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A safe depillaring design for shallow depth of cover with influence of surface ground movements: a study in Jharia Coalfield

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Abstract

Extraction of coal from a shallow depth of cover especially under multi-seam mining condition requires intensive precautions in respect to safe working and impact on surface structures. A detailed geotechnical investigation is imperative for the design of a suitable support system during depillaring operation. A study has been exercised in Maheshpur colliery of Jharia Coalfield to assess the stability between the parting of VIIIA and VIIIB seams through numerical modeling. The support systems have been framed for split galleries, slices, and goaf edges with the help of well-established Central Mining Research Institute-Indian School of Mines rock mass rating (CMRI-ISM RMR) and NGI-Q system. A suitable method of depillaring has been designed keeping in view the safety of surface structures in respect to subsidence.

Keywords RMR · RQD · Rock load · Support design · Subsidence

Introduction

Control of strata movement is one of the vital thrust areas for safe workings in underground mines. Roof fall generally takes place due to detachment of lower strata since the process of re-distribution of stresses takes place around the excavation made. Around 61.1% of the incidences are due to roof fall, which accounted for 28.5% of total fatalities (Sinha et al. 2003). Thus, a proper support design of mine opening is essential for the stability of the roof and safe workings (Anon. 1990).

Surface subsidence from underground mining activity is an unavoidable problem. Surface subsidence is an important environmental consideration for active mine operations. It is an inevitable consequence of underground mining—it may be small and localized or extend over large areas; it may be immediate or delayed for many years as stated in SME (Hartman 1992).

The sickle-shaped Jharia Coalfield occupies an area of about 453.24 km^2, extending for a maximum about 38 km east-west and 19 km north-south and is located in Dhanbad

district of Jharkhand state (Singh 2007). Maheshpur colliery is located at Govindpur area of Jharia Coalfield. A panel "C" in VIIIA seam, located at an average depth of 54 m and having a dimension of 160 m × 105 m, is proposed to be extracted by room and pillar method. A panel in the overlying VIIIB seam having a parting of 12 m from VIIIA seam was depillared with caving as shown in Fig. 1. The topmost IX/X seam, partly extracted by opencast method above the proposed panel "C," was filled with overburden dump up to a height of 20 m above the surface. The subsidence study was also carried out to assess the safety of the surface structures.

Geotechnical parameters

The immediate roof above VIIIA seam was sandstone; thickness varied from 10 to 15 cm. The gallery width varied from 3.6 to 4.8 m. The geological features like joints and slips were observed in the roof. Joints were random in tight condition and filled with foreign materials. Two sets of cleat were found in coal seam. Groundwater condition was dry.

CMRI-ISM RMR and NGI-Q system

CMRI-ISM RMR system (Anon. 1987) and NGI-Q system (Barton et al. 1974) were used for the classification of roof rocks for galleries and splits and design of support in

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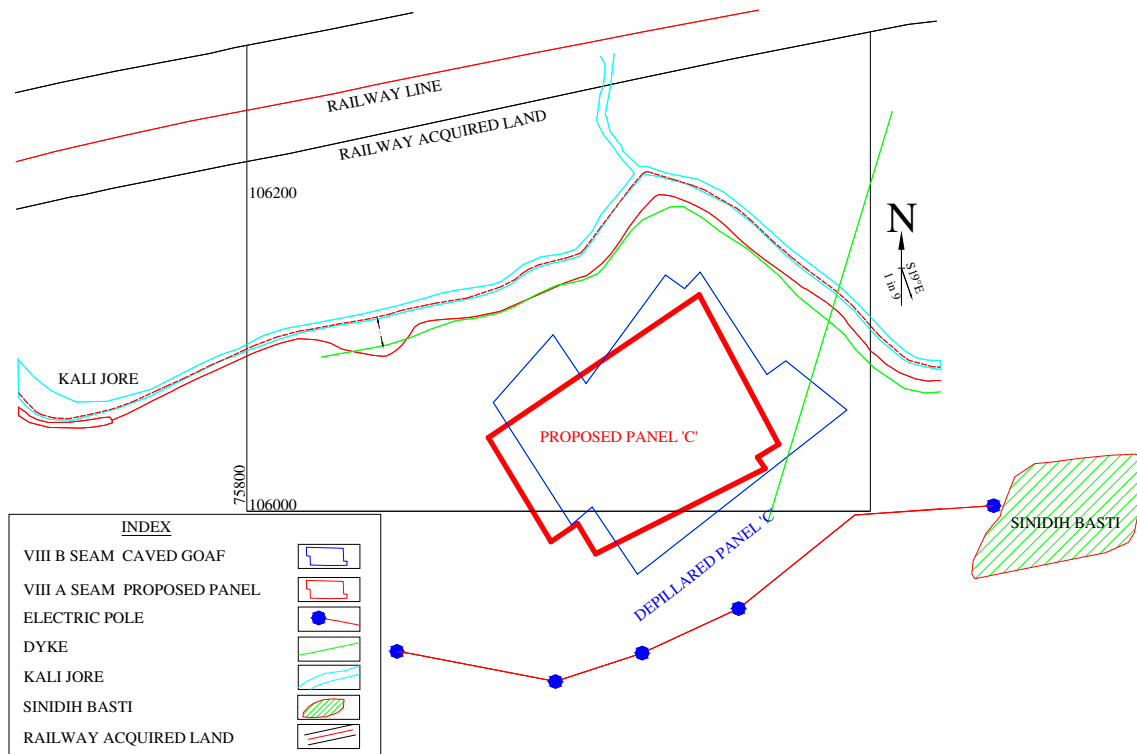1 CSIR-Central Institute of Mining & Fuel Research, Dhanbad 826015, India

