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A Numerical Modelling Approach to Find the Stability of RIB and Snook in Mechanised Depillaring Panel — A Case Study of Kurja Mine

Subrata Samanta[1], Rabindra Kumar Sinha[1*], Pulak Baran Chakrabarty[2] and Hemant Kumar[1]

[1]*Department of Mining Engineering, Indian Institute of Technology (Indian School of Mines), Dhanbad – 626004, Jharkhand, India; subrata.16dp000003@me.ism.ac.in, rksinha@iitism.ac.in, hemantismd@gmail.com*
[2]*Eastern Coalfield Limited, Coal India Limited, Sec. V, Rajarhat, Kolkata, India; pulakbaran@gmail.com*

Abstract

Depillaring by conventional techniques created many irregular shaped pillars, mainly located at shallow depths in different coalfields of India. Continuous Miner Technology (CMT) has been introduced to extract these coal pillars scientifically with safety and productivity with greater depth. Leaving a proper sized rib or snook is a legal requirement as it decides the efficiency and safety of the pillar. However, CMT of square/rectangular shaped pillars created irregular shaped rib and snook. With the conventional empirical formula it is difficult to estimate the strength of such ribs or snooks. These ribs or snooks should be of sufficient size to protect the adjacent slicing operation and junction as well as they should fail in a controlled manner when machines shift inside the extraction. Field studies of different mines found several factors affecting the design of rib or snook. They are various types of induced stress, geological disturbances, manner of extraction etc. Proper assessment of the performance of different sizes and shapes of ribs and snooks in the field is complex due to the problematic underground mining environment for depillaring. For successful operation of the CMT required to study every depending parameter scientifically. Therefore, a numerical model is conducted to estimate the factor of safety of rib and snook to calculate the rib and snook stability. Results of field and simulation studies are presented and discussed in this paper to determine the rib's stability in the continuous miner panel to create a safe environment for the men and machines.

Keywords: *Coal Mining, Continuous Miner, Numerical Modelling, Rock Mechanics, Rib and Snook*

1. Introduction

During depillaring operation, the safety of the machine and the coal extraction rate entirely depend on the structures of the panel, including the rib and snook design. This rib and snook are essential to support the roof while the continuous miner moves. The induced stress will be developed on the nearby pillars and the snooks and ribs with the coal extraction. The bed separation occurs during depillaring, and the load on the ribs and snooks gradually increases as the mining continues. Failure occurs if the value of induced stress is unbearable by the rib pillar. However, the immediate roof separates from the main roof with the advancement of goaf. While depillaring, tensile stresses are developed at the edge of the adjacent rib or snook, especially at the contact points in the immediate roof. For the failure of the snook when it goes inside the goaf, the size of the rib and snook should be small to enable the immediate roof to release the pressure but should have sufficient residual strength to hold

**Author for correspondence*

the immediate roof to enable maximum extraction with safety. Some studies on pillar loading identify a few techniques to determine pillar stress: beam theory, electrical analogue, numerical methods, and photo-elastic physical studies. Robert *et al.* (2016) derived pillar FoS calculations with loading estimated by the empirical strength calculation based on tributary area theory, which is applied for square pillar only. The rib is considered irregular in shape during the depillaring operation created by Continuous Miner (CM). Therefore, quite difficult to measure the pillar load by the tributary area theory. Hence, the need arises to estimate the load on ribs and snooks using methods other than the tributary area theory. In this research paper, some numerical methods have been carried out to estimate the stability of the rib and snook.

1.1 Introduction to Continuous Miner Technology

Recently continuous miner with shuttle car or ram car technology has become very popular to produce coal from underground mining. Where the immediate roof is supported with the roof bolts of adequate length installed by quad bolter or twin bolter. The behaviour of the roof strata is monitored with the help of various advanced and scientific strata monitoring instruments instead of depending on the efficiency developed by experiences. All machines and instruments in operation together is called Continuous Miner Technology (CMT). The CMT can produce a continuous flow of coal from the faces without interruption. The manpower requirement in this technology is low, thus improving the mine's Output per Man Shift (OMS). By CMT, optimum height of coal seam can be extracted, reducing the in situ loss of coal in mechanised depillaring method by the continuous miner. CMT (Figure 1) was presented in India in May 2002 (Vuuren 2002) at Anjan Hill mine in Chirimiri, SECL. Later the success of this technology prompted other coal companies in India to adopt the CMT. Due to the roof and side falls, few accidents are recorded in CMT mining during the depillaring operation. Other than the human failures, it happened mainly due to the improper geometry of the rib and snook.

