ANALYSIS OF SUBSIDENCE AND STABILITY OF PILLARS IN A COAL MINE

A THESIS SUBMITTED IN PARTIAL FULFILMENT OF THE REQUIREMENTS FOR THE DEGREE OF

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By

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CERTIFICATE

This is certified that the thesis entitled "**ANALYSIS OF SUBSIDENCE AND STABILITY OF PILLARS IN A COAL MINE"** submitted by **Mr Sanjay Kumar,** Roll No. **111MN0393**, in partial fulfilment of the requirement for the award of Bachelor of Technology degree in Mining Engineering at the National Institute of Technology, Rourkela (Deemed university) is an authentic work carried out by him under my supervision and guidance.

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ABSTRACT

The significance of mining will be certainly immense to human development. Truly, as one of the biggest of human endeavours, Mining and its improvements relate nearly with social progress. One of underground mining methods is the **Bord and Pillar Method** of Mining and this method is one of the oldest Mining Methods. By the help of Bord and Pillar Mining, extraction will be high. The key to successful Bord and Pillar Mining is selecting the optimum pillar size and stability of the pillar. If the pillars are too small the mine will collapse. If the pillars are too large then significant quantities of valuable material will be left behind reducing the profitability of the mine. The issues relating to the stability of pillars and Subsidence from it will be a major concern now-a-days. The most critical parameter before outlining and stability of pillars will be the Safety component.

The primary objective of this project is to increase the extraction proportion with the help of stability of created pillars and subsidence control and Based on laboratory testing of coal samples, empirical and numerical modelling studies related to the stability of the pillar and analysis of observation data on subsidence profiles for underground coal mine, the following conclusions are done. Field observations at Kumda underground mine indicated stability of the pillars without any perceptible side spilling and crushing. Uniaxial compressive strength, triaxial testing of coal samples indicated UCS, cohesion(C) and internal friction angle(Ø) of 27.9 MPa, 1.85 MPa and 30^0 respectively. Empirical and numerical modelling using two dimensional continuum analyses indicated the maximum stress of 5 MPa over the pillar and safety factor exceeding 2 confirming to the qualitative observation of stability of the pillar. Maximum subsidence at 1.58 m was observed over the over the extracted panels with a subsidence factor of 0.63 for the depillaring panels with the width and depth of 64 m-14 m, 30 m-45.5 m respectively.

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CHAPTER 1

INTRODUCTION

1.0 INTRODUCTION

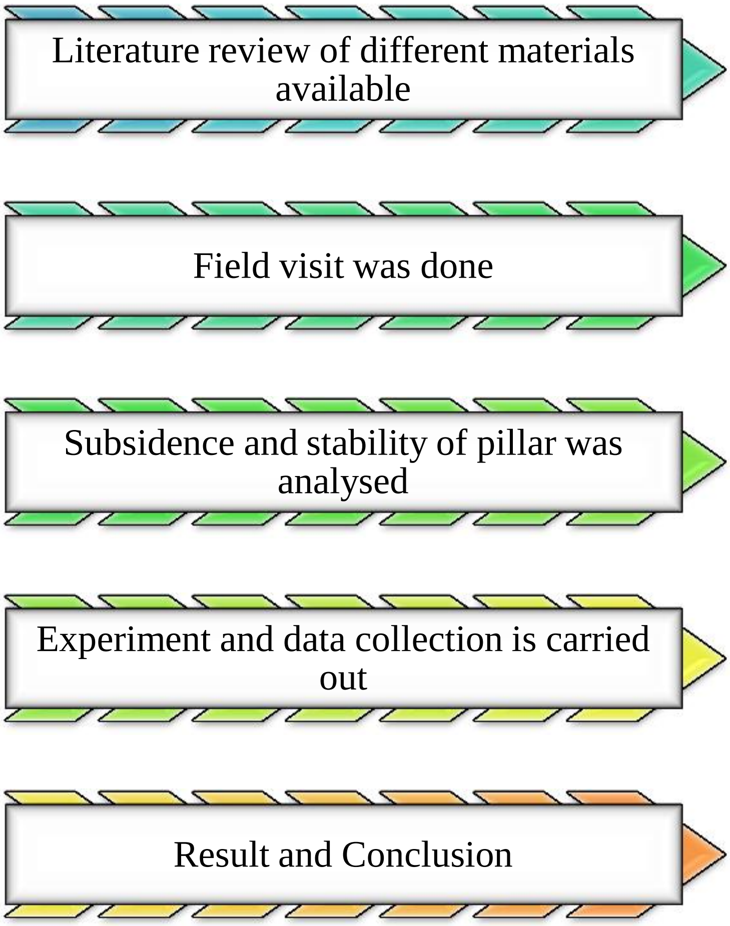
Mining is one of the most important sectors for the continuous development or growth of any nation. Mining is the economic extraction of valuable minerals like coal, iron talc, zinc, uranium etc. from earth for many purposes like generating electricity, pharmaceutical applications, infrastructures, cosmetic cream, etc. It provided a base for the civilization to grow in all its form and acted as an example for the other sector of industries to breed. Mining can be done in two ways, underground and open cast. Underground mining broadly consists of two types, a) Long wall method and b) Bord & Pillar method. The latter is predominantly followed in India compared to the former. Most of the coal mining is done by the Bord and Pillars Method.