Fig. 1 Proposed panel with overlying goaf and surface features

depillaring areas. Rock mass rating (RMR) determined by CMRI-ISM Geomechanical Classification System is the summation of the ratings of five individual parameters (Venkateswarlu et al. 1989). It provides useful information for selection of support system and rock load values are used for optimum support design. The RMR values of the immediate roof above VIIIB seam determined based on the existing geo-mining condition is given in Table 1.

The net RMR rating of 48.6 was obtained after 10% adjustment due to solid blasting and adjacent workings, i.e., the value comes under Class III A category (fair roof condition). The rock mass quality (Q) was determined using Q system using the following relation:

$$Q = \left[\frac{RQD}{Jn}\right]\left[\frac{Jr}{Ja}\right]\left[\frac{Jw}{SRF}\right] \tag{1}$$

Table 1 RMR of immediate roof determined for VIIIB seam

Parameter	Description	Rating
Layer thickness	12 cm	17
Structural features	Joints/slips (indices = 08)	14
Weatherability (1st cycle slake index)	96.41%	13
Compressive strength	266 kg/cm^2	06
Groundwater	Dry	10

where

RQD	rock quality designation,
J_n	joint set number,
J_r	joint roughness number,
J_a	joint alteration number,
J_w	joint water reduction factor and
SRF	Stress reduction factor.

The RQD was estimated using empirical relation developed between bedding frequency (number of bedding planes per meter thickness of strata) and RQD by Priest and Hudson (1976) and volumetric RQD (number of joint sets in 1 m volume of rock mass) as given in Eqs. 2 and 3.

$$RQD = 100(0.1\lambda + 1)e^{-0.1\lambda} \tag{2}$$

$$RQD = 115\text{–}3.3 \ J_V \tag{3}$$

where

λ bedding frequency

The value of RQD obtained by Eqs. 2 and 3 were 79.66 and 77% respectively and the later was taken for determination of Q value. The measured value of J_V was 11.5. Other parameters were as follows: J_n = joint sets and random; J_r = planer joint; J_a = unaltered joint walls with surface staining; J_w = groundwater condition was moist/dripping. SRF was taken as 5 for moderately high-stress concentration slices and 10 for high-

stress concentration, i.e., at the goaf edge. For estimation of Q, ratings were allocated to parameters as given in Table 2.

Rock load

Rock load in the splits in depillaring areas was determined for immediate roof using CMRI-ISM RMR system from the following empirical relations:

Rock load in gallery/split (t/m^2)

$$= B.D. (1.7 - 0.037 \, RMR + 0.0002 RMR^2) \quad (4)$$

Rock load at junction (t/m^2)

$$= 5.B^{0.3}.D.(1 - RMR/100)^2 \quad (5)$$

where B = width of splits = 4.2 m; D = rock density = 2.38 t/m^3; and RMR = 48.6.