Moreover, there was no scientific approach to restrict such happenings. For this technology's successful and safe operation during the depillaring, it is essential to calculate the actual load over the rib and snook, which will lead to safer extraction. The present research thus focuses on estimating load on the ribs and then calculating the Factor of Safety (FoS) of these ribs. The methodology discussed in this paper will help mine planners to design a safe and optimum layout of depillaring panels in the underground coal mine.

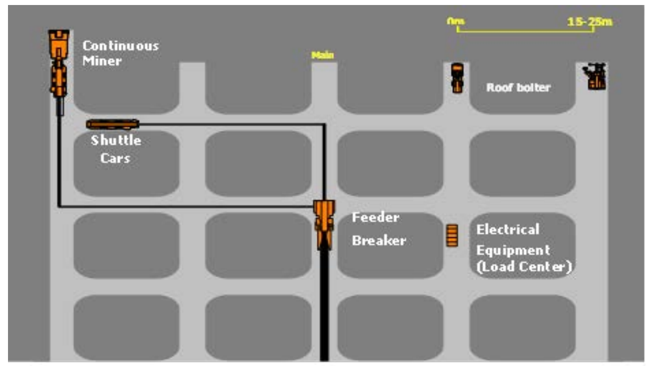

Figure 1. A layout of five heading CM panel (development).

1.2 Importance of the RIB During Depillaring

In mine working, depillaring with the continuous miner by leaving small remnant pillars is a regular practice during pillar extraction, particularly at intersections, to provide temporary support of the working area. This small remnant pillar or rib (Figure 2) is required for the safe withdrawal of the continuous miner after successful extraction. In practice, there is no consensus on the size and geometry of this rib and snook, which can be blindly followed. As such, in practice, the experience plays the only role in determining the size of the rib and snook. Few records of the premature collapse of the rib while operating the continuous miner during the depillaring panel proves the limitations based on the experience of assessing the proper size. Therefore, the strength of the ribs and snooks needs to be studied in detail which will reduce the risk of premature roof failure in the depillaring area. The rib or snook size may be kept sufficient to provide support to the roof strata above the intersection, but small enough to allow caving to commence in the goaf area to reduce the pressure in adjacent pillars. Lind (2005), based on the Australian pillar extraction experience, suggests that in weaker roof conditions, the snook sizes can be left small, which may fail in a localised load of the roof of the area under extraction. Similarly, in strong roof conditions, the final snook left has to be large enough to induce the competent roof to break off to prevent goaf override into the intersection.

Table 1. Important empirical pillar strength formulae

Sl. No.	Author	Equation
1.	Zern Edward Nathan (1928)	$C_p = C_1\sqrt{(\frac{W_p}{h_p})}$
2.	Greenwald et al. (1939)	$C_p = 0.67k_z\frac{\sqrt{w_p}}{h_p^{0.83}}$
3.	Holland (1964), Gaddy (1956), Steart (1954)	$S = 0.16k\frac{w^{0.6}}{h}$
4.	Obert and Duvall (1967)	$S = k\left(0.78 + 0.22\frac{w}{h}\right)$
5.	Salamon and Munro(1967)	$S = 7.2\frac{w^{0.46}}{h^{0.66}}$
6.	Cook (1971)	$C_p = \frac{71.8w_p^{0.46}}{h_p^{0.66}}$
7.	Bieniawski and Van Heerden (1975)	$S = k\left(0.64 + 0.36\frac{w}{h}\right)$
8.	Agapito (1982)	$\frac{c_p}{K} = 1 + C_{ah}(\frac{\sigma H_{av}}{K})^{\alpha_{ah}}$
9.	Pariseau (1982)	$\sigma_p = \frac{\gamma H\frac{(1+k)+(1-k)\cos 2\alpha_s}{2}}{(1-R)}$
10.	Madden (1991)	$C_p = k_M\frac{R_0^{0.5933}}{V^{0.0667}}\left\{\frac{0.5933}{\grave{o}}\left[(\frac{R_w/h}{R_0})^{\in} - 1\right] + 1\right\}$
11.	Sheorey (1992)	$S = 0.27\, h\, {}_{\mathrm{C}}^{-0.36} + \left(\frac{\mathrm{H}}{250} + 1\right)\left(\frac{\mathrm{w}}{\mathrm{h}} - \right)$
12.	Galvin et al. (1995)	$S = 7.40\frac{w^{0.46}}{h^{0.66}}$
13.	Galvin et al. (1996)	$S = 8.60\frac{w^{0.51}}{h^{0.84}}$

