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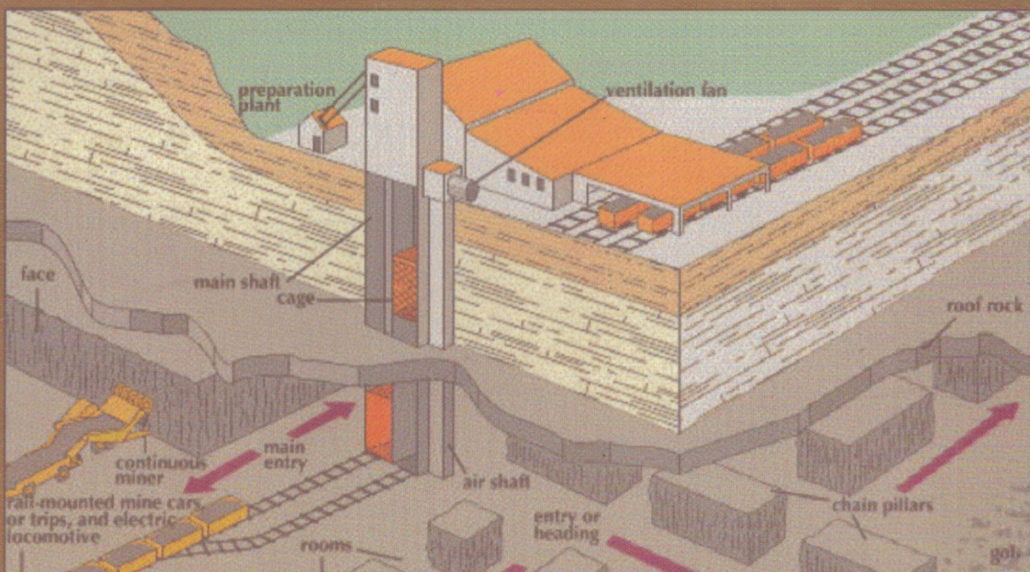

SCHEMATIC LAYOUT OF UNDERGROUND MINING

Development of support guidelines for depillaring panels in Indian coal mines

CIMFR, Dhanbad*

सारांश

भारत में भूमिगत खनन की सबसे प्रमुख विधि बोर्ड एवं पीलर प्रणाली से कोयला निष्कासन है। परंन्तु इसमें संस्तर नियंत्रण एक बड़ी समस्या है जो खान की सुरक्षा एवं उत्पादकता दोनों को प्रभावित करता है। डीजीएमएस की मौजूदा निर्देशों के अनुसार सभी डिपिलरिंग फेस में सिस्टमेटिक सपोर्ट रूल्स (एसएसआर) लागू करना अनिवार्य है। परंतु कई बार एसएसआर उचित समर्थन प्रणाली के बारे में बताने में विफल रहता है। कभी तो सपोर्ट घनत्व काफी कम होता है जिससे छत दरकने का खतरा होता है तो कभी यह काफी अधिक होता है जिससे खर्च बढ़ जाता है। विभिन्न भूवैज्ञानिक परिस्थितियों में अनुसंधान एवं विकास कार्य के फलस्वरूप भारतीय खानों के लिये सपोर्ट गाईडलाईन विकसित किये गये जिससे स्लाइस जंक्शन पर सपोर्ट घनत्व का स्टीक आकलन हो सकेगा। यह रिपोर्ट खनन अभियंताओं एवं माईन प्लानरों के लिये काफी लाभकारी सावित होगा। अध्याधिक विश्लेषन के उपरांत चार अनुभव जन्य समीकरण विकसित किये गये जिनका तीन खानों में सफलतापूर्वक परीक्षण किया गया।

ABSTRACT

Bord and pillar system of mining is the most dominant underground method of coal extraction in India. Being associated with this method of extraction, strata control is a major problem and it affects both the safety and productivity in mines. As per the existing DGMS guidelines application of Systematic Support Rules (SSR) is mandatory at the depillaring faces irrespective of rock type and its competency. But many a times SSR fails to suggest appropriate support system. Sometimes the estimated support density is inadequate causing roof failure and sometimes it is unnecessarily very high adding more to the expenditure.

The support guidelines for depillaring panels in Indian coal mines developed as a result of R&D work carried out under this project have been proved very effective in estimating more precise required support load density at the slice junction, within slice, in the split gallery and at the goaf-edge under different geo-mining conditions. This report is very useful for the practising mining engineers and mine planners during deployment of SDL, LHD and continuous miner in the depillaring panels of Indian coal mines. In this project 18 underground coal mines were selected for the study, where RMR and other geo-mining parameters of the immediate roof rock strata were determined. Based on data collected and generated 612 three-dimensional numerical models were run using FLAC 3D software. After regression analysis, four best possible empirical equations have been developed to estimate the required support load density for different places of the face and these were successfully tested in three mines.