1.1 Objective

- Stability analysis of subsidence in coal mine and Study of strength of the pillar, stress distribution over the pillar through empirical and numerical modelling.

1.2 Methodology

The methodology of the project was described in a flow chart format below:



CHAPTER 2

LITERATURE REVIEW

2.1 Basic Principle of Pillar Design

Basic Principle of Pillar Design Pillars are the natural structural member of a coal mine to support the roof and to assist in transferring the overburden load to floor for wider area dissipation. An optimum dimension/design ensures minimum coal being blocked in the pillar while maintain its stability. Nevertheless it continues to experience stresses throughout its life or till failure occurs. Pillar loading is of three types: preliminary loading or loading immediately following excavation of opening; subsequent loading or the abutment pressures (i.e. after a portion of coal has been extracted, the roof beds are detached and the beds above are relieved of the weight of the higher strata and the load which was originally acting vertically over the excavated area then deflects and bridges over the working area and transmits its weight forward to some region ahead of the coal face known as “front abutment” and backward behind the coal face at a region where the strata again make contact by subsidence of the higher beds known as “back abutment”) and progressive failure theory for post-mining loading. The “classic” methods consisted of three steps:

i. Estimating the pillar load using tributary area theory.

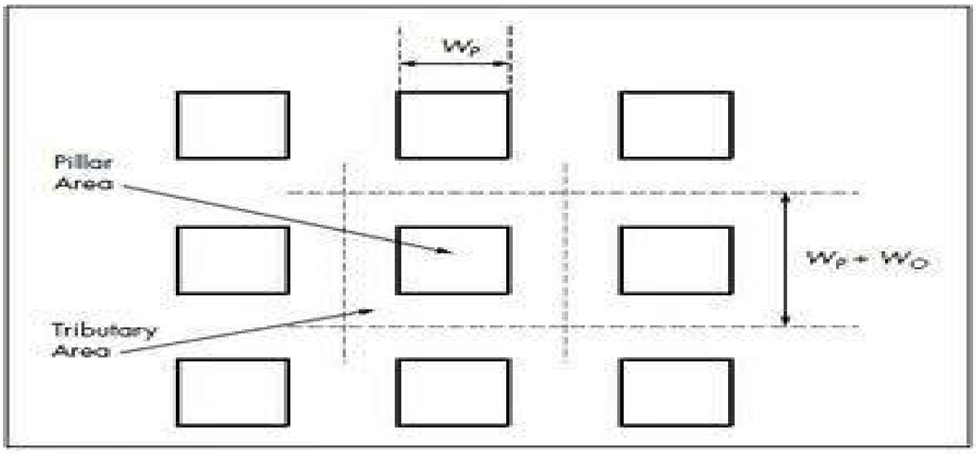
ii. Estimating the pillar strength using a pillar strength formula.

iii. Calculating the pillar safety factor.

Figure (1) Pillar Design

2.1.1 Estimating the pillar load using tributary area theory

Estimating the load was fairly straight forward for an industry that relied almost exclusively on room-and-pillar mining at relatively shallow depth. The tributary area estimate was considered sufficient, though it was recognized that in narrow panels the pillars near the edges might not experience the full load. More complex were the issues associated with pillar strength. The two big issues were the “size effect” and the “shape effect.” According to this concept, a pillar takes the weight of overlying rock up to a distance of half the opening width surrounding it. The theory assumes that each pillar carries a proportionate share of the full overburden load.



In the figure, W_o and W_p are widths of the opening and pillar respectively, while L_p is the length of the pillar. For square pillars, $W_p = L_p$

The load on the pillar, is $P = (L_p + W_o) \times (W_p + W_o) \times \gamma \times g \times h$

Where γ*g is the weight of the rock per unit volume, and h is the depth of mining.

The stress on the pillar $\sigma_p = \frac{P}{Area\, of\ Pillar} = \frac{(Lp + Wo) \times (Wp + Wo) \times \gamma \times g \times h}{(Lp + Wo)}$

$$= \frac{(Lp + Wo) \times (Wp + Wo) \times \sigma p}{(Lp + Wo)}$$

In case of inclined seams the formula for stress on the pillar is

$$\sigma_p = \frac{(Lp + Wo) \times (Wp + Wo) \times \sigma p}{(Lp + Wo)} \times (cos\theta + m\ sin\theta)$$

Where, θ = angle of inclination,

m = Poisson’s ratio, and

σ_v = vertical stress = γgh

The above equations indicate that the factors influencing pillar load are:

- Depth – the deeper the mining, the higher the load,
- Pillar width – the smaller the pillar, the higher the load,
- Bord width – the wider the bord, the higher the load,
- Extraction ratio- The higher the extraction, the higher the pillar load.