14.	Mark-Bieniawski (1988)/Chase et al. (1997)	$S = k(0.64 + (0.54\frac{w}{h} - 0.18\left(\frac{w}{h}\right)$
15.	Van-der-Merwe (2003)	$S = k\frac{w^{0.46}}{h^{0.66}}$
16.	Van der Merwe and Mathey (2013)	$S = 5.47\frac{w^{0.8}}{h}$

Nevertheless, it is also true that leaving an extensive and robust snook may invite a loss of coal in the area. Mark and Zelanko (2001) also made the study to determine the size of stook to be left after final extraction in the US mines based on the experience of Australian and South African mines. In the conclusions of their study, they suggested that an appropriately engineered snook based on the study may be of 5-10% of the original size of the pillar. However, as per the DGMS circular, the rib size should be 1.5m, and it will be judiciously reduced during the extraction. In the CMT, various mines may be clearly defined in various depths and geological conditions to leave proper sizes of rib and snook before an incident happens. In view of this issue, thought has been given for detailed research to find the actual stability of the rib and snook for safe working.

1.3 Mechanism for Rib Failure

Rocks in a mine experience either elastic or inelastic deformation under different loading conditions. Under unconfined loading conditions, the mode of failure is generally axial splitting or slabbing along the unconfined planes (Hori, 1986; Ashby, 1990), resulting from the interaction and coalescence of the pre-existing and developed micro-cracks (Bobert, 1986). Under confined conditions, the rock fails due to the formation of the shear plane. The inclination of such planes concerning the longitudinal compression axis is controlled by applied confinement (Maurer, 1965). It is also observed that the nonlinearity of the stress-strain curve and the failure criteria curve increases as there is an increase in the confining stress. At high confinement, the brittle-to-ductile transition of the intact rock occurs, and the peak strength of the intact rock does not increase after a specific value of the confinement (Mogi, 1966, 1971). Barron and Wilson (Wilson, 1972; Barron, 1986) obtained the value of the maximum peak stress on the coal pillar during its failure by equating the solid rock's triaxial strength curve and the broken rock's triaxial strength curve. With this approach, Barron, Das *et al.* (Barron, 1986; Das, 2019) estimated the strength of the coal pillars for field conditions. The broken coal mass strength can be written by

$$\sigma_{1bm} = \sigma_{cbm} + \sigma(\sigma_{3bm})^b \quad (1)$$

where, σ_{3bm} is the confining stress (MPa) of the broken coal, σ_{1bm} is the strength (MPa) of the broken coal under confinement, σ_{cbm} is the broken coal's uniaxial compressive strength (MPa), and a and b are the constants.

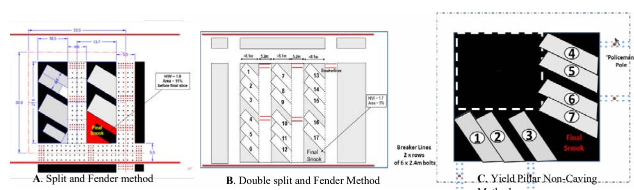

Figure 2. Various types of rib and snook formed during the mechanised depillaring process.

2. Methodology

The methodology adopted for maximisation of coal extraction during depillaring concerning the optimisation of rib and snook is shown in Figure 3. The research objective is to determine the load over the rib and snook in retreat mining. Kurja mines with various physico-mechanical properties of the rock specimens collected were determined following the ISRM suggested methods. The site characterisation included the determination of the RMR as suggested by the Paul committee (1990), collection of stratigraphy of the mine site, the study of the local geology, and the geo-mining conditions. Based on the inputs of the site characterisation, the numerical models for Kurja mines under study were developed. The numerical models so developed were calibrated with the strata monitoring data and the empirical relation of the safety of pillars (Sheorey, 1997). The output data was further validated with the Kurja mines and summarised a conclusion for the safe working in CMT.

