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Estimation of rock load height during development operation in bord and pillar coal mine using numerical simulation method

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ABSTRACT

Bord and pillar mining is the predominant method for extraction of coal from underground in India. Strata control is a major problem affecting safety and productivity of the mine. In conventional method of extraction worked by LHD/SDL has gallery width up to 4.8 m. However, in mechanized mining operation, gallery width goes up to 6.6 m. As per the Director General of Mine Safety (DGMS) guidelines, suggested support pattern is given for gallery width up to 4.8m. In this paper, the study focused to develop generalized equation of roof failure height during development working for wider gallery operation. The input parameters have taken into consideration are rock mass rating (RMR), uniaxial compressive strength (UCS), in - situ stress ratio, depth of working, ratio of gallery width to height. The immediate roof of the model has considered as elasto - plastic in nature. By, considering all these parameters rock load height has been estimated at gallery and junction of the mine during development operation. A generalized equation has been made for gallery as well as junction with the help of regression analysis based on numbers of model results, considering all the input parameters. The required support load density has been estimating with the help of generalized equation. For calibration of the model results, number of case studies has been solved and analyzed.

1. INTRODUCTION

Coal is an important resource in India. It is the primary source of fuel used in power plants throughout the country, apart from power plant it has also used in steel, cement and other industries. In India, the coal is being extracted by mainly two means, opencast and underground working. In coal industry, millions of employs are work below ground. The major accident causes in Indian coal mines are the failure of roof and side. Experience of the past clearly brings out that roof fall is one of the predominant causes of fatalities in below ground coal mines and that trend continues even today. There were 10 accidents due to the ground movement involving fifteen fatalities and four serious injuries during the year 2013, out of which nine accidents were due to fall of roof and one accident were due to the side fall. Roof fall accidents accounted for 13% of all fatal accident in coal mines and it contributed 43% of all fatal accidents in below ground operations. [1]

Rock bolting is more economic than other support system uses in underground mine because its installation is very easy as compared to the other. So, it saves material and manpower consumption to improve the productivity of the mine. It also reduces the hindrance, for the smooth operation of machinery and manpower in the underground working as compared to other support system used in mine. There is various research has been developed to design the support load requirement fall in the category of analytical, empirical and numerical.

Rock mass rating (RMR) plays important role in design and selection of support system. RMR has been estimated based on field experience and the number of case studies has done in different underground mines. The RMR based support design is successfully implemented and observed good result in case of conventional mining using LHD/SDL operation with gallery width up to 4.8 m. But, for mechanized mine operation using continuous miner technology of higher gallery width, RMR based support design does not give the good result.

So, in this paper, an attempt has been made to analyze the complex behavior of an immediate roof during development stage for wider gallery operation. To fulfill the objective of the research parametric study has been done using three – dimensional numerical simulation for estimating the roof failure height. The study includes four variable parameters i.e. strength of the rock, depth of working, stress ratio and area of excavation which play important role for estimating the required support density in gallery and junction.

2. PREVIOUS WORK

There are numbers of research have been done in support design in the form of mathematical, empirical and numerical approach. These approaches are summarized below.

2.1. Analytical approach

In this approach, three basic mechanisms 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. [2]

- Suspension: Whenever an underground opening is made, the strata directly tend to sag. If not properly and adequately supported in time, the laminated immediate 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 an 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 roof line. Roof bolting provides significant frictional forces along 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. First one is the variability in mechanical parameters of rock which cause 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.

2.2. Empirical approach

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 in designing support requirement in an underground coal mine. It includes following parameters such as uniaxial compressive strength of intact rock, rock quality designation (RQD), the spacing of joints, condition of joints, groundwater condition and orientation of discontinuity. These parameters assigning weightage based on quantification. The sum of all the parameters gives Rock Mass Rating (RMR) of the rock. RMR has given by many researchers named as Terzagi [3], Bieiawski [4-7] and Barton et al [8]. Two empirical approaches based on RMR in Indian and US coal mine is discussed in the following paragraph below relating the rock load height.

2.2.1. CMRI RMR classification

Rock mass rating determined by CMRI-ISM (CMRI Report, [9], V. Venkateswarlu et al. [10], Bieniawski [5], Singh et al. [11], Paul et al., [12] is modified the Geo - mechanics Classification of Beiniawski, [4] for estimating roof condition and support in Indian coal mine. In this classification number of cases of different mine has been accessed. After an analysis number of key parameters has been observed. Depend on their importance weightages 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. Weightages of different parameters are (a) layer thickness, 0-30, (b) structural feature, 0 - 25, (c) rock weatherability,0 - 20, (d) Strength of roof, 0 - 15 and (e) groundwater seepage, 0 - 10. Rock Mass Rating (RMR) is estimated to add of all the 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.

Support load density (P_r) in t/m^2 can be estimated by using equation given below.

At gallery

$$P_r = \gamma B\,(1.7 - 0.037RMR + 0.0002RMR^2) \quad (1)$$

At junction

$$P_r = 5\gamma B0.3\left(1 - \left(\frac{RMR}{100}\right)^2\right) \quad (2)$$

where, γ = unit weight of rock in t/m^3, B = gallery width in m, RMR = weighted average rock mass rating. From the figure 1.0, it is clear that this classification is only valid for lower gallery width because the support load density in junction is less than that of in developed gallery for higher bord width which is showing unreasonable.

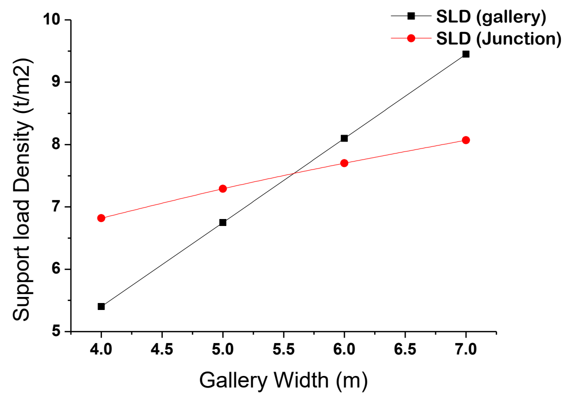


Figure 1. Support load density at gallery and junction by CMRI RMR classification

2.2.2. USBM classification

The coal mine roof rating (CMRR) system is developed by the US Bureau of Mines. The range of CMRR is being 0 - 100. This system has been derived from 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.

$$Pr = 5.7log10H - 0.35CMRR \quad (3)$$

where, H is depth of cover (ft).

2.3. Numerical approach

Numerical Simulation method using gives the reasonable understanding of roof behavior in an underground coal mine. Many researchers give the support design guideline to estimate the dead load in the roof using numerical simulation method. On the basis of rock load height or dead load, computed the support requirement of different place in mine. Some of the research on support design based on numerical simulation has been discussed below.