BACKGROUND AND OBJECTIVE

Bord and pillar system of coal mining is the most dominant underground method of extraction in India. Under this method of extraction, strata control is a major problem affecting safety and productivity of the mines. The statistics of accidents in Indian coal mines, compiled over the years indicate that "fall of roof or sides" is one of the major causes of mine accident (Sinha, 2001). It is important to note that as many as 61.1% of the accidents is due to roof fall, which accounts for 28.5% of total fatalities. Based on the statistical data of the accidents occurred in different underground Indian coal mines, the following conclusions have been drawn:

- A large number of accidents (about 45%) take place in freshly exposed roof areas.
- The thickness of the fall is generally less than 1m (in 80% of the cases).
- Fall occurs in all types of roof, with higher incident rate in coal/shale roof.
- In sandstone roof, geological disturbance is the main cause of accidents.

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- Wherever a fall takes place, it is due to either unsupported roof or inadequate roof support or improper setting of supports. Conventional timber supports are found to be inadequate in a majority of the cases.

Taking cognizance of all the above strata problems, Central Mining Research Institute carried out a grant in-aid project on "Geomechanical Classification of Coal Measure Roof Rocks vis-à-vis Roof Support" in 1987 funded by the Ministry of Coal, Government of India (CMRI Report, 1987). A guideline was evolved to estimate the Rock Mass Rating (RMR) of the immediate roof strata, calculate the required support load density for developed workings in Indian coal mines and design their support patterns. Paul committee in its report in 1990 recommended for implementation of these guidelines in all the underground developed workings in Indian coal mines. The lacunae of this report were that it was silent not only about the effect of insitu stresses on roadway stability but also any guideline for depillaring panels. As per the existing DGMS guidelines, Systematic Support Rules (SSR) has to be followed at the depillaring faces irrespective of the immediate roof rock type and competence (CMR, 1957).

It was, therefore, imperatively felt that a comprehensive guideline should be developed for depillaring panels considering split and slice width, rock characterization, depth and in-situ stresses. This project was undertaken with this primary objective.

WORK DONE

To fulfill the objective of this project, required field data were collected from 30 coal mines spreading over the major coalfields in the country. Out of these, 18 mines were chosen for numerical modeling. Based on the data collected, numerical modelings were conducted and field investigations were carried out in three mines. Plans and relevant information regarding sections, RMR (where available) etc, were collected along with roof rock samples where needed. For numerical modeling, parametric changes in the following factors were made for all the 18 mines:

i) Horizontal to vertical insitu stress ratio, K

ii) Width of split/slice, W_{sp} or W_{sl}

iii) Rock mass rating, RMR

This gave rise to 612 models with as many output data sets. An example of safety factor contours plot (3D numerical model result) in and around a slice junction and slice during simulation of a depillaring panel of GDK5A incline, SCCL at the time of main fall is shown in Fig. 1. Regressions were done to give separate equations for Rock Load Height (RLH) and then required Support Load Density (SLD) for the slice junction, within slice, in the split gallery and at goaf edge.

The developed equations are as follows:

For slice junction,

$$SLD_{jn} = \frac{\gamma . H^{0.50} . K^{0.64} . W^{1.17}}{R^{0.90}} \quad (1)$$

Within slice,

$$SLD_{sl} = \frac{\gamma . H^{0.67} . K^{0.84} . W^{1.74}}{R^{1.42}} \quad (2)$$

In the split gallery,

$$SLD_{sp} = \frac{\gamma . H^{0.52} . K^{0.59} . W^{1.12}}{R^{1.02}} \quad (3)$$

For goaf edge,

$$SLD_{ge} = \frac{\gamma . H^{0.54} \; K^{0.49} . W^{0.89}}{R^{0.70}} \quad (4)$$

where, γ is the weighted average rock density of the immediate roof strata, t/m^3;

H is depth of cover, m;

K is the ratio of horizontal to vertical in-situ stresses,

W is the width of split or slice, m

R is the weighted average RMR of the immediate roof rock.

SLD_{jn}, SLD_{sl}, SLD_{sp} and SLD_{ge} are the required support density in t/m^2 at the slice junction, within slice, in the split gallery and at the goaf edge respectively.

To test the modeling results, four coal mines were selected for the field investigation where instrumented rock bolts were used to determine the axial load, bending moment, etc, developed along the bolts. Rib stability was measured with the help of stress meter. Field results were very much close to the modelling results (Fig-1).