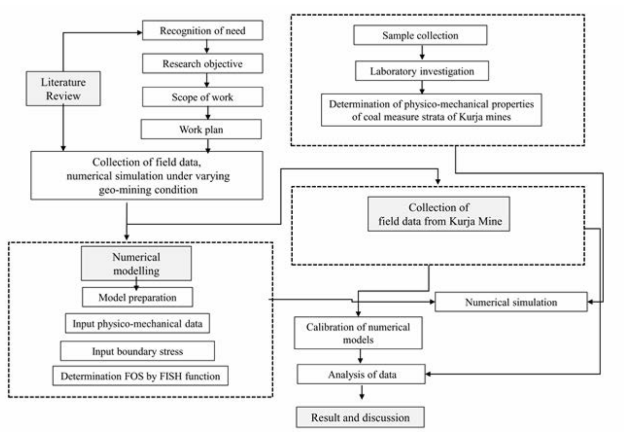

Figure 3. Research methodology for determination of the stability of rib and snook.

3. Various Pillar Strength Formulae

Coal pillar strength has been a focus of research for many years. The work of Salamon and Munro (1967) continues to find wide use in the industry, with over one million coal pillars designed to date. Based on laboratory and field testing, width (w) *and height* (h) and compressive strength of coal are the fundamental inputs for calculating the strength of a coal pillar.

However, with an increase in the number of studied stable and failed cases with time, different researchers Sheorey (1992), Madden (1991), Galvin (1999), Van-der-Merwe (2003), Van der Merwe and Mathey (2013) have attempted to validate these formulations by changing the coefficient and power constants. Important pillar strength formulae are summarised in Table 1.

3.1 Load Transfer Mechanism Through Coal Pillars

Underground excavations interrupt the equilibrium state of the strata, resulting in the setting up of different forces and redistribution of stresses to establish a new equilibrium state at a lower energy level, along with lateral movements and changes in slope in surface beds. Initially, the beds bend downward, which temporarily frees the weight of the beds. Overburdened load above the galleries in the excavated mine is transferred to the sides of the excavation, forming a pressure arch zone of relieving stress. When an opening is made, the stresses shift outward on both sides of the pillar, leaving a de-stressed zone around the pillar in the shape of an arch. The exact shape and size of the arch depend on the stress levels, shape and size of the opening, and strata properties. Subsidence occurs when the arch reaches the surface. If these increased stresses exceed the rock strength, the rock will fracture and fail (Figure 4). Generally, coal seams are weak in strength, and excavations are done over a large area, which enhances the chances of the collapse of overlying strata. The lateral compressive forces acting along the roof compels the immediate roof to bend, thus causing bed separation and subsidence. This happens due to pillars or stocks left in the goaf, which hinder a regular settlement.

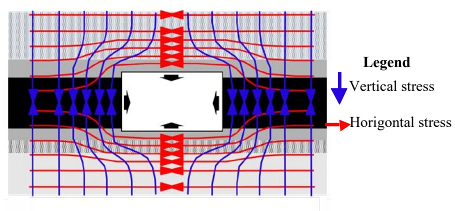

Figure 4. Stress acting over the extraction.

4. Case Study of Kurja Mine, SECL

Kurja mineis a part of the Sitaldhara mine of the Hasdeo area, SECL, situated in the district of Anuppur (Madhya Pradesh)/Korea (Chhattisgarh). The coal-bearing measures of the Hasdeo Formation are un- conformably overlain by poorly consolidated Quaternary laterites, sands and gravels and weathered Gondwana sediments. At Kurja, a 43.0m thick Dolerite sill occurs 104.0m above seam "C". In addition, dolerite intrusions occur across the coalfield as medium to thick dykes. These can be significant enough to form natural boundaries between collieries. The area has a gently rolling topography with elevations ranging from 540m to 590m above sea level. The drainage pattern of the area is radial. The northern and western part of the area is drained by the river Kewai, whereas the channels in the eastern and southern part of the area discharge into Hasdeo river, a tributary of Mahanadi. The lower Gondwana (Permain) deposits generally comprise sandstones, shales, mudstones and a sequence of coal seams. Within the Sheetaldhara block, there are three seams designated 'A', 'B' and 'C' in descending order with a separation of approximately 80m between 'B' and 'C' and 50m between 'B' and 'A'. However, only seam 'C' is considered workable. The lower seam 'C' has been fully developed by bord and pillar to the west of the designated CM site (Figure 5), and the current drilling and blasting operation are confined to the western area of the mine. Kurja mine have been taken for this study where CMT was introduced in 2014.