The rock load in split and junction was found to be 3.74 and 4.83 t/m^2 respectively. Rock load (P_{roof}) in slice and goaf edge estimated using Q System was found to be 6.98 and 8.79 t/m^2 respectively using following relation:

$$P_{roof} = (2/3) * \left(Jn^{1/2}/J_r \right) * (5Q)^{-1/3} \quad (6)$$

where Jn = 6, Jr = 1, Q = 2.56 for slice, and Q = 1.28 for goaf edge.

Numerical modeling

"Rib-and-slice" depillaring method was proposed in VIII A seam, the most commonly practiced pillar liquidation method in India. The proposed width of level split and slice were 4.2 and 4.0 m respectively with a rib width of 2 m (minimum) for a 14 m × 14 m (corner to corner) pillar size. The number of slices varied depending upon the pillar size. Caved goaf between worked out seam VIIIB and overburden dump was 48 m and there was 10 m sandstone parting between VIIIA seam and VIIIB seam. The thickness of VIIIA seam was 3.0 m.

A numerical modeling study was conducted by 2D Finite Difference Method (2D FDM) of simulation (FLAC 3D, 1997) to assess the factor of safety of the parting between the pit bottom and the roof contact of VIIIB seam under the influence of the overburden dump during development and depillaring operation. The overburden dump and the caved

strata up to VIIIB were considered as a dead load. The vertical and horizontal in situ stresses were estimated from the following equations:

$$\sigma_v = 0.025 \, H \quad (7)$$

$$\sigma_h = [\upsilon/(\upsilon-1)] \, \sigma_v + [\beta \, E \, G/(1-\upsilon)](H + 1000) \quad (8)$$

where

σ_v in situ vertical stress in MPa
σ_h in situ horizontal stress in MPa
H   depth of cover in m
β thermal expansion coefficient = 3×10^{-5} per °C
G   geothermal gradient = 0.03 °C/m
E   modulus of elasticity = 2000 MPa
   Poisson's ratio = 0.25

The lithology of the site was taken from the litholog. The average physico-mechanical properties of rock types are given in Table 3.

Sheorey (1997) adopted Balmer's criterion for intact rock after applying it to 201 triaxial data sets for different rocks including coal. This criterion is expressed as:

$$\sigma_1 = \sigma_c \left(1 + \frac{\sigma_3}{\sigma_t} \right)^b \quad (9)$$

where

σ_1 major principle stress, MPa
σ_3 minor principle stress, MPa
σ_t tensile strength for intact rock, MPa
σ_c uniaxial rock mass compressive strength for intact rock, MPa
b exponent in failure criterion for intact rock

The equation for the rock mass is expressed as

$$\sigma_1 = \sigma_{cm} \left(1 + \frac{\sigma_3}{\sigma_m} \right)^{bm} \quad (10)$$

where

σ_{cm} uniaxial rock mass compressive strength, MPa
σ_{tm} rock mass tensile strength, MPa
b_m exponent in failure criterion for rock mass

These constants are related to *RMR* of Bieniawski (1976) as

$$\sigma_{cm} = \sigma_c \exp \left(\frac{RMR - 100}{20} \right) \quad (11)$$

$$\sigma_{tm} = \sigma_t \exp \left(\frac{RMR - 100}{27} \right) \quad (12)$$

Table 2 Estimation of rock mass quality

Parameters	RQD	J_n	J_r	J_a	J_w	SRF		Q	
						Slice	Goaf edge	Slice	Goaf edge
Value	77	6	1	1	1	5	10	2.56	1.28

Table 3   Input parameters for numerical simulation

Rock type	Dry density (kg/m³)	UCS (MPa)	Tensile strength (MPa)	Young's modulus (GPa)	Poisson's ratio
Caved goaf	1700	–	–	0.7	0.10
Medium grained sandstone (parting)	2500	41	4.6	6.9	0.25
Coarse grained sandstone (parting)	2384	26	3.6	5.5	0.25
Coal	1813	21	1.9	3	0.25
Floor	2420	41	4.6	6.9	0.25

$$b_m = b^{\frac{RMR}{100}} \quad b_m < 0.95 \tag{13}$$

Safety factors were evaluated for each and every element in the numerical modeling to estimate the stability or instability of the rock mass, defined as:

$$F = \frac{\sigma_1 - \sigma_3}{\sigma_{1i} - \sigma_{3i}} \tag{14}$$

except when $\sigma_{3i} > \sigma_{tm}$

$$F = \frac{\sigma_m}{-\sigma_{3i}} \tag{15}$$

where, σ_{1i} and σ_{3i} are the major and minor induced stresses from numerical model output.