PROCEDURE FOR ESTIMATION OF DIFFERENT VARIABLES

As per the developed equations (Eq. 1 to Eq.4), it is clear that five variables are to be known to estimate the required **Support Load Density** at different places of the face during depillaring operation. These variables are the **depth of cover (H), in-situ stress ratio (K), Rock Mass Ratings** of the immediate roof rock **(R), split and slice**

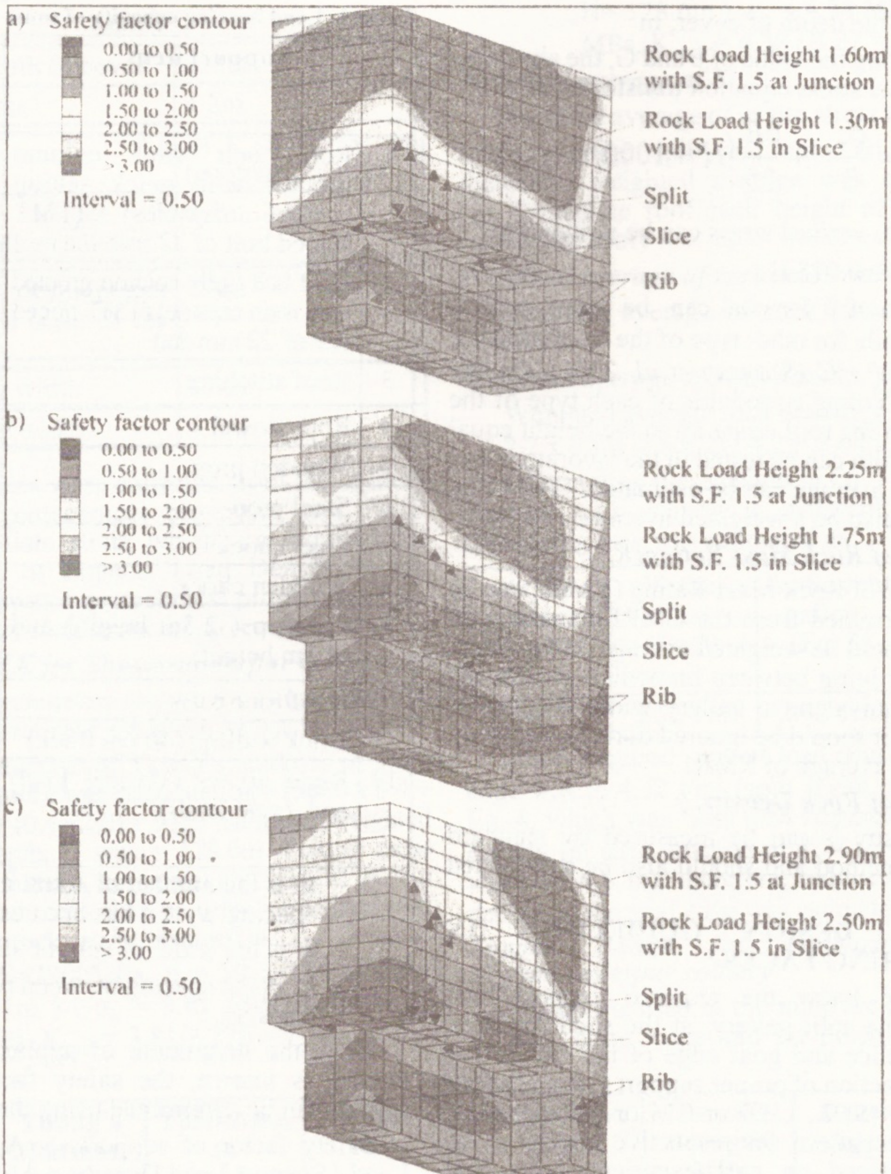

Fig.-1: Safety factor contours in and around slice junction and slice for panel no.31 of GDK5A incline, SCCL for different horizontal to vertical stress ratio, K (a) Lowest value of K (b) Moderate value of K (c) Highest value of K

width (W) and rock density (γ). Estimation procedure for these variables, except for **H and W** which are directly obtained, is as follows:

Estimation of K

Measurement is the best method to determine the in-situ ratio (K), which is the ratio of in-situ horizontal stress to vertical stress for any particular mine. In the absence of the in-situ measurements of stress values, theoretical values can be used. Theoretical value and measured in-situ stress world wide showed that the in situ horizontal stress depends on elastic constants of the rock (Sheorey 1993).

The in-situ horizontal stress equation is as follows:

$$\sigma_h = \frac{\nu}{1-\nu}\sigma_v + \frac{\beta E G}{1-\nu}(H+1000)$$

where, σ_h and σ_v are the horizontal and vertical in-situ stress respectively in MPa

υ is the Poisson's ratio = 0.25

β is the coefficient of thermal expansion /°C

E is the Young's modulus of the rock, MPa

G is the thermal gradient = 0.03°C/m for coal measure rocks

H is the depth of cover, m

After putting the value of υ and G, the above *in-situ* horizontal stress equation transforms as:

$$\sigma_h = \frac{\sigma_v}{3} + \frac{\beta E}{25}(H + 1000) \quad (5)$$

The in-situ vertical stress can be written as:

$\sigma_v = \gamma H = 0.025\ H$ (6)

The value of β for coal can be taken as 30 x 10-6 / °C while for other type of the coal measure rocks 8 x 10^{-6} / °C (Sheorey *et al.* 2001). On the other hand, Young's modulus of each type of the rock lying in the roof strata up to the height equal to gallery width can be tested in the laboratory and its weighted average can be estimated. The value of β should also be a weighted average.