2.3.1. Numerical simulation on support design for depillaring panel

The empirical design has been developed by A. Kushwaha, et al. [13] during depillaring operation. In this design methodology, a generalized empirical equation has been developed to estimating the required support load density at different places of the face based on geo - technical parameters of the mine and physic - mechanical properties of the immediate roof rocks during mechanized coal pillar mining. The equation depends on various parameters such as RMR, depth, gallery width and stress ratio. The elastic model has been used to estimate the rock load height using numerical simulation approach. The minimum and maximum principle stress σ1i, σ3i around an excavation are computed, and the rock load height can be estimated by safety factor at different points and drawing its contour. In this method, factor of safety has taken as ≤1.5.

2.3.2. Efficient placement of roof bolt as breaker line support

This study was conducted by Sahendra Ram, et al. [14]. Depending upon the site conditions, field studies of roof bolt-based breaker line support (RBBLS) found that its positions need to vary from 0.5 to 2 m out-bye sides from the goaf edge for a better performance. This variation is done to adjust the extent of spalling/loosening of sides of the surrounding natural supports. As per the studied site conditions of different mechanized depillaring operations, a detailed parametric investigation is conducted on the simulated models to estimate rock load height (RLH) for a given site conditions. Results of the static (elastic) simulation studies are used for an estimation length of a bolt in the RBBLS. But the field measurements found that the bolts were subjected to dynamic loading during caving of the roof strata. Accordingly, the support resistance of the RBBLS is estimated considering the dynamic effect of caving, derived from the field measurements. On the basis of these investigations, empirical formulations are attempted among the relevant geo-mining parameters for the design of a RBBLS.

3. METHODOLOGY

There are following four parameters have been observed to contribute to the development of rock load height in an underground coal mine. The range of different parameters has been taken into consideration for model simulations are explained below and also shown in Table 1.0.

Table 1. Variation of parameters considered for simulation process

Sl. No	Parameter		Minimum	Maximum	Interval
1	Depth`(m)		100	500	100
2	In - situ stress Ratio		0.5	1.5	0.5
3	G_w/G_h Ratio	G_w	Constant gallery width 6.0 m		
		G_h	2	4	1
4	Strength (MPa) ***(*Estimated by using eqn. 11)***	UCS	30	40	10
		RMR	40	60	10

The strength of rock: Strength of the rock mass directly depends on the RMR and UCS of rock which is estimated by using equation 10. The value of RMR is ranging from 40 to 60 and UCS from 30 to 40 has been taken for simulation process.

Depth: The depth of cover varies by 100 m to 500 m considering of an interval of 100 m getting sufficient range.

Stress Ratio: The in - situ stress ratio (horizontal to vertical stress) is ranging from 0.5 to 1.5 m with an interval of 0.5.

$\frac{Gw}{Gh}$: It has been taken as 3.0, 2.0 and 1.5of varying height of extraction ranging from 2 m to 4 m with an interval of 1 keeping fixed gallery width of 6.0 m

An attempt has been made to analyze the complex geometry of the mine roof by three-dimension numerical simulation method using finite difference method. Input parameters are taken into consideration are geo - technical properties of immediate roof and geo - mining condition. The elastic model has been taken into consideration except for immediate roof rock. It has been observed that its behavior is elasto - plastic in nature.

Numerical simulation of different model with various combination of parameters discussed in the above paragraph has been using to estimate the rock load height. With the help of rock load height, estimate support load density, by which design the support requirement in the gallery.

A parametric study has been chosen for study of different model. With the help of regression analysis, generalized equation has been developed to estimate the support load density in the gallery.

4. NUMERICAL SIMULATION

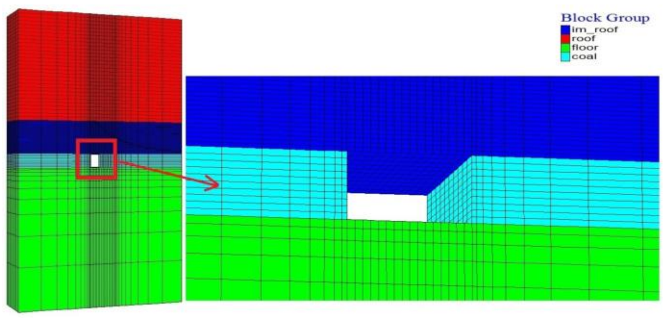


Figure 2. Three – Dimensional model with different layers for simulation of mine gallery

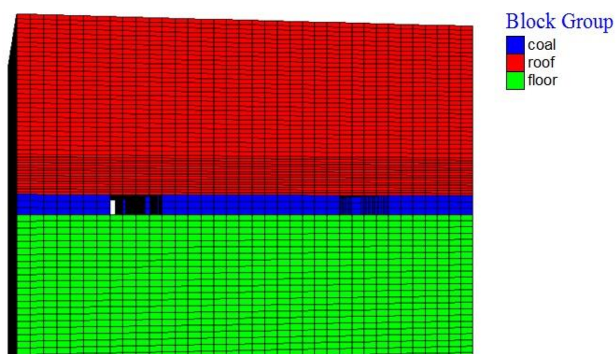


Figure 3. Three – Dimensional model for simulation of junction

Three - dimensional numerical simulation gives the accurate understanding to analyze the complex roof strata. Finite difference method using FLAC 3D software of Itasca group has been chosen for study.
The model input parameters include geometry of the mine which covers depth, $\frac{Gw}{Gh}$, geotechnical properties such as Poisson's ratio, UCS, RMR, cohesion, friction angle, dilation and stress ratio. By applying these input parameters, the model simulates until it achieves its equilibrium condition.

4.1. Model geometry

Three - dimension model of mine gallery has shown in figure 2.0 and junction in figure 3.0, which consist four numbers of layers i.e. floor, coal, immediate roof, and main roof. The dimension of three-dimensional model for gallery is 38 m width, 10 m long, 30 m roof and 30 m floor height and for junction 56 m width, 56 m long 30 m roof and 30 m floor height shown in figure 3.0. The immediate roof has highly discretized because the focus is to interpret the behavior of immediate roof rock.

4.2. Boundary condition

The depth of cover of the coal seam is varying from 100m to 500m and model height is around 30.0 m from the coal seam. So, vertical stress has applied to the top of model, which has calculated by using the formula in equation 4 with gravity loading, while the horizontal stress has calculated by using the formula in equation 5 [13]. Model is fixed in all the six side.