On creating a 25-m-wide span, the factor of safety of the immediate roof of coarse-grained sandstone (4 m thick) was less than 1.0 (Fig. 2). Hence, the local fall may occur at this stage. However, the factor of safety of the parting above this layer was more than 1.0; therefore, failure of the total parting may not occur. The extraction at this juncture should not be stopped or else the parting may fail with increased duration. With a 35-m-wide span, the safety factor was found to be less than 1.0 indicating roof failure with caving. Main fall can be expected with the advancement of face up to 35 m, and thereafter, regular fall will take place (Fig. 3). The parting between VIIIA seam and the caved goaf remains stable till the main fall. After the main fall, the goaf will be compacted with caved material and will not allow this parting to fail with further face advance.

Design of support

For split galleries

Split galleries of 4.2 m wide should be supported with four full vertical column grouted rock bolts with slow set resin capsule by keeping a bolt spacing of 1 m row spacing and 1.5 m along row. The two side bolts should be at a distance of 0.60 m from the pillar edge (Fig. 4). The 20 to 22 mm diameter ribbed TMT steel rod rock bolt should be 2.1 m long. The hole diameter should not be more than 32 mm. Four full column grouted rock bolts in a row at 1 m spacing and the row spacing should be 1.05 m at junctions. The total number of roof bolts should be 20 (Fig. 4).

For 4.2 m wide split, every 1.5 m length of the roadway would be supported by four roof bolts. A 15-ton anchorage strength of resin bolt is taken. The support resistance offered by this support system would be

$$\text{Support resistance} = \frac{(4 \times 15)}{(4.0 \times 1.0)} = 9.52\,\text{t/m}^2$$

Hence, the safety factor = 9.52 / 3.74 = 2.55.

For slices

Slices of 4.0 m wide should be supported with four full column grouted roof bolts at 1 m interval leaving 0.5 m space
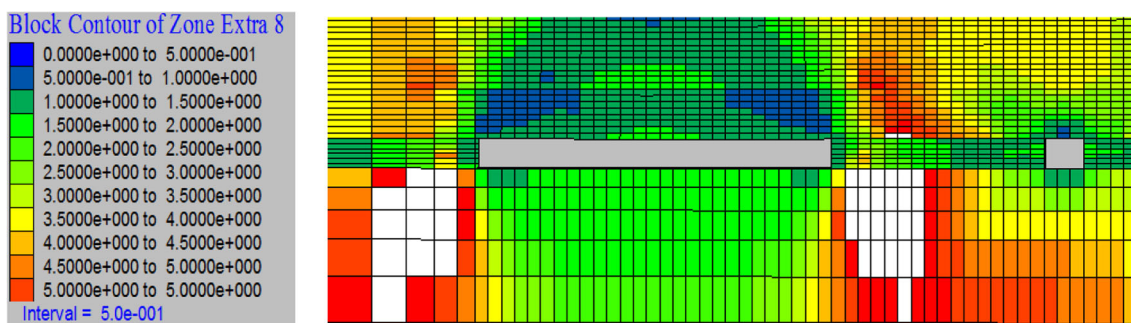

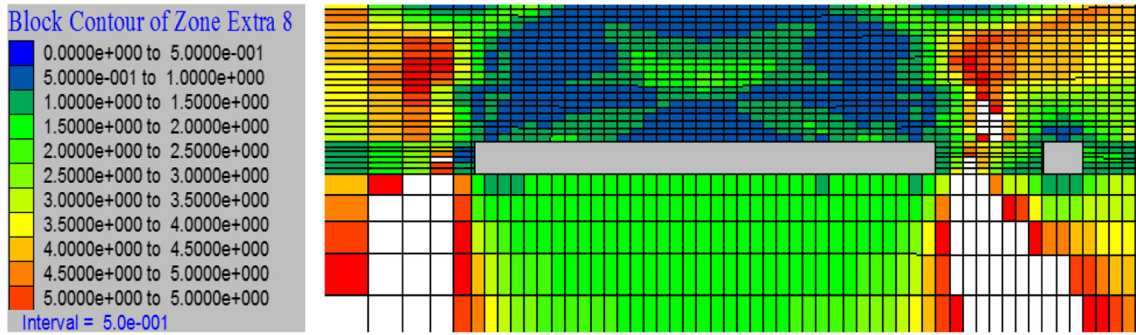**Fig. 2** Block contour of safety factors after one pillar extraction (25 m)