Estimation of Rock Mass Rating(R)

The value of Rock Mass Rating (RMR), can be readily determined from the CMRI classification tables. It should be weighted average. If the RMR of any layer lying between immediate roof rocks of height equivalent to gallery width is more or equal to 70, it should be ignored during estimation of weighted average of RMR.

Estimation of Rock Density, γ

The density γ can be measured by standard laboratory method and should also be a weighted average.

SUPPORT DESIGN GUIDELINES IN DEPILLARING FACES

Once we know the required support load density in the split gallery, at the slice junction, within the slice and goaf edge of the depillaring face, the selection of proper support systems as per requirement (SDL, LHD or CM or conventional) and support pattern for respective areas can be designed. Based on past experiences on roof supports and their effectiveness along with pull test of different types of the bolts used in different underground mines, the load bearing capacities of different support items are given below in Table-1 (CMRI Report, 1987).

To estimate the applied support load density by different support used in the mine, Eq.-7 can be used as (Sheorey & Mukarjee, 1985):

$$Applied\ \ Support\ \ Load\ (ASL) = \frac{n\ .\ A + m\ .\ Q}{W\ .\ a}\ \ t/m^2 \quad (7)$$

where, n is the number of bolts in a row,

A is the anchorage strength of each bolt, t

Q is the load bearing capacity of the additional support if done, t

m is the number of additional support at a spacing 'a', if it has been used

W is the width of split or slice, m

a is the spacing between two consecutive rows, m.

Table-1: Load bearing capacity of some support systems

	Support item	Load bearing capacity (t)
1	Roof bolt (full column grouted with quick setting cement capsules) (TMT ribbed bolt of 22 mm dia)	6
2	Roof bolt (fully column grouted with resin capsule) (TMT ribbed bolt of 22 mm dia)	12
3	Roof stitching	8
4	Rope dowel	4
5	Wooden prop	10
6	Steel prop	30
7	Steel chock	30
8	Wooden chock	20
9	Pit prop (2.5m height) and (4.5m height)	15 & 12
10	Roof/rope truss	10
11	Brick walling (40 cm thick)	10
12	Rigid arches (vertical load) and (side loads)	7(/m length) & 2(/m length)

Once the magnitude of applied support load density is known, the safety factor of support system can be determined using the equation:

Safety factor of supports = Applied Support Load / Support Load Density = ASL / SLD

While using the equations for SLD, it should be taken into account that they have a built-in safety factor of 1.5. As such ASL / SLD $\geq$ 1.0

CASE STUDIES

Shyamsundarpur Colliery, ECL

Jambad bottom seam (RVII) of 4.2 m average thickness at Shyamsunderpur colliery of Eastern Coalfields Limited (ECL) was developed in a single lift along the floor by conventional drilling and blasting methods 10 to 15 years ago. Panel No.24 of the above colliery was extracted by using SDL up to the height of 3.6 m leaving coal of 0.6m along the roof. The immediate roof of the seam is shaly sandstone and moderately cavable. The details of the geometrical parameters of the seam

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and panel are given below:

Table-2

Maximum depth of cover	131m
Seam thickness	4.2m
Pillar size	21.3 x 21.3 m (centre to centre)
Width and height of developed gallery	4.2m, 2.4m
After development thickness coal layer in the roof	1.8m
Average split width	4.0m
Average slice width	4.8m

Rock samples were collected from the colliery to estimate geotechnical properties and RMR of the immediate strata which are required for determination of Support Load Density at the face. The estimated geotechnical properties of the immediate roof rock strata are given in Table-3.

Estimation of K for Shyamsunderpur colliery

Relevant parameters require for estimation of K of the immediate roof are given below in Table-4.