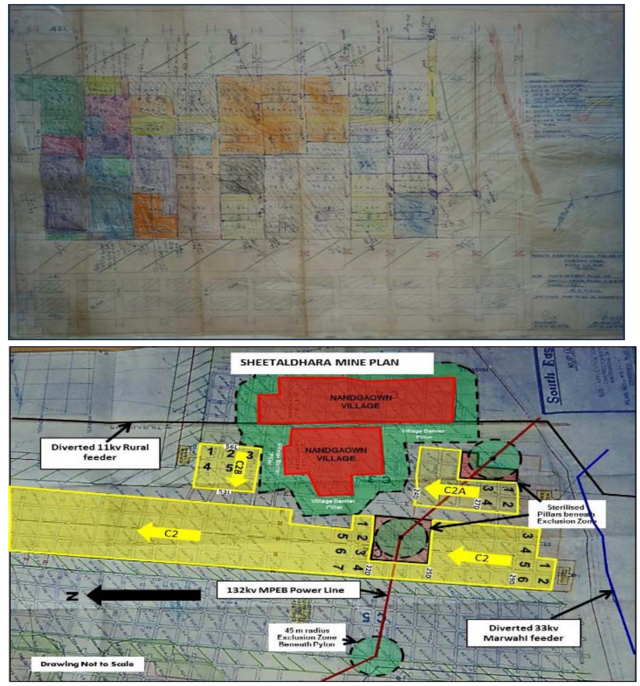

Figure 5. Hand plan of Kurja mine, SECL, showing the CM district.

5. Numerical Modelling Approach

Numerical modelling is a scientific approach and an efficient tool to determine operational safety and verify the support system in the mines. The depillaring method used in the Kurja mine is split and fender method, which is very popular in Indian coal mines. The irregular sizes of the rib and snook are formed by the depillaring method by CMT, which is very difficult to calculate the factor of safety (FoS). Hence, determining the stability of rib/pillar by numerical modelling becomes very easy. A study of Kurja mines is carried out to determine the actual stability of the area of rib and snook by numerical modelling. Using the laboratory determining strength and the RMR value (Bieniawski 1976), the corresponding rock mass strength parameters were scaled down from equations 2 through 8.

$$\sigma_1 = \sigma_{cm}\left(1+\frac{\sigma_3}{\sigma_{tm}}\right)^{bm} \tag{2}$$

$$\sigma_{cm} = \sigma_c exp\left(\frac{RMR-100}{20}\right) \tag{3}$$

$$\sigma_{tm} = \sigma_t exp\left(\frac{RMR-100}{27}\right) \tag{4}$$

$$b_m = b^{RMR/100} \tag{5}$$

Table 2. Geotechnical data is used in the numerical model

Kurja Mine, SECL RMR 48									
Layers	**Layer thickness (m)**	**Bulk Modulus K (GPa)**	**Shear Modulus G (GPa)**	**Modulus of Elasticity E (GPa)**	**Poisson Ratio**	**Cohesion (MPa)**	**Friction Angle (deg)**	**Compressive strength of rock mass (MPa)**	**Density (Kg/M³)**
Layer 1	5	3.80	2.28	5.7	0.26	2.17	37.44	55.80	2510
Layer 2	10	4.67	2.80	7.0	0.28	2.43	39.23	60.60	2270
Layer 3	60	3.20	1.92	4.8	0.26	0.85	42.73	38.20	2330
Coal	3.7	2.00	1.20	2.5	0.24	0.78	36.50	32.00	1420
Floor	60	3.50	2.10	5.25	0.25	1.38	34.88	53.50	2350

where, σ_1 is triaxial strength of rock mass (MPa), σ_c is compressive strength of intact rock (MPa), σ_{cm} is the compressive strength of rock mass (MPa), σ_{tm} is the tensile strength of rock mass (MPa) and b is the exponent in failure criteria. RMR is Bieniawski's rock mass rating.

$$\tau_{sm} = \left(\sigma_{cm}\sigma_{tm} \frac{b_m^{b_m}}{(1+b_m)^{1+b_m}} \right)^{1/2} \tag{6}$$

$$\mu_{0m} = \frac{\tau_{sm}^2 (1+b_m)^2 - \sigma_{tm}^2}{2\tau_{sm}\sigma_{tm}(1+b_m)} \tag{7}$$

$$\phi_{0m} = tan^{-1}\left(\mu_{0m}\right) \tag{8}$$

where, τ_{sm} is the shear strength of rock mass, ϕ_{0m} is the angle of internal friction.