Fig. 3 Block contour of safety factors after one and half pillar extraction (35 m)

towards the pillar in grid pattern. The spacing of row should be 1.0 m. The rock bolt should be 2.1 m long made of 20 to 22 mm diameter ribbed steel rod (Fig. 4). The support resistance offered by such support systems would be as follows:

$$\text{Support resistance} = \frac{(4 \ \text{x} \ 15)}{(4.0 \ \text{x} \ 1.0)} = 15 \, \text{t/m}^2$$

Hence, the safety factor = 15/6.98 = 2.15.

For goaf edges

The goaf edges should be supported with two rows of wooden cogs and roof bolts till main fall. Four full column grouted roof bolts in a row at spacing of 1 m should be installed

vertically. The row spacing should be 0.5 m. Three number of cogs of 1.2 m × 1.2 m with seasoned timber, skin to skin, should be set at all goaf edges (Fig. 4). The purpose for two rows of roof bolt at goaf edges ahead of chocks/cogs is to provide additional support and to make a prominent breaker line to avoid overriding or encroachment. This will also facilitate in removal of vertical support during shifting of the goaf edges. The support resistance offered by the support system with eight roof bolts and three cogs would be as follows:

$$\text{Support resistance} = \frac{(8 \ \text{x} \ 15) + (3 \ \text{x} \ 30)}{(4.2 \ \text{x} \ 1.7)} = 29.41 \, \text{t/m}^2$$

Hence, the safety factor = 29.41/8.79 = 3.34.

Fig. 4 Support design in Rib and Slice method of pillar extraction at Maheshpur colliery
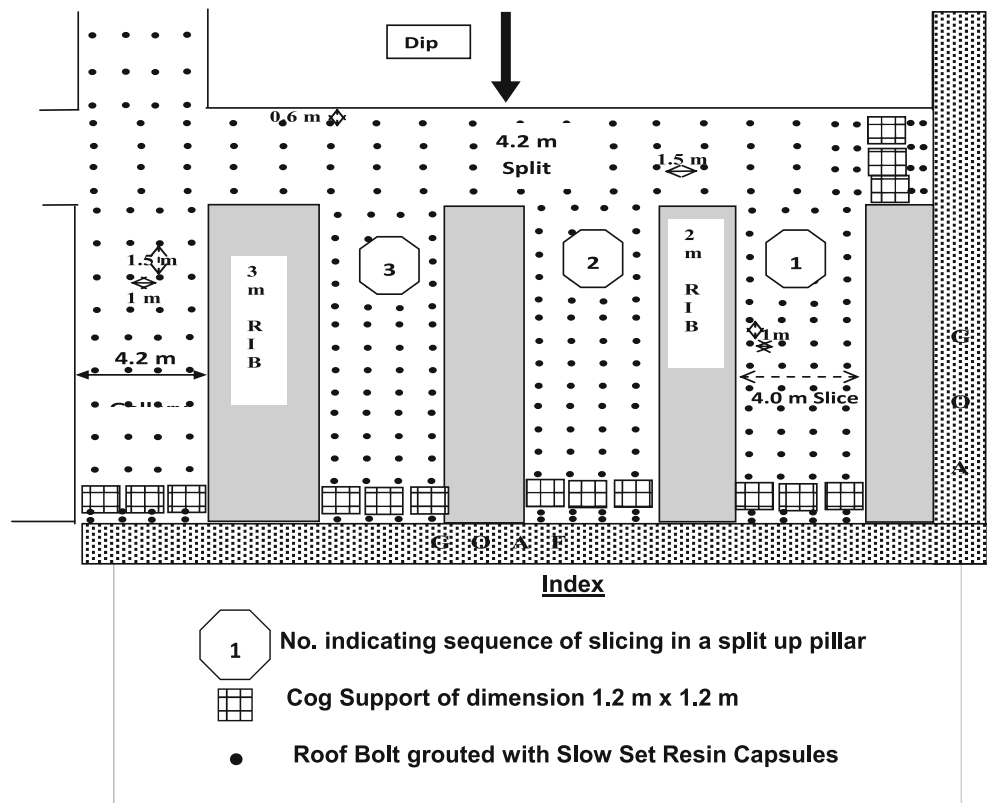
Subsidence studies