2.1.1.1 Limitations of the tributary area theory

1. Average pillar stress is calculated by assuming that pillars uniformly support the entire load overlying both the pillars and the mined-out areas.
2. Tributary area theory assumes regular geometry and ignores the presence of abutments. The effect of deformation and failure in the roof strata resulting from the mining operation are disregarded.
3. The concept does not take into account abutment stress distributions and deformation or failure of the pillar. Also, if there is displacement interaction between the surrounding strata and the pillar itself, stress may be redistributed within the system, resulting in a stress state significantly different to the theoretical state.
4. The average pillar stress is purely a convenient quantity representing the state of loading of a pillar in a direction parallel to the principal direction of confinement. It is not simply or readily related to the state of stress in a pillar that could be determined by a complete analysis of stress. The implicit assumption that the other components of the pre-mining stress field have no effect on pillar performance is not generally tenable. Furthermore the tributary area theory ignores the effect of the location of the pillar within a mine panel (Brady and Brown, 1993).
5. The tributary area theory is only valid for cases where the width of the panel is as great as or greater than the depth and where the pillars in a panel are of the same size. Other factors that have been found to influence the validity of the tributary area theory include the percentage extraction and the stiffness of the overburden (Van der Merwe, 1998).

2.1.2 Pillar Strength Formula

Numerous pillar strength formulas have been proposed, but five formulas are used most commonly (Bieniawski, 1984; Peng, 1986). Each formula specifies its own appropriate factor of safety. These are given below:

2.1.2.1 Obert-Duvall Approach (Obert and Duvall, 1967)

It was derived from laboratory tests on hard rock and elasticity considerations the same relationship as did Bunting (1911). Greenwald et al. (1939) mention that this form of an expression for pillar strength was proposed in 1900 for anthracite after laboratory tests made for the Scranton Engineers Club. This formula is given as

$$\sigma p = \sigma 1(0.778 + 0.222 \times \frac{W}{H})$$

Where, σ_p is pillar strength,
σ_1 is uniaxial compressive strength of a cubical specimen (w/h = 1),
and w and h are pillar dimensions in meters.

According to Obert and Duvall, this equation is valid for w/h ratios of 0.25 to 4.0, assuming gravity-loading conditions. Through back calculations from mining case histories and utilization of laboratory rock properties, safety factors of 2 to 4 were derived for short- and long-term pillar stability, respectively.

2.1.2.2 Central Mining Research Institute approach (now CIMFR, India)

Formula for pillar strength taking into account the pillar w/h ratio, the uniaxial compressive strength of the pillar, the height of seam and depth of cover. The developed equation is nothing but a reflection of the triaxial state of different stresses involved. It is given by:

$$S = (0.27 \times \sigma c \times h^{-0.36}) + [\frac{H}{160} \times (\frac{W}{H} - 1)]$$

Where, S = Pillar strength (in MPa),

σ_c = Uniaxial compressive strength, UCS (in MPa)

h = Working height or seam height (in m)

H = Depth of cover (in m) w = Pillar width (in m)

2.1.2.3 Bieniawski approach (1968, 1969)

This approach is based on large-scale in situ tests on coal pillars. Such tests were first undertaken in the United States by Greenwald et al. (1939) during the period 1933–1941. Extensive tests were conducted in South Africa during 1965–1973 by Bieniawski (1968, 1969), Wagner (1974), and Bieniawski and van Heerden (1975). Wang et al. (1977) conducted in the United States the largest test of all involving one full-sized coal pillar measuring 80 ft. (24 m) in width. All these investigations examined the various pillar-strength formulas

The general normalize form of the Bieniawski equation is $\sigma p = \sigma 1(0.64 + 0.36w/h)$ Where σp is pillar strength, w is pillar width (m), h is pillar height (m), and $\sigma 1$ is the strength of a cubical specimen of critical size or greater (e.g., about 3 ft. or 1 m for coal)

2.1.2.4 Holland & Gaddy approach

They extended the work by Gaddy (1956) and proposed the following formula:

$$\sigma p = k(\frac{\sqrt{W}}{h})$$

Where, k is the Gaddy factor,

w and h are pillar dimensions in in.,

σp is pillar strength in psi.

Holland specified a safety factor between 1.8 and 2.2 for the design of coal pillars, with a suggested value of 2.0. The width-to height ratio, for which the Holland formula is valid, ranges from 2 to 8. Although popular in the 1970s, the Holland-Gaddy formula is no longer recommended because it was found to be overly conservative at higher ratios (> 5).