In - situ vertical stress can be written as

$$\sigma_v = \rho g H \tag{4}$$

$$\sigma_h = \sigma_v \left(\frac{\mu}{(1-\mu)}\right) + \left(\frac{\beta E G}{(1-\mu)}\right) (H + 1000) \tag{5}$$

where,

σ_v = vertical stress in MPa
H = depth in m
ρ = average density in t/m3
g = acceleration due to gravity in m/s^2
σ_h = horizontal stress in MPa
μ = Poisson's ratio
β= is the coefficient of thermal expansion in /°C
E = Young's Modulus in (MPa)
G = is the thermal gradient °C/m

4.3. Material property

An elastic model has been used to simulate the rock strata except for immediate roof, which was taken into considered as an elasto- plastic model. The actual behavior of immediate roof as strain softening model but, in this study, it has been taken as Mohr Columb model because it is assumed that the roof is massive in nature and there is no layer within the bolt length. Sheorey [15] failure criterion is using for simulation of the model because it gives reasonable understanding of rock behavior in Indian coal mines.

$$\sigma_1 = \sigma_{cm} \left(1 + \left(\frac{\sigma_3}{\sigma_{tm}}\right)^{b_m}\right) \tag{6}$$

$$\sigma_{cm} = \sigma_c \, exp\left(1 + \left(\frac{RMR-100}{20}\right)\right) \tag{7}$$

$$\sigma_{tm} = \sigma_t \, exp\left(1 + \left(\frac{RMR-100}{27}\right)\right) \tag{8}$$

$$b_m = b^{\frac{RMR}{100}},\ b_m<0.95 \tag{9}$$

where, σ_1 is triaxial strength of rock mass and σ_3 is confining stress in MPa, σ_c and σ_{cm} is the uniaxial compressive strength for intact rock and rock mass respectively in MPa, σ_t and σ_{tm} is the tensile strength of intact and rock mass in MPa, RMR is Bieniawski Rock Mass Rating and b and b_m are the exponent in failure criterion for intact rock and rock mass, respectively. The value of b for coal measure rock has been taken as 0.5[13].

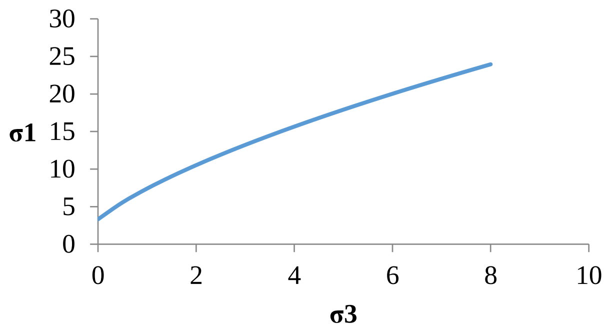


Figure 4. Graph shows the Sheorey - Failure criterion

For estimating the value of c (cohesion) and ϕ (friction angle) for an elasto - plastic material of immediate roof convert Sheorey failure criteria (figure 4.0) to Mohr Columb failure criteria. Following below expression can be used.

$$\sigma_1 = 2c\left(\frac{\cos\phi}{(1-\sin\phi))}\right) + \sigma_3\left(\frac{(1+\sin\phi)}{(1-\sin\phi)}\right) \tag{10}$$

Differentiating both side with respect to σ_3 in eqn. 6 and eqn. 10 we get,

$$\frac{d\sigma_1}{d\sigma_3} = b_m\left(\frac{\sigma_{cm}}{\sigma_{tm}}\right)\left(1 + \left(\frac{\sigma_3}{\sigma_{tm}}\right)^{(b_m-1)}\right) = x \tag{11}$$

$$\frac{d\sigma_1}{d\sigma_3} = \left(\frac{1+\sin\phi}{1-\sin\phi}\right) = x \tag{12}$$

Friction angle (ϕ),
From equation (12),

$$\phi = \sin^{-1}\left(\frac{x-1}{x+1}\right)$$

$$\phi = \sin^{-1}\left(\frac{\left(b_m\left(\frac{\sigma_{cm}}{\sigma_{tm}}\right)\right)\left(1+\left(\frac{\sigma_3}{\sigma_{tm}}\right)^{(b_m-1)} - 1\right)}{\left(b_m\left(\frac{\sigma_{cm}}{\sigma_{tm}}\right)\right)\left(1+\left(\frac{\sigma_3}{\sigma_{tm}}\right)^{(b_m-1)} + 1\right)}\right) \tag{13}$$

Cohesion (c),

$$c = \frac{\sigma_{cm}\left(1+\left(\frac{\sigma_3}{\sigma_{tm}}\right)^{b_m}(1-\sin\phi)\right) - \sigma_3\,(1-\sin\phi)\tan\phi}{2\cos\phi} \tag{14}$$

A dilatancy model of Mohr-Coulomb plastic potential surface has been proposed by Alejano and Alonso [16] based on the analysis of triaxial test. It was proposed that the dilation angle is a function of instantaneous internal friction angle (ϕ_{inst}), confinement (σ_3), and plastic shear strain (γ^p). At the zero plastic shear strain ($\gamma^p = 0$), dilation angle is referred to as the peak dilation angle (φ_{peak}). It was expressed as given below.

Dilation angle (φ),

$$\varphi = \phi \, log\left(\frac{\sigma_{cm}}{\sigma_3 + 0.1}\right)\left(\frac{1}{1 + log\, \sigma_{cm}}\right) \tag{15}$$

4.4. Simulation

A total of around 450 numerical models were run with different combinations of parameter ranging from minimum to maximum value. Table 1.0 shows the variation of different parameters taken into consideration for simulation process has explained below.

5. RESULTS AND DISCUSSION

It is observed from the simulation process that the rock load height depends on the numbers of parameters. The simulation using FLAC 3D has solved the model with the change of properties in terms of cohesion, friction angle and dilation in each zone in the model. In plastic model, material properties are updated every100 steps of the simulation process to understand and analyze the behavior of roof and solved until the stage of equilibrium has achieved. Simulation results have been discussed below in developed gallery and at the junction during development operation

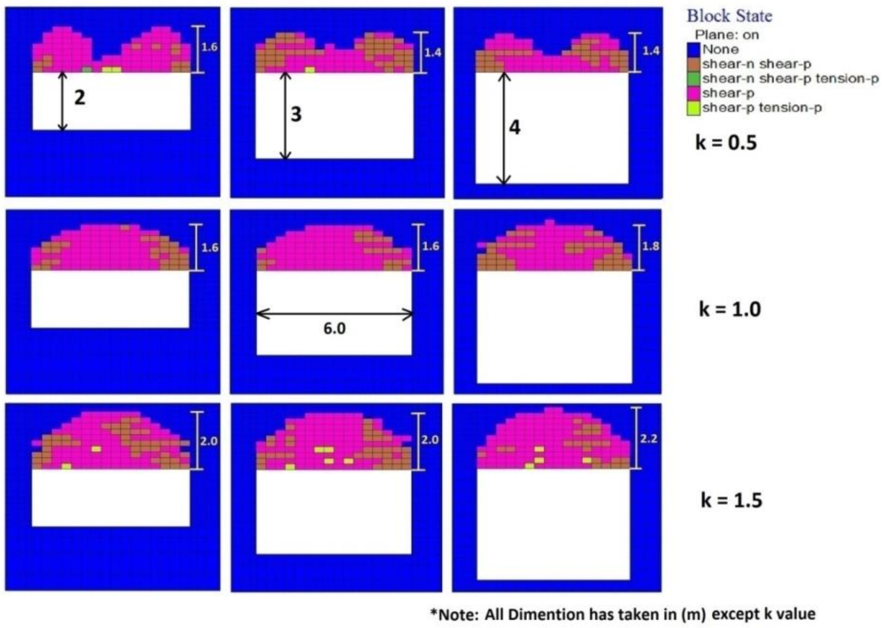


Figure 5. Rock Load Height at 100m depth and strength 3.25MPa (RMR = 40, UCS = 30 MPa)