The subsidence study was carried out to assess the safety of the surface structures like Kali jore, railway line, Sinidih Basti, and high-tension electric poles, over and around the vicinity of the proposed panel "C" (Fig. 1). There was a dike on the eastern side of the panel. There was also an overlying caved panel in VIIIB seam, with a parting of around 10 m. The outcrop IX/X seam was mined by opencast and presently filled with overburden dump. Hard cover was considered as the depth of the seam from the surface for subsidence prediction. It was proposed to extract 3.05 m height with 85% extraction by caving. The average depth of the panel was 54 m.

Subsidence prediction

Subsidence prediction was done with the help of 3D subsidence prediction model using modified influence function method, validated from subsidence observations from various Indian coalfields (Anon. 1999 and Sheorey et al. 2000). This method is suitable for all types of depillaring workings, irrespective of shape, size, or different rates of retreat of extraction in successive seams. Each infinitesimal (small) sector "dA" is regarded as an extraction element and has its own weighting factor according to its placement in the influence circle. An influence circle has radius $H\tan\theta$ where H is depth of working and θ is angle

Table 4 Anticipated subsidence, slope, and strains at surface

Seam	Subsidence (mm)	Slope (mm/m)	Compressive strain (mm/m)	Tensile strain (mm/m)
VIIIA	2065.31	59.71	25.76	24.08
VIIIA + VIIIB	4164.84	122.19	52.47	35.49

of draw (Fig. 5). The following is the modified influence function used in CIMFR—subsidence software.

$$K_z = \frac{0.532}{R^2}\left(1 + \cos\frac{\pi r}{R}\right) \tag{16}$$

where

K_z   influence function for subsidence,
R   radius of influence circle and
r   radial distance of sector centroid from center of influence circle

$$\iint_{A_0} K_z dA = 1 \tag{17}$$

where, A_0 is the area of influence circle.

The subsidence at point P on the surface or a chosen plane of reference is simply the sum of multiplications of weighting factors and maximum possible subsidence (S_{max}), considered