2.1.2.5 Salamon and Munro (1967)

They conducted a survey of failed and standing coal pillars in South Africa. Based on the studies of Holland (1964) and Greenwald et al. (1939), they selected the following form of pillar strength to apply to square pillars

$$\text{Strength} = \mathrm{k} \times \mathrm{h}^{\mathrm{a}} \times \mathrm{w}^{\mathrm{b}}$$

The constants for the above equation were derived from a statistical survey of data reflecting actual mining experience. In all, 125 case histories were used, of which 98 were standing pillars and 27 were failed pillars (collapsed at the time of the analysis). In deriving a pillar strength formula, it was assumed that those pillars that were still intact had safe dimensions, while the collapsed pillars were too small.

The following pillar strength formula was proposed:

$$= 1.32 \times \mathrm{w}^{0.46}/\mathrm{h}^{0.66}$$

Where, σ_p the strength is in psi,

And the pillar dimensions, w and h are in feet.

The recommended safety factor for this formula is 1.6, the range being 1.31 to 1.88.

In si unit pillar equation becomes

$$\sigma p = 7.2 \times w^{0.46}/h^{0.66}$$

Where, σ_p the strength is in MPa while w and h are in meters.

2.1.3 Pillar Safety Factor

Factor of Safety (SF) is the ratio of strength of pillar and stress on pillar.

$$FOS = \frac{\sigma p}{Sp}$$

Where σp = strength of pillar and Sp= stress on pillar the above approach of pillar design incorporates the following assumptions:

- The seam is subjected only to vertical pressure, which is constant over the mined area. However, stress transfer occurs where stiff abutments exist in underground workings. Thus this vertical pressure may be relieved partially.
- Each pillar supports the column of rock over an area that is the sum of the cross-sectional area of the pillar plus a portion of the room area, the latter being equally shared by all neighbouring pillars. However, this is certainly not valid if the area of development is small since the pillars in the centre of the excavation are under more stress than the pillars close to the sides. It is usually only accepted as valid if the mined-out area is greater than the depth below surface.
- It is assumed that the load is uniformly distributed over the cross-sectional area of the pillar

2.2 Type of Pillars

Based on the W/H ratio, coal pillars are divided into three categories:

- **Slender pillar**: pillars which have W/H ratio less than 3 or 4. When these pillars are loaded to their max. Capacity, they fail completely, shedding nearly their entire load.
- **Intermediate pillar**: pillars which have W/H ratio in the range of 4-8. These pillars neither shed their entire load when they fail nor can accept any more load.
- **Square pillar**: pillars which have W/H ratio greater than 10. These pillars can carry very large loads.

2.3 Subsidence

Mining subsidence is manifestation of the action of gravity on strata which have been rendered unstable by the withdrawal of their natural support over a sufficiently large area.in such cases all the strata from the seam to the surface sink slowly at different rates, until in the course of time with the development at opposing forces equilibrium is restored at the lower level.

2.4 Technical Parameters of Subsidence

a) Angle of Fracture/Crack

Underground coal extraction causes subsidence movement in the form of trough and cracks on the surface. The strata between surface and coal extraction have been subjected to intensive fracturing and dilation of discontinuities. In the panels, first time subsidence induced cracks appeared along with formation of trough on the surface and during the first main fall. The outer ring of the cracks formed on the surface during first main fall was almost circular in shape. The diameter of the outer ring of the cracks was about 40m and lies in Centre of the panel between 10m retreat to 50m retreats in the short wall panels. After further advance of the face and subsequent falls in the goaf the subsidence cracks reached above the barrier of the panel and front cracks reached up to 5-30m behind the face on the surface. It depends on major fall interval, surface blasting (position, time and blasting geometry) and rock mass characteristics. The cracks becoming widened and extended gradually up to 350mm in convex portion and were again closed down due to compressive forces in concave portion of the trough. It was experienced that for better strata management in the panel, the fronts cracks should not be lies far behind from the face and it should be lies between 5m to 20m in ideal condition. Generally cracks were found 5-6 m behind face line and so the angle of crack was (-) 05^0 56’ 48’’ in longitude direction. In transverse direction angle of fracture was observed between (-) 1^0 47’ and (+) 21^0 48’.

b) Slope of the Ground

Differential vertical subsidence between adjoining points along the surface of the subsidence trough will form a slope and induce tilting. The slope is expressed by the derivative of the vertical subsidence with respect to the horizontal. When the ground subside slopes (tilts) are induced. The maximum tilt occurs at point of inflection (where half of the maximum subsidence occurs) which is about 20-25 away from the barrier on transverse line and between panel edge and the Centre line. The maximum slopes of the ground surface were computed as 47mm/m to 109mm/m on longitudinal and transverse lines over the panels.