5.1. Simulation result of developed gallery

Figure. 5 shows the yield of the roof at a depth of 100 m and poor rock having strength 3.25 MPa (RMR = 40, UCS = 30) with the variation of G_w/G_h ratio. The details of failure of roof are explained below.

In, figure. 5(a) at k = 0.5 the failure is initiated from both the corner of roof. The height of failure is maximum at the corner as compared to the center of roof. In the figure, shows the type of failure is shear failure because the vertical stress is more as compared to the horizontal stress. Rock load height is decreasing while increasing the area of excavation.

In, figure. 5(b) at k = 1.0 the failure of the roof has been initiated like a dome shape. The height of failure is maximum at the center as compared to the corner of the roof. In the figure, shows the type of failure is shear failure because the vertical stress is equal to the horizontal stress. Rock load height is increased with increasing the area of excavation.

In, figure. 5(c) at k = 1.5 the failure of the roof has been initiated like a dome shape. The height of failure is maximum at the center as compared to the corner of the roof. In the figure, shows the type of failure is shear and tension failure because the horizontal stress is more as compared to the vertical stress. The shear failure is started from the corner of the gallery and develops tension failure at the center shown in the figure. The rock load height is increasing with the increasing the area of excavation of the gallery.

The similar type of yield pattern has been observed from the result at varying depth with different parameters but the magnitude of the yield height or rock load height has different. The detail result of varying parameters has been discussed with the help of graph below.

In, figure. 6 graphs show the rock load height with different combination of parameters. In the graph x-axis shows the strength of rock (with the combination of RMR and UCS), the y-axis is shown the rock load height and the variation of graph in terms of different area of excavation with different k value. The details of graph are explained below.

At depth 100 m:

The variation of rock load height at depth 100 m, shown in figure 6(a) with different k value and area of excavation. It has been observed from the graph that when the area of excavation increases the value of rock load height decreasing in case of k = 0.5 value, but in case of k>0.5, it gives the opposite result i.e., the value of rock load height increasing with the increasing value of area of excavation. At k = 0.5 the maximum value of rock load height is2.2 m in case of height of extraction 2 m, and the minimum value is 0.8 m when the height of extraction is 4 m. But, At k>0.5 the maximum value of rock load height3.0 m has been estimated when the height of extraction is maximum i.e., at 4 m and minimum value of rock load height 0.6 m has been observed at minimum height of extraction i.e., at 2 m.

At depth 200 m:

The variation of rock load height at depth 200 m shown in figure 6(b) are varying in different k value and area of excavation. It has been observed from the graph that when the area of excavation increases the value of rock load height decreasing in case of k = 0.5 value, but in case of k> 0.5, it gives the opposite result i.e., the value of rock load height increasing with the increasing value of area of excavation. At k = 0.5 the maximum value of rock load height is2.8 m in case of height of extraction 2 m, and the minimum value is 1.6 m when the height of extraction is 4 m. But, at k>0.5 the maximum value of rock load height4.2 m has been estimated when the height of extraction is maximum i.e., at 4 m and minimum value of rock load height 1.6 m has been observed at minimum height of extraction i.e., at 2 m.

At depth 300 m:

The variation of rock load height at depth 300 m shown in figure 6(c) are varying in different k value and area of excavation. It has been observed from the graph that when the area of excavation increases the value of rock load height decreasing in case of k = 0.5 value, but in case of k >0.5, it gives the opposite result i.e., the value of rock load height increasing with the increasing value of area of excavation. At k = 0.5 the maximum value of rock load height is3.4 m in case of height of extraction 2 m, and the minimum value is 1.8 m when the height of extraction is 4 m. But, At k >0.5 the maximum value of rock load height5.0 m has been estimated when the height of extraction is maximum i.e., at 4 m and minimum value of rock load height 1.8 m has been observed at minimum height of extraction i.e., at 2 m.

At depth 400 m:

The variation of rock load height at depth 400 m shown in figure 6 (d) are varying in different k value and area of excavation. It has been observed from the graph that when the area of excavation increases the value of rock load height decreasing in case of k = 0.5 value, but in case of k >0.5, it gives the opposite result i.e., the value of rock load height increasing with the increasing value of area of excavation. At k = 0.5 the maximum value of rock load height is4.0 m in case of height of extraction 2 m, and the minimum value is 2.0 m when the height of extraction is 4 m. But, at k > 0.5 the maximum value of rock load height5.6 m has been estimated when the height of extraction is maximum i.e., at 4 m and minimum value of rock load height 2.4 m has been observed at minimum height of extraction i.e. at 2 m.

At depth 500 m:

The variation of rock load height at depth 500 m shown in figure 6 (e) are varying in different k value and area of excavation. It has been observed from the graph that when the area of excavation increases the value of rock load height decreasing in case of k = 0.5 value, but in case of k > 0.5, it gives the opposite result i.e., the value of rock load height increasing with the increasing value of area of excavation. At k = 0.5 the maximum value of rock load height is4.4 m in case of height of extraction 2 m, and the minimum value is 2.4 m when the height of extraction is 4 m. But, at k >0.5 the maximum values of rock load height.6.4 m has been estimated when the height of extraction is maximum i.e., at 4 m and minimum value of rock load height 2.8 m has been observed at minimum height of extraction i.e., at 2 m.

Also, it has been observed that at same value of RMR with different value of UCS the value of rock load height gives different value; it means that the value of UCS is also playing an important role for development of rock load height in the gallery. So, for strength parameter of the rock, it has been taken with the combination of RMR and UCS using the equation 7.

The yield pattern has been changed based on different value of stress ratio. So, from the above observation, it has been analyzed that the generalized equation has been made for k = 0.5 and k > 0.5 for simplifying the equation.