Fig. 5 Illustration of influence function method (Sheorey et al. 2000)
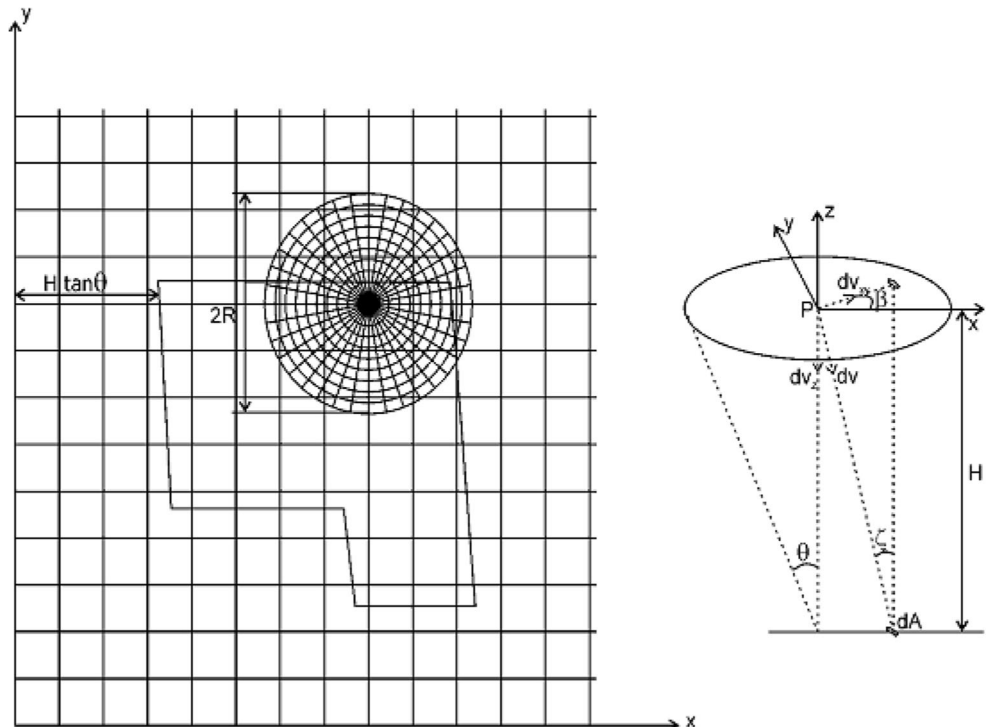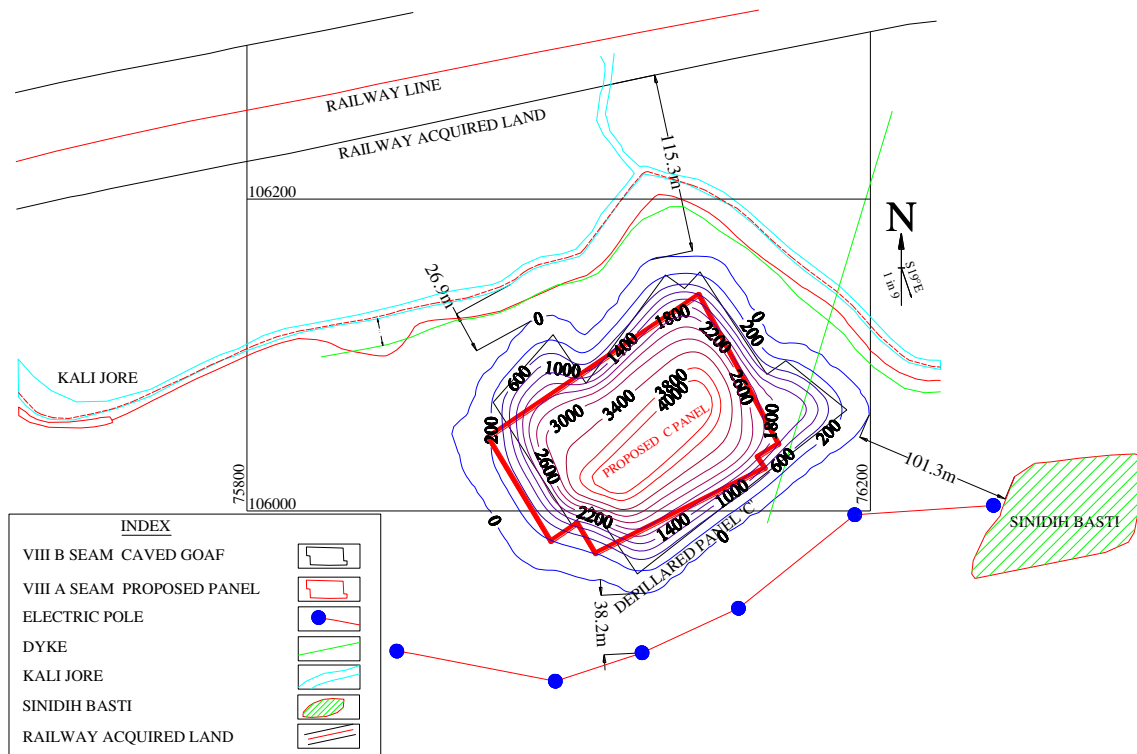
Fig. 6 Anticipated cumulative subsidence contour at surface

all the extraction elements of the proposed extraction panel. S_{max} is given as follows:

