumption in computation. At present, the well-known computer code for rock engineering problems using the FDM approach is the FLAC$^{3D}$ (Itasca, 2012).

### 3.8.4 Selection of suitable Numerical Modelling Method

Different researchers (Goodman et al., 1968; Crouch, 1976; Shen, 1991; Weaver, 1977; Kuriyama et al., 1995; Poliakov, 1999; Fang, 2001; Mandal et al., 2008; Kushwaha et al., 2010; Li S et al., 2018 and Itasca, 2012) have advocate in favor of the finite difference method (FDM) to model a geo-technical problem. The utilization of an explicit solution scheme and attaining a solution in an iterative manner by way of finite pseudo–time steps alleviate the computer's load. This gives more liberty to the user to use finer mesh size and achieve better accuracy. Geo-technical investigation generally involves larger problems, where computer memory is an important issue and, therefore, finite difference method may be a suitable approach for the planned investigation.

It is suggested (Sheorey et al., 2000) that DEM may become a more suitable approach for a geo-technical investigation if a large amount of deformation is taking place. Due to the emerging computational power of different high-capacity machines, a number of the professional package are evolving with many newly developed algorithm/ mathematical techniques. As per the various researcher's observations (Dasgupta and Lorig, 1996; Ardito and De-La-Suta, 1997; Kumar, 1999; Jade et al., 2000; Porter et al., 2001; Singh et al., 2001 and Sitharam and Latha, 2002; Jing and Hudson, 2002; Itasca, 2012), it is found that among the various available numerical modelling tools, FLAC$^{3D}$ (Fast Langrangian Analysis of Continua in three dimensions) and 3DEC are the two professional package for performance evaluation of the underground structures. But it is found that the simulation of underground depillaring operation in 3DEC is difficult (Mandal, 2009).

Considering the difficulties in modelling by 3DEC, it is decided to use continuum analysis software package FLAC$^{3D}$. The package can easily incorporate bedding planes, which are the main discontinuities of the proposed study. It is also decided to use Rock

Mass Rating (RMR) (Bieniawisky, 1976), which could automatically take care of discontinuities. Another reason for choosing $FLAC^{3D}$ is its affordability, acceptance, and available experience for Indian geo-mining conditions.