5.2. Simulation result at junction

Figure. 7 shows the yield of roof at a depth of 100 m and poor rock having strength 3.25 MPa (RMR = 40, UCS = 30) at k = 0.5 and height of extraction is 3m. The details of failure of roof are explained below.

From the simulation result it has been observed for all value of k, failure of the roof has been initiated from both the corner of the gallery and ultimately built like a dome shape. The height of failure is maximum at the center as compared to the corner of the roof. In the figure, shows the type of failure is shear and tension. The shear failure is at the corner of the gallery and tension failure developed at the center shown in the figure 7. The rock load height is increasing with the increasing the area of excavation of the gallery.

6. REGRESSION ANALYSIS

Based on above analysis it has been observed that each individual parameter has its important role to develop the rock load height in the developed gallery. Thus, it can be expressed as simple continuous functions for all the parameters for estimation of rock load height (RLH) during development operation.

6.1 Generalized equation at the developed face in gallery

A suitable equation is thus chosen for two different k value at k = 0.5 and k > 0.5 because in the result section it has been observed that for different k value the result gives opposite to each other.

At k = 0.5,

$$RLH/m = Z_1 D^{a1} S^{a2} \left(G_w/G_h\right)^{a3} \tag{16}$$

At k > 0.5,

$$RLH/m = Z_2 D^{a11} S^{a12} \left(G_w/G_h\right)^{a13} K^{a14} \tag{17}$$

where, RLH is the rock load height in m, D is the depth of cover in m, S is the strength of the rock which is the combination of RMR and UCS expressed in MPa, $\frac{\mathrm{Gw}}{\mathrm{Gh}}$ is the ratio of width and height of extraction, K is the stress ratio in in-situ horizontal to vertical stress and Z_1, Z_2, a1, a2, a3, a11, a12, a13, a14 are constants.

Based on statistical analysis of 360 datasets observed from the simulation results, following equation has been formed to estimate the rock load per meter (RLH/m) of gallery width.

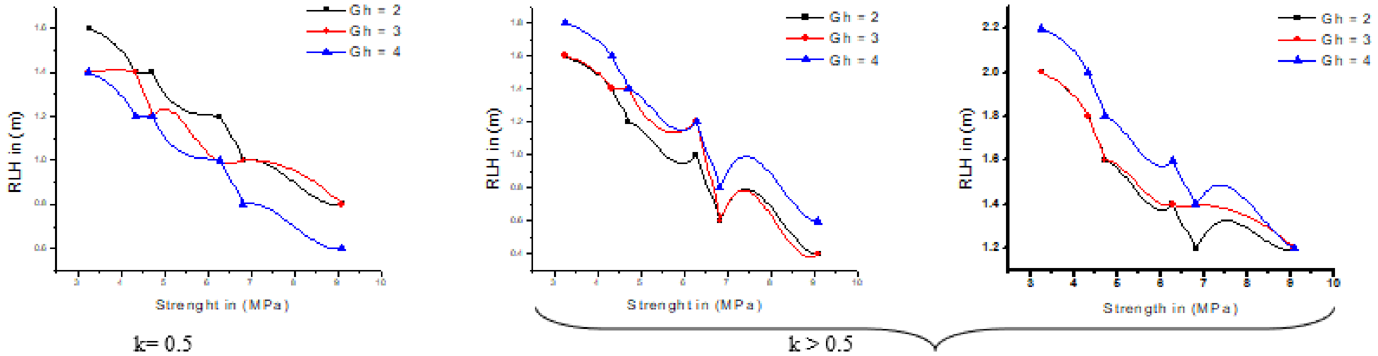
At k = 0.5,

$$RLH/m = \left(\frac{0.08\ D^{0.33}(G_w/G_h)^{0.2}}{S^{0.46}}\right) \tag{18}$$

At k> 0.5,

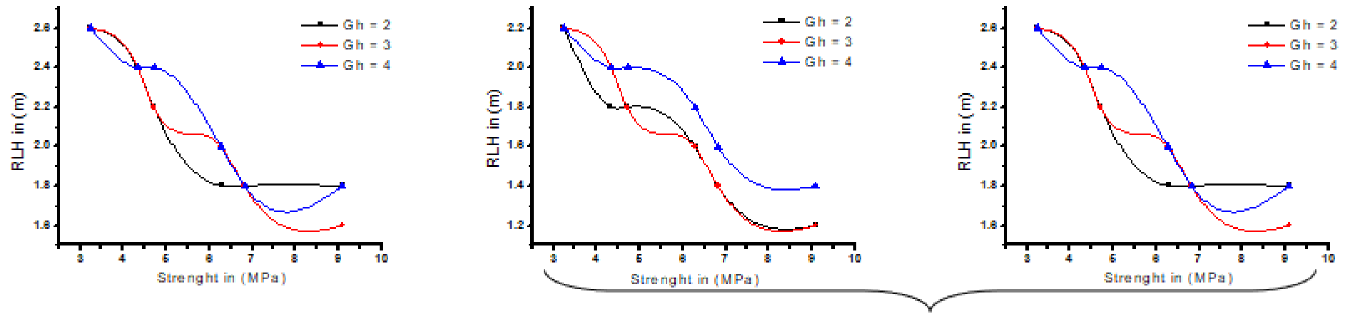
$$RLH/m = \left(\frac{0.07\ D^{0.46}K^{0.63}}{(G_w/G_h)^{0.7}S^{0.73}}\right) \tag{19}$$

Thus, for particular gallery width, it has been estimated the rock load height in (m) by multiplying the equation (18) and (19) with gallery width.