For determination of the pre-mining stresses, the vertical stress (σ_υ) and the horizontal stress (σ_h) are calculated by the formulae (Sheorey 1994) given in equations no (9) and (10).

$$\sigma_\upsilon = 0.025H \tag{9}$$

$$\sigma_h = 2.4 + .01H \tag{10}$$

where, *H*= depth of cover in meters

For Indian coal measure rocks, Sheorey's failure criterion (Sheorey, 1997) has been found suitable (Mohan *et al.*, 2001). This criterion uses in the 1976 version of the RMR of Bieniawski. The basic CMRS- RMR value is used directly instead of Bieniawski's RMR since the failure criterion thus obtained has worked well in an Indian coal mine (Kushwaha *et al.*, 2010; Sinha, 2013).

5.1 Laboratory Experiments and Preparation of the Model

Physico-mechanical properties of coal measures such as UCS, modulus of elasticity, tensile strength, and poisson's ratio are vital input parameters used in numerical simulation. The samples were prepared on 54mm in diameter and 125mm in length for determination of UCS (L/D ratio is 2.5) and 54mm in diameter and 32mm in length (L/D ratio is 0.5) in the case of tensile strength. Tests were conducted as per ISRM suggested methods. All the inputs for geotechnical data are tested at the Rock mechanics laboratory of IIT (ISM), Dhanbad (Table 2).

Generating complex geometry and producing the finite-difference grid by the command-driven method could be pretty time-consuming (Sinha, 2015). So, the model is prepared in the AutoCAD (Figure 6a) and later imported in FLAC3D (Jawed and Sengupta, 2013) after a few processing in mesh generations.

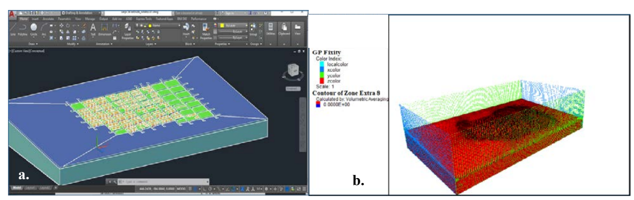

Figure 6. a. Model preparation in AutoCAD and b. The boundary condition of Kurja mine.

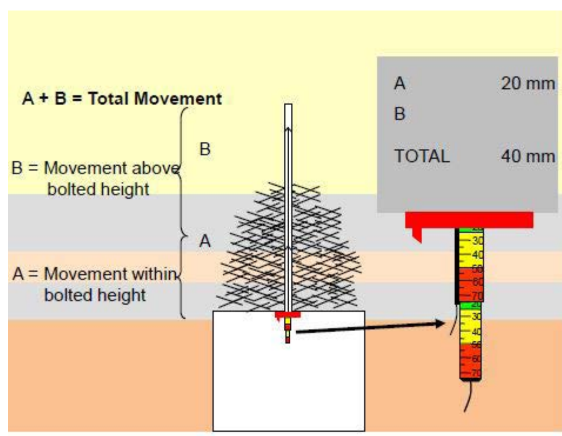

Figure 7. a. Dual height Tale-tell and b. Displacement difference between the numerical model and DHTT data of Kurja mine.

5.2 Calibration of the Numerical Model

The numerical model is calibrated by the roof displacement obtained from the numerical modelling, compared to the instrumentation data, i.e., Dual Height Tale-Tell (DHTT). DHTT gives the reading of the vertical strata movement by two different anchors. The length of the one anchor is usually above the bolted height, and another is below bolted height shown in Figure 7a. This instru-

Table 3. A comparison of the safety/stability factor between ARMPS and the Numerical model

Sl. No.	Name of the Mine	Parameters	Values are in a meter	ARMPS (USA) FOS/ Stability Factor	Avg. FOS from Numerical Modeling
1	Kurja Mine. SECL	Depth of Cover (D)	180	3.22	3.35
2		Hight of extraction (h)	3.7		
3		Gallery width (W)	6		
4		Pillar Size (C to C)	35 × 35		

ment provides a visual indication of the movement of roof strata in the opening of a coal seam. The cut-off values are also designated on the instruments, warning of possible roof failure. It can be observed from Figure 7b that the vertical displacements recorded at the monitoring points by the DHTT and the displacement results of the numerical modelling by the FISH programming follow the same trend, and their values match within a tolerable error.