C) Strain of the Ground

Vertical displacement can be obtained by taking the first derivative of the slope or the second derivate of the vertical subsidence with respect to the horizontal. The value of vertical curvature can be expressed by the radius of curvature i.e. the reciprocal of the curvature. Where a curve is induced in the ground surface, it will either stretch or compress the surface depending on whether the curve is convex or concave. Near the edge of the profile, the shape is convex and a tensile strain develops, manifesting itself as an open crack or a zone of cracks. In the Centre of the trough, where the shape is concave a compressive strain develops. This may lead to small ridges being created. These are some time difficult to find being a few millimeters high. The maximum compressive strain of the ground was in the range of 29mm/m to 69 mm/m and the maximum tensile strain was in the range of 28mm/m to 66mm/m in the ground over the panels at the mine.

However, DGMS guidelines for maximum surface strain is 1.5 mm/m for important buildings and plants, 3 mm/m for main railways lines, road, rivers, water tanks and only for small buildings and branch railway lines they permit a strain of 6mm/m so it is statutory obligation to plan the mining activity accordingly.

d) Angle of Draw

The angle of draw is the angle between a vertical line from the edge of the workings and a line to the point at which the subsidence becomes negligible. The surface area laying outside the zone of draw will not be affected by the workings. Therefore the angle of draw defines the limit of subsidence and is also called the limit angle. The field observations indicate that measurable surface movement does not extend outside a zone bounded by a (+) 20^0 8’ limit angle in the mine. The average angle of draw for the areas containing all short wall panels is 10^0.

2.5 Theories of Subsidence

Some of the old and modern theories of subsidence have been briefly described here

2.5.1 Vertical and Normal Theories

According to dome theory it is believed that the rocks overlying an excavation are acted on by two forces cohesion and gravity only. If the gravity overcomes under cohesion, the roof will fall forming an enlarging arch. Subsidence will take place only if the dome breaks through to the surface. The axis of dome is vertical when seams are horizontal, but it lies between normal and vertical when seams are inclined.

2.5.2 Beam and Plane Theory

The immediate roof to be a cantilever beam and considered that the lowest part would be under compression and upper part under tension. The break in immediate roof occurs. The theory has been improved by considering the beam built in at both ends and loaded by its own weight.it is assumed that no bond exists between the layers and thus they are free to deflect independently.

2.5.3 Trough Theory

Distinguish between a main break and an after break. In flat seams the main break is vertical and the after break is in a direction bisecting the vertical and the angle of slide.in dipping seam the angle of draw increases.it is 35.8 degree from the vertical for a 40 degree dip and the main break occur over the seam at an angle from the vertical equal to the half the dip and the angle of draw and the angle of slide are the main factors used to determine the extent of subsidence. The angle of slide $\propto$ can be protected on the basis of coulomb's theory and is given by

$$\propto = 45 + \emptyset/2$$

Where, $\emptyset$ is the angle of internal friction of the material.

2.5.4 Continuum Theory

In continuum theory it is assumed that the ground acts as a continuous body bounded by the surface above and the excavation below. If the elastic moduli, the initial stress in the ground and the boundary conditions that is the distribution of stress on surface, on the roof or on the floor are given i9t is possible to predict stress and placements at any point of the medium by using the theory of elasticity (salamon 1994).

2.5.5 Particulate Theory

The rock medium for which stochastic equation determines movement has been called a stochastic medium. The concept upon which the theory of stochastic media is based is not restricted in application to loose media only.it can be used in explaining subsidence in continuous media also.

2.5.6 Dome Theory

The fracture extends vertically to the surface from the boundaries of the underground excavation. later this view evolved into normal theory according to which it was assumed that the strata subside normal to the seam.it was suggested that the subsidence; limits varied from vertical to the normal depending upon rock sequence and limiting lines bisect the angle between the vertical and normal lines when the dip is less than 45 degree and the dip exceed 45 degree the of fracture lies at an angle 45 minus half the angle of dip.

2.6 Subsidence Management

The subsidence induced cracks generated due to extraction of coal in underground panels should be filled in immediately particularly during rainy season. The width of the cracks is varying from few mm to 350mm. In order to prevent the entry of water in the subsidence area over the work-outs/workings of short wall workings, the cracks developed were filled in and compacted properly.