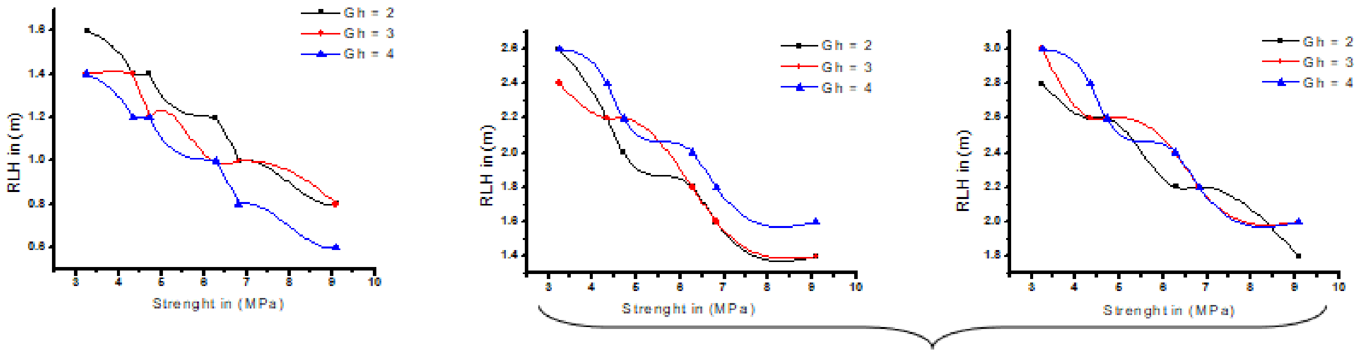
k= 0.5 k > 0.5

(a). at, 100 m depth



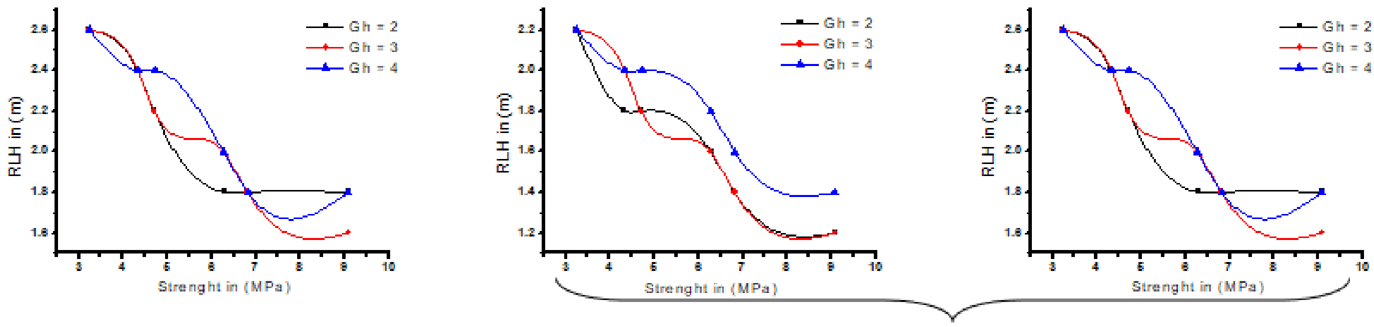
k= 0.5 k >0.5

(b). at, 200 m depth



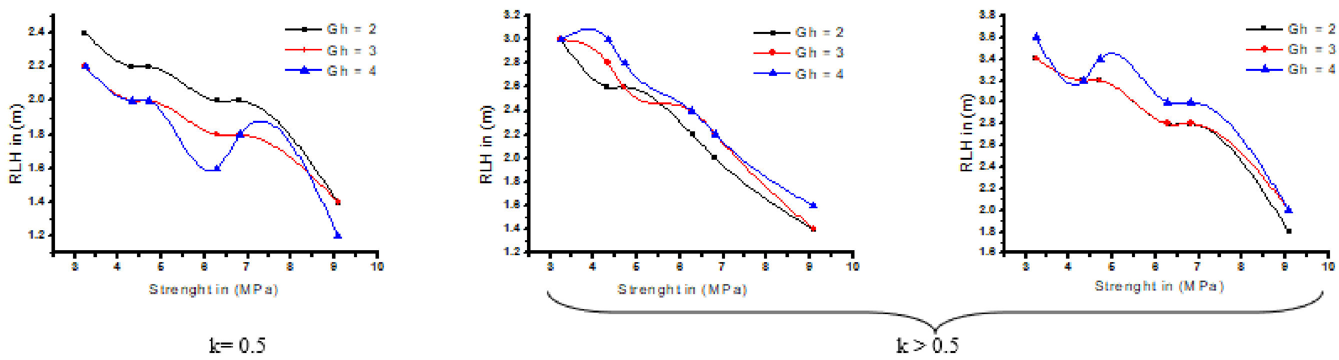
k= 0.5 k >0.5

(c). at, 300 m depth



k= 0.5 k >0.5

(d). at, 400 m depth



(e). at, 500 m depth

Figure 6. (a) to (e) shows the Rock Load Height at different depth of working at mine gallery

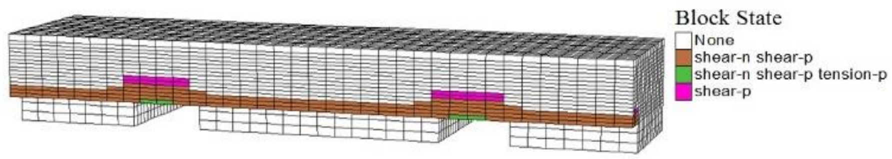


Figure 7. Rock Load Height at 100m depth and strength 3.25 MPa (RMR = 40, UCS = 30 MPa) at k = 0.5 and height of extraction = 3m

At k = 0.5,

$$RLH=\left(\frac{0.08\,D^{0.33}(G_w/G_h)^{0.2}}{S^{0.46}}\right)\times G_W \quad (20)$$

At k> 0.5,

$$RLH=\left(\frac{0.07\,D^{0.46}K^{0.63}}{(G_w/G_h)^{0.7}S^{0.73}}\right)\times G_W \quad (21)$$

6.2. Generalized equation at the junction during development operation

From the simulation results, it has been observed that for all value of k at varying parameters the yield pattern is same. Thus, simplified equation can be expressed by

$$RLH/m = Z_3 D^{a21} S^{a22}\,(G_w/G_h)^{a23}\,K^{a24} \quad (22)$$

where, RLH is the rock load height in m, D is the depth of cover in m, S is the strength of the rock which is the combination of RMR and UCS expressed in MPa, $\frac{G_w}{G_h}$ is the ratio of width and height of extraction and Z_3, *a21, a22, a23* and *a24* are constants.

$$RLH/m=\left(\frac{0.55D^{0.04}(G_w/G_h)^{0.11}K^{0.08}}{S^{0.06}}\right) \quad (23)$$

Thus, for particular gallery width, it has been estimated the rock load height in (m) by multiplying the equation (23) with gallery width.

$$RLH=\left(\frac{0.55D^{0.04}(G_w/G_h)^{0.11}K^{0.08}}{S^{0.06}}\right)\times G_w \quad (24)$$

7. ESTIMATION OF SUPPORT REQUIREMENT

Once we know the rock load height in the gallery by multiplying the unit weight of immediate roof rock (γ) [13] in equation 20 and 21 we can easily get support load density (SLD) using expression given below.

7.1. At developed gallery

At k = 0.5,

$$SLD=\gamma\left(\frac{0.08\,D^{0.33}(G_w/G_h)^{0.2}}{S^{0.46}}\right)\times G_W \quad (25)$$

At k> 0.5,

$$SLD=\gamma\left(\frac{0.07\,D^{0.46}K^{0.63}}{(G_w/G_h)^{0.7}S^{0.73}}\right)\times G_W \quad (26)$$

7.2. At junction

$$SLD=\gamma\left(\frac{0.55D^{0.04}(G_w/G_h)^{0.11}K^{0.08}}{S^{0.06}}\right)\times G_w \quad (27)$$

To estimate the applied support load density (ASL) in support system used in the mine we can use the equation given below [13].

$$\boldsymbol{ASL} = \frac{\boldsymbol{N_b A_s}}{\boldsymbol{wR_s}} \quad (28)$$

where, $\boldsymbol{N_b}$ is the number of bolt in row, $\boldsymbol{A_s}$ is anchoragestrength of bolt in tonne (t), $\boldsymbol{w}$ is gallery width in m and $\boldsymbol{R_s}$ row spacing between bolt in m.

$$\boldsymbol{FOS} = \frac{\boldsymbol{ASL}}{\boldsymbol{SLD}} \quad (29)$$

By using equation 28 and 29 it has been estimated the requirement of bolt in gallery maintaining factor of safety1.5 or more.