Weighted average of Young's modulus of shale and hard sandstone was calculated as = (5000 x 0.4 + 10,000 x 2.0) / (0.4 +2.0) = 9167 MPa and Weighted average of depth, H was = 128.6m. Putting these values in Eqs.5 and 6, horizontal and vertical in- situ stresses and then their then ratio for immediate roof coal and other rock can be obtained as:

For Coal (E=2000MPa, β = 30 x 10^{-6} /^{0}C, H= 131m), σ_h = 3.81 MPa, σ_v = 3.275 MPa, K_{coal} = 3.81/3.275 = 1.16

For other rock (E=9167MPa, β = 8 x 10^{-6} /^{0}C, H= 128.6m), σ_h = 4.38 MPa, σ_v = 3.22 MPa, K_{rock} = 4.38 / 3.22 = 1.36

Weighted average of K

Weighted average of in situ stress ratio K = 1.27

(Here average split and slice width is 4.2m, therefore, weighted average was estimated for the immediate roof rock height of 4.2m in the immediate roof)

Weighted average of rock mass ratings, R

Weighted average of rock mass ratings R = 53.52

Weighted average of rock density, γ

Weighted average rock density γ = (1.5 x 1.8 + 2.2 x 0.4 + 2.5 x 2) / 4.2 = 2.04 t/m^3

After putting these values in Eq. 1 to 4, required support load density at different places in a depillaring face was estimated as:

At the slice Junction, (Eq.1)	Within slice, (Eq.2)	In the split gallery, (Eq. 3)	At the goaf edge, (Eq.4)
4.74 t/m^2	3.52 t/m^2	2.42 t/m^2	4.74 t/m^2

Since, required support load density at the goaf edge was to 4.72 t/m^2 (split width of 4 m) as per Eq.-4, which was less than required support load density for slice junction of 4.74 t/m^2 (due to slice width of 4.8 m), therefore, required support load density at the goaf edge was estimated as 4.74 t/m^2.

Support design for depillaring face at Shyamsunderpur colliery

If supports used in the mine are a combination of rock bolts, cogs and pit props, then applied

Table-3

Rock type	Young's Modulus, MPa	Thickness, m	Poisson's ratio	Compressive strength, MPa	Tensile strength, MPa	Density, kg/m^3	RMR
Coal	2000	1.8	0.25	18.4	1.2	1500	52
Shale	5000	0.4	0.25	29.5	1.9	2200	53
Hard sandstone	10,000	2.0	0.25	41.0	2.8	2500	55

Table-4

Rock type	Avg. Depth H, m	Layer thickness, m	Density, t/m^3	Young's modulus E, MPa	Rock mass rating, RMR	b x 10-6 / ^{0}C
Coal	131.0	1.8	1.5	2000	52	30
Shale	130.6	0.4	2.2	5000	53	8
Sandstone	128.6	2.0	2.5	10,000	55	8

support load density can be estimated from the modified equation 8 as:

$$Applied\ Support\ Load\ (ASL) = \frac{N_b \cdot C_b}{W_s \cdot S_b} + \frac{N_c \cdot C_c}{W_s \cdot S_c} + \frac{N_p \cdot C_p}{W_s \cdot S_p} \quad (8)$$

where, N_b = Number of rock bolts in a row,

C_b = Rock bolt capacity, 6 tonne

N_c = Number of cogs in a row,

C_c = Cogs capacity, 20 tonne

N_p = Number of pit props or timber props in a row,

C_p = Pit prop capacity, 15 t, timber prop capacity, 10 t

W_s = Slice and split width 4.2m,

S_b, S_c & S_p = Spacing between two consecutive rows of bolts, cogs & pit props respectively, m

In actual practice, in the slice two cogs were set along the rib side at the interval of 2.4m in addition to one timber prop in-between and 4 timber props along the pillar side at the interval of 1.2m as shown in figure 5.1. Therefore, applied support load density in the slice was calculated from the above equation as:

Applied support load (within slice) ASL_{sl} = (3 x10 + 15 + 20) / (4.8 x 3.6) = 3.76 t/m²

Similarly in the split and at the slice junction, two rock bolts in a row at the interval of 1.2m between two consecutive rows were grouted in the roof in addition to one pit prop set in the same row at the same interval as shown in figure 5.1. Therefore, applied support load density in the slice junction and split gallery was calculated as:

Applied support load (in junction or split) ASL_{jn} or ASL_{sp} = (2 x 6 +15)/(4.0 x 1.2) = 5.63 t/m².

Similarly, at the goaf edge, three cogs were erected for a split width of 4.0 m and roof support width of 1.8 m, therefore,

Applied support load at the goaf edge ASL_{ge} = (3 x 20) / (4.0 x 1.8) = 8.33 t/m².

Safety factor for supports was calculated by dividing the applied support load density (ASL) with required support load density (SLD) at different places of the face as given in Table-5

From Table-5, it is clear that the safety factor of support system applied at different places of the face was more than 1.0 and with this pattern of support, panel was extracted successfully. Therefore, the required support load density at different places of the face obtained from the developed guidelines was well within the field requirement.