In shallow depth working, there may be a chance of pot-hole or abrupt subsidence particularly in low lying area where overlying strata including hard cover got lower competency or hardness due to long percolation of the surface water. Therefore, the low lying surface area over the panels were filled up to the level of the nearby ground and compacted properly to give the shape similar to the surrounding, to ameliorate the chance of inundation from surface water. In the property where the depth of working is very low, adequate and proper stiffness of supports in the advance gallery and a regular face advance should be maintained to prevent pot-hole and consequent strata control problem in the workings for safe and smooth operation of the short wall panel.

a) A garland drain around the panel over the barrier was made to drain out the accumulated surface water from the subsided or depressed area to keep it free from surface water. The cross section of garland drains was 1mx1m and smaller cross section drains were also made to connect the garland drain with the lowest level of the subsided area to drain out the accumulated. If required, the earthen dam of proper design was also built to arrest inflow of surface water in the subsided area.

b) Constant monitoring/supervision of surface area during the monsoon period and during heavy shower, one competent person with sufficient manpower were deployed round the clock to monitor the discharge at the culverts and at major points where there is chance of accumulation of water. The crack, which was opened during heavy shower, has to be filled in properly to prevent entry of water through it and take precautionary measure to prevent inundation of the mine workings from the surface water.

c) The blast holes, which were used for induced caving in the panels, were filled in properly so that there is no chance of entry of water in the mine.

d) Plantation for environmental management on the surface over the extracted short wall panels were done to upgrade the environmental condition, to stabilize the area and prevent soil erosion where manifestation of the underground extraction in the form of subsidence, subsidence induced cracks, slope, displacement and strain due to formation of subsidence trough were occurred. Some plantations are taken the shape the bushes and in recent goaved out areas sapling has been grown to increase the green cover. Plantation of local species considering requirement of fruits, timbers, and fodder has created an atmosphere for conservation of faunal species in the subsided area.

CHAPTER 3

CASE STUDY

3.1 Description of the SECL Mine

The mine is located 9 km north from the Bishrampur railway station and 30 km northwest from the district head quarter, Ambikapur (Surguja), Chhatisgarh. The topographical area consists of undulated (both cultivated and baron) land. The elevation in the area ranges from 536 m to 560 m above MSL. The drainage of the area is controlled by seasonal Gour bahra nullah flowing west east across the middle of the property. The nullah discharges in Pasang River flowing outside the mine leasehold area.

The enclosing latitude and longitude of geological block are-

Latitude from 230 12' 16" to 230 15' 16" (N)

Longitude from 82058' 58" to 830 02' 53" (E)

The area is free from major geological disturbances.

3.2 Maximum Observed Subsidence

The maximum possible surface subsidence at a point on the surface occurs after a certain minimum width of a seam has been extracted. This minimum width called critical width. The thickness of the excavated seam affects the magnitude of surface subsidence. The thicker the coal seam, the greater will be surface subsidence.

Figure no- (1a") Surface Subsidence

The maximum possible subsidence factor (A) for critical and supercritical widths depends on the rock characteristics of the overburden strata, the method of mining and the roof control. The subsidence factors define as;

$$A = \frac{M}{S\,max}$$

Where, A = Subsidence factor,

S_{max} = Maximum possible subsidence

M = Extracted seam height (m)

For the short wall caving method, properties of the overburden strata provide a major effect on the variation in A. When the overlying rock is comparatively soft and weak, A is a large, whereas when the overlying rock is hard and strong, A is small. Table 3 shows the subsidence factor observed in the worked-out panels.

Table (1): Maximum Subsidence Observed at the Short Wall Panels.

Name of the Short wall Panels	Widths of Panels (M)	Av. Depths of panels (M)	Maximum Subsidence (CM)	Subsidence factor
63 (1ST Panel)	84	45.5	95	0.41
58L (2ND Panel)	84	38	117	0.50
57L (3RD Panel)	84	35.5	127	0.55
34L (4TH Panel)	104	32	111	0.48
S-7 (5TH Panel)	104	32	158	0.63
S-6 (6TH Panel)	84	30	114	0.45
S-5 (7TH Panel)	84	30	148	**0.59**
46L (8TH Panel)	64	41	116.5	**0.47**

Given by the mine manager SECL

The subsidence factors for the panels are varying from **0.41 to 0.63** at the mine. The amount of maximum observed subsidence in short wall panel is generally slightly more than the maximum observed subsidence in the long wall panels at the mine and this may be due to the lesser depth of working in the short wall panels. The maximum observed subsidence in S-7 short wall panel was exceptionally high. Before commencement of extraction in this panel the seasonal nallah flowing over the last portion of the panel, which was suitably diverted outside and the overlying strata over the panel of his portion was soft and this may be the reason for such high subsidence in a particular zone.

3.3 Shape of the Subsidence Trough

When a subsidence trough is formed at the surface, the central part subsides vertically and the remainder moves inwards and downwards toward the Centre. Points located directly above the Centre of the goaf will subsides vertically and be displaced on an axis parallel to the goaf, where points away from the Centre will be displaced in an elliptical fashion. Since the Centre point is not moving horizontally, while the points on either side of it move inwards toward it, the central part of the subsidence trough will be subjected to lateral compression. In the Centre of the excavation, the vertical subsidence is maximum whereas the horizontal displacement is zero.