8. CASE STUDY

8.1 Churcha (RO), Mine, (SECL)

Churcha (RO) is working mine under Baikunthpur area of South Eastern Coalfield Limited (SECL). It is situated in Korea district of Chhattisgarh. There are total seven number of coal seam present in this mine block, out of which only Seam number V is workable seam having working thickness 4.5m. The mine block surface topography is hilly. Presence of competent and strong dolerite sill of thickness varies from 38 m to 190 m is in the mine block. Method of working is Bord and Pillar mining with continuous miner technology. Immediate roof is massive sandstone.

8.1.1 Support design for development gallery at Churcha (RO), Mine, (SECL)

Horizontal and vertical in-situ stresses and their ratio k for different layers of the immediate roof strata were estimated using equation 4 and 5 and other relevant parameters can be taken as given in Table 2, 3 & 4. The value of thermal gradient (G) is used 0.03×10^{-6} °C/m for all cases. Putting these values in equation 4 and 5, horizontal and vertical in situ stresses and their ratio for immediate roof and coal can be obtained as follows.

For coal,

Vertical Stress,

$$\sigma_v = \rho g H$$
$$\sigma_v = 1400 \times 9.81 \times 400$$
$$\sigma_v = 5.49\ MPa$$

Horizontal Stress,

$$\sigma_h = \sigma_v\left(\frac{\mu}{1-\mu}\right) + \frac{\beta E G}{1-\mu}(H + 1000)$$
$$\sigma_h = 5.49\left(\frac{0.2}{1-0.2}\right) + \frac{30 \times 2000 \times (0.03 \times 10^{-6})}{1-0.2}(400 + 1000)$$
$$\sigma_h = 4.52 MPa$$

So, In - situ stress ratio,

$$k_{coal} = \frac{\sigma_h}{\sigma_v} = \frac{4.52}{5.49} = 0.82$$

Similarly, for roof

$$\sigma_v = 8.80\ MPa$$
$$\sigma_h = 5.68 MPa$$

$$k_{roof} = \frac{\sigma_h}{\sigma_v} = \frac{5.68}{8.80} = 0.65$$

The weighted average of in - situ stress ratio is

$$k_{weighted\ av.} = \left(\frac{\rho_{coal} \times k_{coal} + \rho_{roof} \times k_{roof}}{\rho_{coal} + \rho_{roof}}\right) = \frac{1.4 \times 0.82 + 2.24 \times 0.65}{1.4 + 2.24} = 0.71$$

where,
ρ_{coal} = density of coal in t/m3
ρ_{roof} = density of roof in t/m3

Now, the weighted average of density is

$$\gamma_{weighted\ av.} = \left(\frac{\rho_{coal} \times t_{coal} + \rho_{roof} \times t_{roof}}{t_{coal} + t_{roof}}\right) = \frac{1.4 \times 4.5 + 2.24 \times 6.0}{4.5 + 6.0} = 1.88\ t/m^3$$

where,
t_{coal} = thickness of coal in t/m3
t_{roof} = thickness of roof in t/m3

Strength of rock mass can be estimated by using equation (7)

$$\sigma_{cm} = \sigma_c exp\left(1 + \left(\frac{RMR - 100}{20}\right)\right)$$
$$\sigma_{cm} = 44 \times exp\left(1 + \left(\frac{71 - 100}{20}\right)\right)$$
$$\sigma_{cm} = 10.32\ MPa$$

Table 2. Parameters used for determining in situ stresses and their ratio k for Churcha RO Mines

Rock Type	Average Depth, D (m)	Young's Modulus, E (MPa)	Density (kg/m³)	β (10^{-6}/°C)
Seam (Coal)	400	2000	1.4	30
Immediate roof (Sandstone)	394	8300	2.24	8

Table 3. Geo – technical properties of immediate roof of different underground mines

Name of Mines	Rock Type	Y_m (MPa)	t_{seam} (m)	pr	CS (MPa)	TS (MPa)	RMR
Churcha, RO, SECL	Sandstone	8300	3.0	0.2	44	4.4	71
Gare Palma, Sector IV	Sandstone	5000	3.0	0.2	5.5	0.55	48.6

Table 4. Parameters used for determining in situ stresses and their ratio k for Gare Palma, Sector IV Mines

Rock Type	Average Depth, D (m)	Young's Modulus, E (MPa)	Density (kg/m³)	β (10^{-6}/°C)
Seam (Coal)	122	2000	1.8	30
Immediate roof (Sandstone)	117	5000	2.2	8

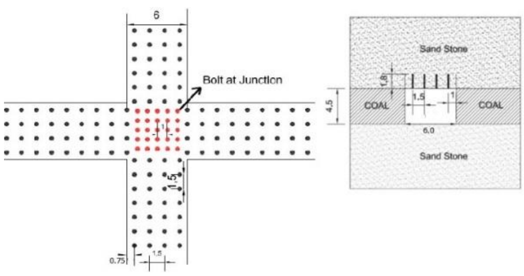


Figure 8. Existing support pattern in Churcha, RO of SECL

For calculation of factor of safety, the anchorage strength of the roof bolt having length 1.8 m, diameter 20 - 22 mm full column grouted with resin capsule has considered as 25 ton. Roof support in galleries having gallery width 6.0m by considering the installation of four nos. of rock bolt in one row with spacing between two rows of bolt are 1.5m.

Roof support in junction of 6.0m × 6.0m area, by the installation of 25 nos. of full resin grouted bolt with spacing of 1.0m in between two bolts as well as in between two rows of bolts. Existing support pattern in this mine has shown in figure 8. Using the above calculation support load density is estimate.