$$S_{max} = a.m.e \qquad (18)$$

where

a   subsidence factor
m   height of extraction and
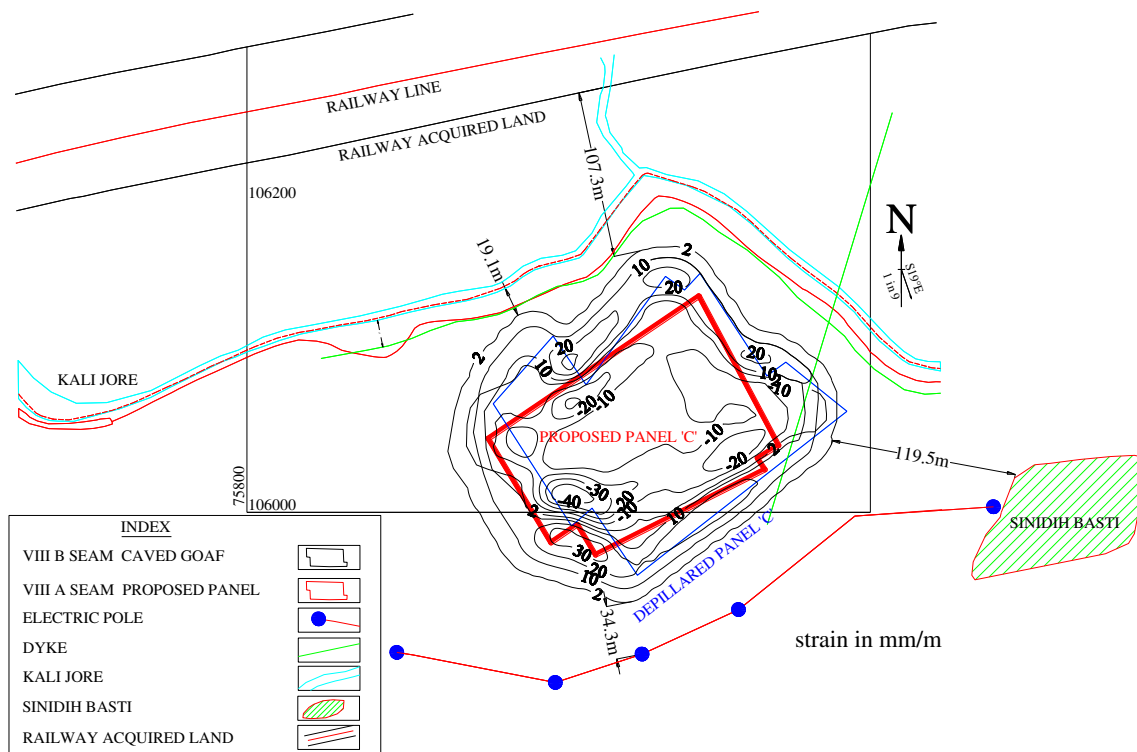e   percentage of extraction expressed in ratio

Fig. 7 Anticipated cumulative strain contour at surface

Table 5 Distance of surface features and structures from subsidence zone

Surface features and structures	Distance from subsidence influence zone (m)	Distance from 2 mm/m strain contour influence zone (mm/m)
Boundary of railway acquired land	115.3	107.3
Kali jore	26.9	19.1
Electric pole	38.2	34.3
Sinidih basti	101.3	119.5

The subsidence factor of 0.81 was taken in the prediction model against maximum possible subsidence due to multi-seam extraction. A Non-Effective Width (NEW) of 0.20 times the depth of extraction and angle of draw of 25° (multi-seam) on the positive side was considered in the model.

The magnitude of anticipated subsidence, strains, and slope due to extraction of panel "C" of VIIIA seam at surface was predicted at every 10 m grid point. Cumulative subsidence movements due to the effect of overlying working in VIIIB seam were also computed as detailed in Table 4.

The anticipated cumulative subsidence and strain contours at the surface after extraction of panel C of VIIIA seam is given in Figs. 6 and 7 respectively. The predictive model showed that magnitude of maximum subsidence, maximum slope, maximum compressive, and tensile strains at the surface due to extraction of 3.05 m, 2066.31 mm, 59.71 mm/m, 25.76 mm/m, and 24.08 mm/m respectively. These strain values were high enough to develop cracks with wide openings. The low value of H/me (< 0.3) indicates discontinuous type of subsidence, i.e., development of step-like formation along the subsidence profile. The location of surface structures and features were beyond the influence of subsidence and strain zone as clearly depicted in Fig. 5 and Fig. 6 respectively, the distance of which is given in Table 5.

Conclusions

The result of the numerical model indicated that during development, no failure was anticipated either in the parting or in the pillars as it was found during the field investigation also. Local fall may occur during depillaring up to an opening of a 25-m-wide span but failure of parting may not occur as factor of safety was found to be above 1 in above layer. Main fall was expected after 35 m wide opening and the parting between VIIIA seam and the caved goaf will remain stable till the main fall. Support for split, slice and goaf edge was designed with a factor of safety of 2.55, 2.15 and 3.34 respectively. The maximum anticipated subsidence, slope, compressive, and tensile strains at the surface were 4164.84 mm, 122.19 mm/m, 52.47 mm/m, and 35.49 mm/m respectively. Development

of cracks around the periphery of the edges of the panel with wide openings was anticipated. Surface structures and features were beyond the zone of influence of subsidence and strain and thus will not cause any effect due to mining-induced subsidence.

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