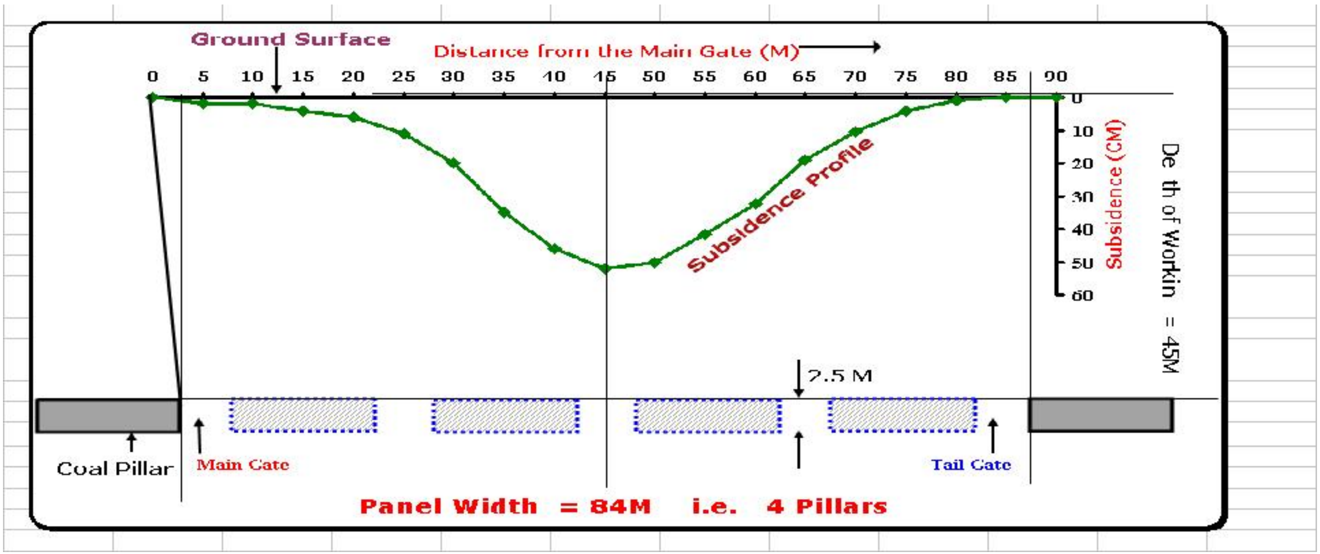


Fig. 1(a): Subsidence profile across the Panel, 84m panel width, 63L Short wall Panel

The rock is so variable in nature and its geometry that there was often a difference between profiles on the same mine. Fig. 1(a), Fig. 1(b) and Fig. 1(c) depicts the subsidence profiles over the panels in across and along the panel at the mine