5.3 Determination of Fosin a Developed Pillar by Numerical Modelling

The modelling procedure consists of grid generation, appropriate material behaviour, insertion of material properties, in situ stresses (Equations. 9–10), boundary conditions (Figure 6b), the convergence of the model to the equilibrium. The rib should have sufficient residual strength to carry the load, but it should not be strong enough to delay the caving. The factor of safety (SF) is defined as: -

$$SF = \frac{\sigma_{1m} - \sigma_{3mi}}{\sigma_{1mi} - \sigma_{3mi}} \text{ for } \sigma_{3mi} < \sigma_{tm} \tag{11}$$

$$SF = \frac{\sigma_{tm}}{\sigma_{3m}} \, for \, \sigma_{3mi} > \sigma_{tm} \tag{12}$$

where, σ_{1mi} = induced major principal stress (MPa) of the rock mass, σ_{3mi} = induced minor principal stress (MPa) of rock mass obtained from the numerical model. Subscript' m' stands for the rock mass.

Then restoring the in situ models and development of the gallery, and finally extracting the pillars. A relation between bulk modulus (K) and shear modulus (G) with young's modulus (E) and Poisson's (í) ratio is shown in equations (13) and (14).

$$K = \frac{E}{3(1-2\ )} \tag{13}$$

$$G = \frac{E}{2(1+\ )} \tag{14}$$

The Youngs modulus of the intact rock of coal measures was converted to rock mass value. An empirical relation given by Mitri *et al.* (1995) to relate the rock mass deformation modulus E_m is given in equation (15) was utilised in the numerical models.

$$\frac{E_m}{E_r} = \frac{1 - COS\left(\pi \times \frac{RMR}{100}\right)}{2} \tag{15}$$

where, E_m= rock mass deformation modulus, E_{r} = intact rock deformation modulus, RMR = rock mass rating, and $\pi \times \frac{RMR}{100}$ is expressed in radians.

A comparison of the FoS determined by the ARMPS (NIOS)) software and the numerical modelling of the developed gallery of Kurja mine is shown in Table 3. The FOS from the numerical model was 3.35, and the ARMPS said 3.22 (Figure 7), and their values match within a tolerable error.

Table 5. The factor of safety of the rib/snook pillar 1, 2,3,4,5 of Kurja mine

Name of the Mine	Rib and Snook area (m2)		Panel Name	Avg. stress from the numerical modelling (Mpa)	Strength by the CMRI strength formulae (Eqn.)	Factor of Safety
Kurja Mine	1.	26.78	Panel C, Sub I	4.39	2.57	0.6
	2.	63.01		5.69	5.37	0.94
	3.	52.22		5.86	5.12	0.87
	4.	102.56		3.09	6.5	2.1
	5.	70.95		5.02	5.68	1.1

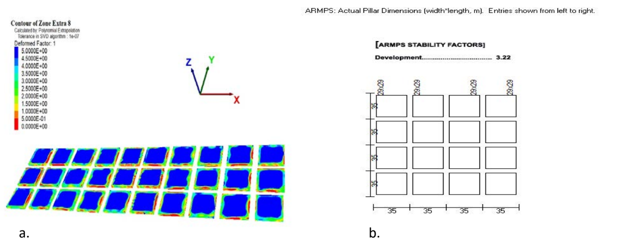

Figure 7. Factor of Safety contour and ARMPS result during the development of Kurja mine.

The FoS contour of the developed gallery is shown in Figure 7a, and the ARMPS stability factor of the developed panel is shown in Figure 7b.

5.4 Determination of FOS of the RIB by Numerical Modelling during Depillaring in Kurja Mine

Due to increased tensile strength at the top of the roof, failure occurs, and the immediate roof starts to fail as a crack. An immediate roof is just like a cantilever, and failure of the roof is ongoing after gradually increased extraction during development or depillaring.

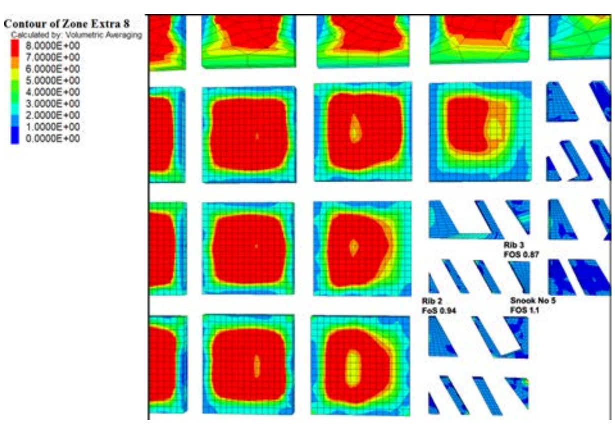

Figure 8. FoS contour in the depillaring panel of Kurja mine.