At Gallery,

By using equation 26, Because k value is > 0.5.

$$SLD = \gamma \left(\frac{0.07\, D^{0.46} K^{0.63}}{(G_w/G_h)^{0.7} S^{0.73}} \right) \times G_W$$

$$SLD = 1.88 \left(\frac{0.07 \times 400^{0.46} \times 0.71^{0.63}}{(6.0/4.5)^{0.7} \times 10.32^{0.73}} \right) \times 6.0$$

$$SLD = 1.49\ t/m^2$$

$$ASL_{gallery} = \frac{N_b A_s}{w R_s} = \frac{4 \times 25}{6.0 \times 1.5}$$

$$ASL_{gallery} = 11.11\ t/m^2$$

$$FOS = \frac{ASL}{SLD} = \frac{11.11}{1.49} = 7.45$$

At Junction,

By using equation 27

$$SLD = \gamma \left(\frac{0.55 D^{0.04} (G_w/G_h)^{0.11} K^{0.08}}{S^{0.06}} \right) \times G_w$$

$$SLD = 1.88 \left(\frac{0.55 \times 400^{0.04} (6.0/4.5)^{0.11} 0.71^{0.08}}{10.32^{0.06}} \right) \times 6.0$$

$$SLD = 6.79\ t/m^2$$

$$ASL_{Junction} = \frac{N_b A_s}{w R_s} = \frac{25 \times 25}{(6.0 \times 6.0) \times 1.0}$$

$$ASL_{Junction} = 17.36\ t/m^2$$

$$FOS = 2.55$$

8.2 Gare Palma, sector IV (M/S Aditya Birla)

Gare Palma sector IV coal block is allotted to M/s Aditya Birla. It is located in Raigarh District Chhattisgarh state. There are three workable seams in the mine block namely seam II, III and IV. The thickness of seam III is varying from 0.15m to 10.23m. Total geological resource in the block is about 42.08MT. Coal has proposed to be extracted by bord and pillars system of mining. It has been visually observed at various locations that shale/sandstone is present above the coal seam.

8.2.1. Support design for development gallery at Gare Palma, Sector IV (M/S Aditya Birla)

Horizontal and vertical in-situ stresses and their ratio k for the different layers of the immediate roof strata were estimated using equation 4 and 5. For the use of these equations, other relevant parameters can be taken as given in table 2, 3 &4. Putting these values in equation 4and 5, horizontal and vertical in situ stresses and their ratio for immediate roof and coal can be obtained as follows.

For coal,

In - situ Stresses are,

$$\sigma_v = 2.15\ MPa$$
$$\sigma_h = 3.06\ MPa$$

So, In - situ stress ratio,

$$k_{\ coal} = \frac{\sigma_h}{\sigma_v} = \frac{3.06}{2.15} = 1.42$$

For roof,

In - situ Stresses are,

$$\sigma_v = 2.63\ MPa$$
$$\sigma_h = 2.33\ MPa$$

So, In - situ stress ratio,

$$k_{\ roof} = \frac{\sigma_h}{\sigma_v} = \frac{2.33}{2.63} = 0.88$$

The weighted average of in - situ stress ratio is

$$k_{\ weighted\ av.} = 1.12$$

Now, the weighted average of density is (Table 4.)

Strength of rock mass can be estimated by using equation (7)

$$\sigma_{cm} = 1.53\ MPa$$

For calculation of factor of safety, the anchorage strength of the roof bolt having length 1.8 m, diameter 20 - 22 mm full column grouted with cement capsule has considered as 16 ton. Roof support in galleries having gallery width 4.8m by considering the installation of four nos. of rock bolt in one row with spacing between two rows of bolt are 1.5m.

Roof support in junction of 4.8m × 4.8m area, by the installation of 16 nos. of full column cement grouted bolt with spacing of 1.0m in between two bolts as well as in between two rows of bolts. Existing support pattern in this mine has shown in figure 9. Using the above calculation support load density is estimate.

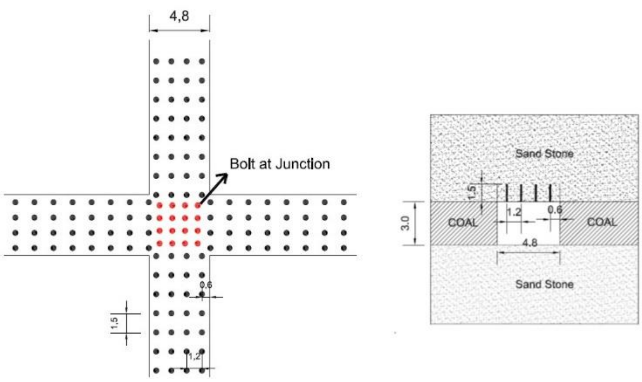


Figure 9. Suggested support pattern in Gare Palma Sector IV, of M/S Aditya Birla

At Gallery,
By using equation 26, since k value is > 0.5.

$$SLD = 3.29\ t/m^2$$

$$ASL_{gallery} = \frac{N_b A_s}{w R_s} = \frac{4 \times 16}{4.8 \times 1.5}$$

$$ASL_{gallery} = 8.88\ t/m^2$$

$$FOS = \frac{ASL}{SLD} = \frac{8.88}{3.29} = 2.70$$

At Junction,
By using equation 27

$$SLD = 6.23\ t/m^2$$

$$ASL_{Junction} = \frac{N_b A_s}{w R_s} = \frac{16 \times 16}{(4.8 \times 4.8) \times 1.0}$$

$$ASL_{Junction} = 11.11\ t/m^2$$

$$FOS = 1.78$$

9. CONCLUSION

The numerical simulation method is used to analyze the rock load height considering key parameters such as the strength of rock mass, stress ratio, gallery width, depth and $\frac{\mathrm{Gw}}{\mathrm{Gh}}$ ratio. Following conclusion has been drawn from the analysis.

Generalized empirical equations have been developed for estimation of rock load height during development stage of bord and pillar mining. Estimation of Support load requirement has been divided into two categories named as gallery and junction.

At Gallery or face the empirical equation of rock load height is further divided into two parts based on k value because of its importance. For, k = 0.5 equation 25 shows that rock load height is directly proportional to the $\frac{\mathrm{Gw}}{\mathrm{Gh}}$ and inversely proportional to $\frac{\mathrm{Gw}}{\mathrm{Gh}}$ in case of k > 0.5 shown in equation 26. From the simulation process, it has also been observed for k = 0.5, the yield of the roof has maximum at the corner of pillar and type of failure is shear. For, k > 0.5, yield of roof is like a dome shape, and maximum failure has been observed at the center of the gallery. Type of failure is tension at the center of pillar near roof surface and shear failure at both the corner of the gallery.

Equation 27, represent the rock load height at junction during development stage. From the analysis it has been observed that the failure does not change with change the value of k. So, only one simplified equation represents the rock load height at junction including all the key parameters.

Using the expression 29, it can easily estimate the number of bolts required in the gallery during development stage of an underground coal mine maintaining factor of safety more than 1.5. The developed equations can be used for all type of the development gallery either by conventional or mechanized method of operation with SDL, LHD or continuous miner. This equation cannot be used for laminated roof where the thickness of immediate roof strata is within the range of bolt length.

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