Bankola Colliery, ECL

The details of the geometrical parameters of the seam and panel are given below:

Name of the seam	RVII seam
Gradient	1 in 21
Name of the panel	RVII/5
Maximum depth of cover	85m
Thickness of the seam	4.2m
Pillar size	21.3m x 21.3m (c-c)
Width & height of developed galleries	4.5m & 2.4m
Height of extraction	3.6m
Width of split & slice	4.5m
Rib width against the goaf	2m

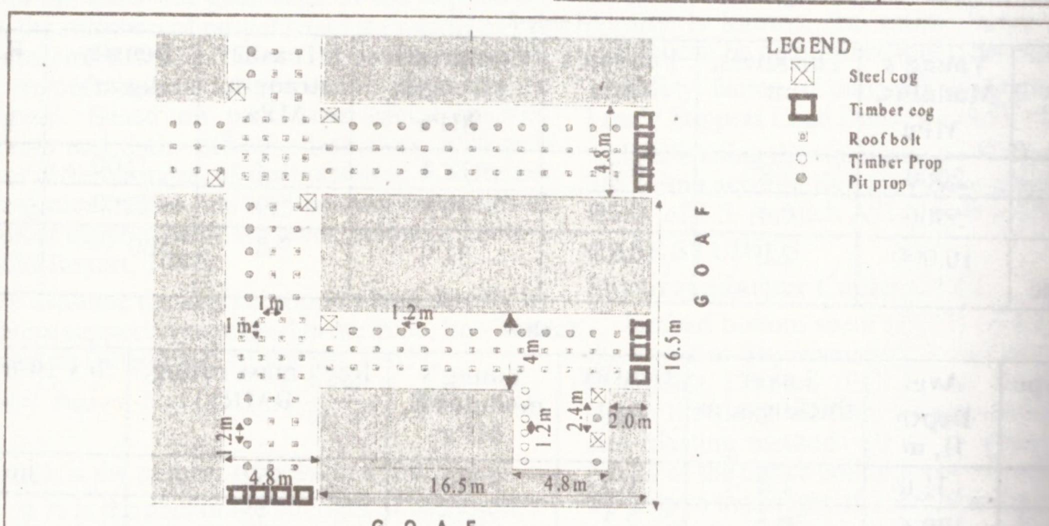

Figure-2: Support pattern for depillaring face at Bankola Colliery, ECL

Table-5

At the slice junction			Within slice			In the split gallery			At the goaf edge		
SLD_{jn}, t/m²	ASL_{jn}, t/m²	S.F.	SLD_{sl}, t/m²	ASL_{sl}, t/m²	S.F.	SLD_{sp}, t/m²	ASL_{sp}, t/m²	S.F.	SLD_{gc}, t/m²	ASL_{gc}, t/m²	S.F.
4.74	5.63	1.19	3.52	3.76	1.07	2.42	5.63	2.33	4.74	8.33	1.76

Table-6

Rock type	Young's Modulus, MPa	Depth, m	Thickness, M	Poisson's ratio	Compressive strength, MPa	Tensile strength, MPa	Density, t/m³	RMR
Coal	2000	85	1.8	0.25	38.5	2.57	1.5	56
Shale	7000	84.4	0.6	0.25	27.8	1.85	2.2	51
Sandstone	10000	82.3	2.1	0.25	41.0	2.8	2.5	55

Immediate roof of the developed workings was 1.8m coal followed by 0.6m thick shale band then sandstone. Rock samples were collected from the colliery to estimate geotechnical properties and RMR of the immediate strata to estimate the required Support Load Density at the face. The estimated geotechnical properties of the immediate roof rock strata up to 4.5m height equivalent to width of split and slice galleries are given in Table-6.

Estimation of K for Bankola Colliery

For Coal (E=2000MPa, $\beta = 30 \times 10^{-6}$ /°C, H= 85m)

$\sigma_h = 3.3123$ MPa $\sigma_v = 2.125$ MPa

$K_{coal} = 3.3123/2.125 = 1.56$

For other rock (E=9333MPa, $\beta = 8 \times 10^{-6}$ / °C, H= 82.3m)

$\sigma_h = 3.92$ MPa $\quad \sigma_v = 2.06$ MPa

$K_{rock} = 3.92 / 2.06 = 1.90$

Weighted average of in- situ stress ratio K= (1.56 x 1.8 + 1.90 x 2.7) / 4.5 = 1.76

Weighted average of rock mass ratings, R = (56 x 1.8 + 51 x 0.6 + 64 x 2.1) / 4.5 = 59

Weighted average of rock density, γ =(1.5 x 1.8 + 2.2 x 0.6 + 2.5 x 2.1) / 4.5 = 2.06 t/m³

After putting these values in Eqs. 1 to 4, required support load density at different places in a depillaring face was estimated as:

At the slice Junction, (Eq.1)	Within slice, (Eq.2)	In the split gallery, (Eq. 3)	At the goaf edge, (Eq.4)
4.04 t/m²	2.72 t/m²	2.26 t/m²	4.28 t/m²