The factor of safety may be defined as strength/ stress, and it is decided whether a pillar or rib or snook fails or not. It is also calculated from the average model stress by fish programmed (Figure 8) of individual rib and snook and empirically calculates the strength by formulae given by Sheorey equation (16). A chart (Table 4) to clear understanding of various FoS is bellowed:

Table 4. Range of factors of safety to determine the stability

FOS of pillar/rib/ snook	Stability
$2 \geq$ FOS	Long term stability may be treated as indestructible.
FOS = 1-2	Short term stability, i.e., it may fall within a few years.
$0.6 \geq$ FOS	Stable for a few days

6. Result and Discussion

Numerical models can incorporate the effects of different geological conditions, joints, parting planes, complex geometries and in-situ stresses. However, they are limited by the need for realistic estimates of many in-situ material properties that are difficult to determine. The ratio of a pillar's estimated strength to the pillar's stress is expressed as the factor of safety (FoS). The nominal FoS for a pillar's design is dependent on the consequence of the failure of that pillar. For the long-term stability of underground structures of a developed coal seam, the value of the safety factor of standing pillars should not be less than 1.5; instead, a safety factor value equal to 2 or more is preferable. To estimate the factor of safety of the remaining rib and snook, the stress is measured by the FISH function in FLAC3D. The strength of the pillar, rib and snook is calculated by using the pillar strength formulae given by Sheorey (1992).

$$S = 0.27\sigma_C h^{-0.36} + \left(\frac{H}{250} + 1\right)\left(\frac{w}{h} - 1\right) \tag{16}$$

where, S = Strength, h = height of the extraction, H = depth, σ_C = compressive strength of the coal. w = effective width. The relation proposed by Wagner (1974) based on servo-controlled insitu test is

$$W_e = 4A / C_p \tag{17}$$

where, W_e = effective width, A=Area, C_p = perimeter of the pillar

Based on the present experiences of the pillar recovery in the room and pillar method, the factor of safety of pillar and rib in bord and pillar depillaring works, a range between 0.6-1.0 is suitable to work with ribs as such ribs fail in 1-3 days (Jawed and Sinha 2018). Table 4 shows the FOS range and the stability of the rib and snook in a mechanised depillaring panel. With proper planning, the factor of safety of ribs can be adjusted suitably in the range of 0.6 to 1.0 to enable a faster rate of extraction (Sinha 2013; Yejerla and Agrawal 2016).

The FOS compared with the different sizes of rib and snook of Kurja mines. The result compared, and the size of the rib of Kurja mine is 26.78m^{2,} and the FOS came 0.6 (**Table 5**). The pillar was not stable enough and failed within 2days. The other rib sizes of the Kurja mine is 52.22 m^2 63.01m^2; the factor of safety are 0.87, 0.94. The solid pillar is stable enough and can stay for a few weeks without any failure, getting from the numerical modelling and the combination of the empirical formula and finding the field rib is strong enough for a few months, which has validated the model. The snook size of the Kurja mine is102.56m^2 and 70.95m^{2,} and the FoS is 2.1 and 1.1, respectively. In actual conditions, the rib numbered 2, and 3 (Figure 8) of the depillaring panel C (Sub-I) of Kurja mine failed after three weeks.

Furthermore, it says that the 5th no snook may fail in a few weeks. In actual conditions, the 5th numbered snook was failed after 28days. Therefore, it can be said that the numerical approach to determine the pillar load is a valid and reasonable result to understand the actual behaviours of rib and snook pillars in any underground mines.

7. Conclusions

Most coal pillars are designed using empirical formulae to determine their strength. However, these can give misleading results in complex geo-mechanical environments and for pillars that are not square in the CMT. The load over the rib and snook pillar is essential to measure during the depillaring operation. In a few cases, the operators or miners engaged with the operation, but the inevitable failure of the rib pillar can damage the men and the machinery. The Factor of Safety (FoS), determined by the numerical model, indicates rib and snook behaviours. The field observation of the individual rib and snook gives the actual conditions of the panel. The actual condition of the rib and snook are as per the FOS said. This paper emphasised the particular rib pillar and the snook pillar load determination to prevent abnormal failure. Many kinds of research are going on in the regular shape rib to determine the FOS; it is complicated to calculate the FoS of irregular rib and snook of the mechanised depillaring panel. As underground mass production techniques by continuous miners have become very popular, this study may help determine the safe workings.

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