Support design for depillaring face at Bankola colliery

In the slice, two cogs were set along the rib side at the interval of 2.4m in addition to one timber prop in-between & four timber props erected along the pillar side at an interval of 1.2m as shown in figure 5.2 in the slice. Applied support load density in the slice was calculated as (Fig-2):

Applied support load (within slice) ASLsl = (20 + 4 x 10) / (4.5 x 3.6) = 3.70 t/m²

Similarly, in the split and at slice junction, four rock bolts in a row at the interval of 1.0m between two consecutive rows were grouted in the roof as shown in figure 5.2. Applied support load density in the slice junction and split gallery was calculated as:

Applied support load in junction ASL_{jn} = (4 x 6) / (4.5 x 1.0) = 5.33 t/m²

Applied support load in split ASL_{sp} = (4 x 6) / (4.2 x 1.0) = 5.71 t/m².

At the goaf edge, three cogs were erected for a split width of 4.2m and support roof width of 1.8m, therefore applied support load density at goaf the edge ASL_{gc} = (3 x 20)/(4.2 x 1.8)=7.94 t/m².

SLD and the safety factor of support system at different places of the face are given in Table-7.

From Table-7 it is clear that the safety factor of support system applied at different places of the face was more than 1.0 and with this pattern of support, panel was extracted successfully.

GDK 5A incline, SCCL

The details of the geometrical parameters of the seam and panel are given below:

Name of the seam : seam No.1

Name of the panel : P-24

Table-7

At the slice junction			Within slice			In the split gallery			At the goaf edge		
SLD_{jn}, t/m²	ASL_{jn}, t/m²	S.F.	SLD_{sl}, t/m²	ASL_{sl}, t/m²	S.F.	SLD_{sp}, t/m²	ASL_{sp}, t/m²	S.F.	SLD_{gc}, t/m²	ASL_{gc}, t/m²	S.F.
4.04	5.33	1.32	2.72	3.70	1.36	2.26	5.71	2.53	4.28	7.94	1.86

Maximum depth of cover	: 180m
Thickness of the seam	: 2.1m
Pillar size	: 32 m x 30 m (c-c)
Height of extraction	: 2.1m
Width & height of developed gallery	: 4.0m & 2.1m
Width of split & slice	: 4.0m both
Rib width against the goaf	: 2m

The estimated geotechnical properties of the immediate roof rock strata up to 4.0m height equivalent to width of split and slice galleries are given in Table-8.

Table-8

Rock type	Young's Modulus, MPa	Depth, m	Thick-ness, M	Poisson's ratio	Compressive strength, MPa	Tensile strength, MPa	Density, t/m³	RMR
Grey Pyritic Sst.	15,000	179.8	0.2	0.25	18.4	1.85	2.5	52
White Sst.	10,000	176	3.4	0.25	15	1.5	2.5	58

Weighted average of Young's modulus E of grey pyritic sandstone and white sandstone was estimated as = (15000 x 0.2 + 10,000 x 3.8) / (0.2 +3.8) = 10250 MPa. Weighted average of depth, H was = 176m, $\beta = 8 \times 10^{-6}$ /°C. Therefore, $\sigma_h = 5.32$ MPa and $\sigma_v = 4.4$ MPa

Weighted average of in-situ stress ratio K = 3.92 / 2.06 = 1.21

Weighted average of rock mass ratings, R = (52 x 0.2 + 58 x 3.8) / 4 = 58

Weighted average of rock density, γ =2.5 t/m³

Required Support Load Density (SLD) estimated for GDK5A Incline at different positions of the face is given below:

At the slice Junction, (Eq.1)	Within slice, (Eq.2)	In the split gallery, (Eq. 3)	At the goaf edge, (Eq.4)
4.96 t/m²	3.33 t/m²	3.13 t/m²	6.30 t/m²

Support design for depillaring face at GDK5A Incline

At GDK-5A incline of SCCL, conventional split and slice method of extraction was being performed manually for developed pillars. During depillaring operation, three bolts in a row at spacing of 1.2m in split galleries and two wooden props in a row at spacing of 1.2m in each slice were used as support system. In addition to that, a wooden cog was also erected at the centre of the slice junction. At goaf edge, three sets of wooden cog were erected skin to skin as shown in Fig.-3. Induced caving was performed at regular interval of face advance, so that chances of overriding could be avoided. Therefore, with this support system, applied support load was estimated using Eq. 7 and Table-1.

Applied support load (at the junction) ASL_{jn} = (3 x 6 + 20) / (4.0 x 1.8) = 5.28 t/m²

Applied support load (within slice) ASL_{sl} = (15+10) / (4.0 x 1.2) = 5.20 t/m²

Applied support load (in the split gallery) ASL_{sp} = (3 x 6 +10) / 4 x 1.2 = 5.83 t/m²

Applied support load (at the goaf edge) ASL_{ge} = (3 x 20) / (4.0 x 1.8) = 8.3 t/m².

The safety factors of each support at different places of the depillaring face are given in Table-9.

From the above results it is clear that the safety factor of each supports at depillaring face was more than 1.0 and with this support system panel was extracted successfully.

Here, point is to be noted that with only three bolts in a row at the interval of 1.2m between two consecutive rows at the slice junction, applied support load density was only 3.75 t/m², which was less than the required support load density calculated from developed Eq.1 i.e. 4.96 t/m². Practically in the field also it was realized that without putting additional chock supports at the centre of the slice junction, condition of the junction roof deteriorated

Table-9

At the slice junction			Within slice			In the split gallery			At the goaf edge		
SLD_{jn}, t/m²	ASL_{jn}, t/m²	S.F.	SLD_{sl}, t/m²	ASL_{sl}, t/m²	S.F.	SLD_{sp}, t/m²	ASL_{sp}, t/m²	S.F.	SLD_{ge}, t/m²	ASL_{ge}, t/m²	S.F.
4.96	5.28	1.06	3.33	5.20	1.56	3.13	5.83	1.86	6.30	8.3	1.32

fastly. Keeping this view in mind an additional wooden chock support was erected at the centre of slice junction as shown in Fig.-3. With this additional support, applied support density reached 5.28 t/m² and stability of the area improved. The safety factor was more than 1.0.

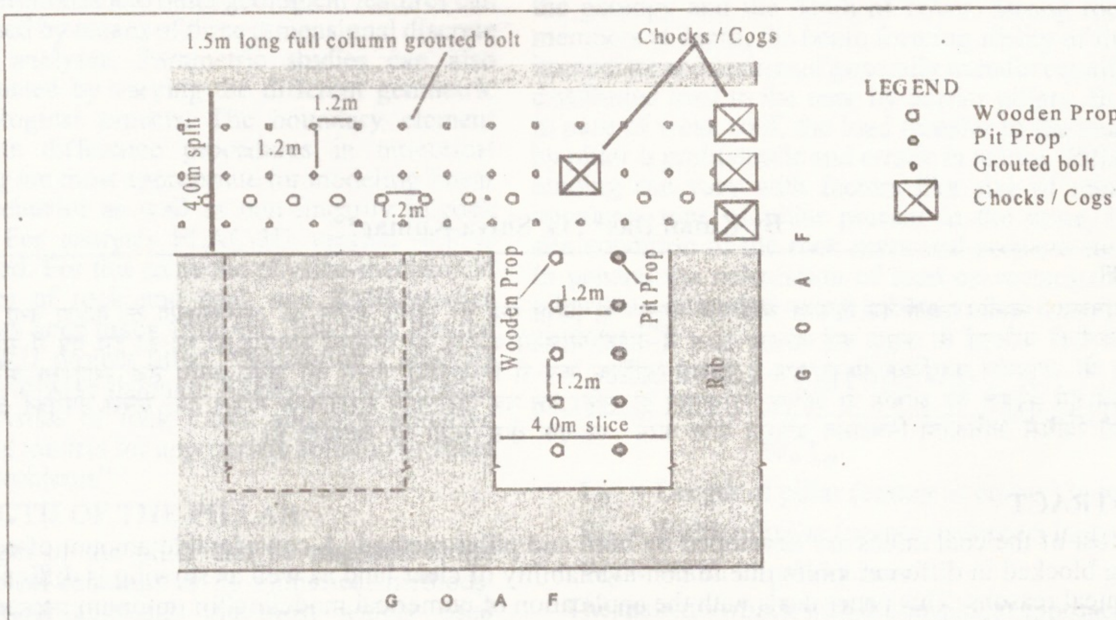

Figure-3: Support pattern for depillaring face at GDK-5A Incline, ECL

required Support Load Density (SLD) at different places of the depillaring face, design of support system can be readily planned as per requirement. For this purpose, Eq. 7 can be used to estimate the Applied Support Load (ASL) taking into account of load bearing capacity of different type of the supports generally used in Indian coal mines Table-1.

The safety factor of the support system planned for different places can be calculated as the ratio of ASL to SLD which must be more than 1 for all the places of the depillaring face.

Sometimes required Support Load Density at the goaf edge comes to be less than the slice junction value from the developed equations, in such cases, slice junction value should be taken for the goaf edge also.

CONCLUSIONS

For the estimation of required Support Load Density at the slice junction, within the slice, in the split gallery and goaf edge four equations (1 to 4) have been proposed.

These equations can be used for all types of depillaring caving faces by either conventional method or mechanised method using SDL, LHD or Continuous miners. But these are not applicable for contiguous depillaring faces. After estimating the