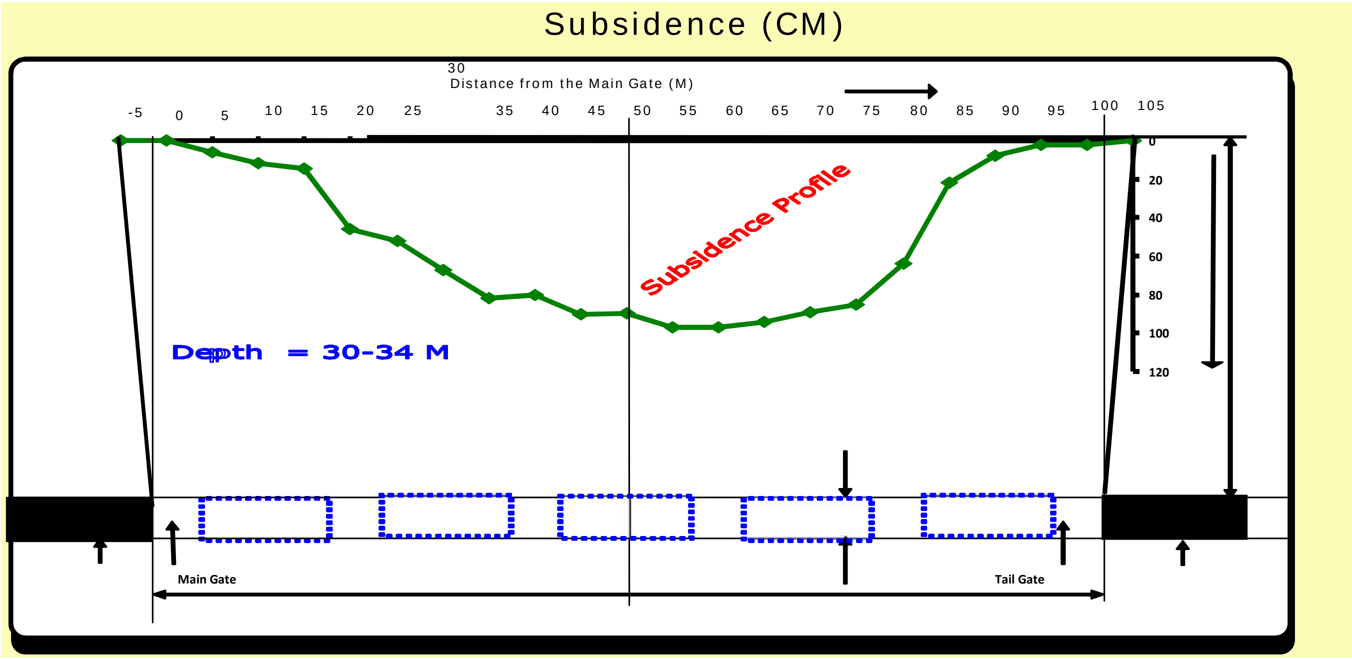


Fig. 1 (b): Subsidence Profile across the Panel, 104m panel width, 34L short wall Panel

The subsidence profile has a part which is convex and where the ground surface is getting elongated and another part is concave where the elongated ground surface is getting compressed. Thus, over the advancing face and the nearby goaf where ground tilt was occurring tensile strain develops. The maximum tensile strain develops where the angle of break reaches the ground surface from the edge of the solid face. Further on the goaf side the shape of the subsidence profile changes to concave and the particles on the ground now comes nearer to each other giving rise to compressive strain.

After the ground tilt has been fully recovered in an extraction of supercritical width the subsidence profile takes the shape as in the original ground and there is neither tensile nor compressive strain at the central part of the subsidence trough. It was also observed that due to induced blasting, slope of the subsidence profile increased i.e. full subsidence reached on surface quicker.

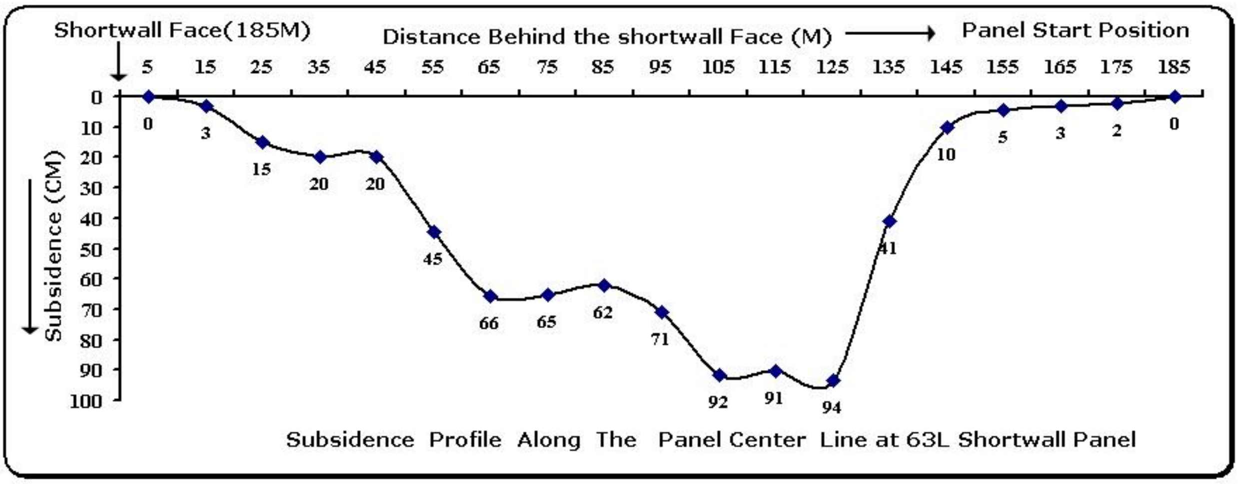


Fig. 1 (c): Subsidence profile along the panel Centre line at 63L Short wall Panel

Subsidence was not reaching on surface instantaneously. The major surface subsidence was achieving in 6-9 days i.e. after an advance of the face about 54 to 61m. The rate of subsidence at a particular point depends on the rate of face advance, location of points at the grids and depth of workings. Subsidence as a function of time and distance from face is illustrated in Fig. 1(d), Fig. 1(e) and Fig. 1(f).

It was found that the first settlement of the ground which constitutes about 1-9 % of total subsidence takes around 2-4 days' time or at an advance of about 12-24m and was occurred in steady stage. In the accelerated settlement, around 5-26% of the total subsidence occurred and it takes another 2-4 days i.e. a further about 10-25m advance of the face. Major subsidence occurred in the rapid settlement stage and it constitutes about 50-90% of the total subsidence, and it takes 1-3 days i.e. further advance of 3-15m. During final settlement periods negligible change in subsidence occurred. The magnitudes of the residual subsidence after the extraction of the panel was measured and it was found that at stations 57L/60C and 58L/310C was 4.08% and 6.45% of the total subsidence observed respectively and total stabilization of ground was taken place after elapsed of 12 months from completion of extraction of the panel.

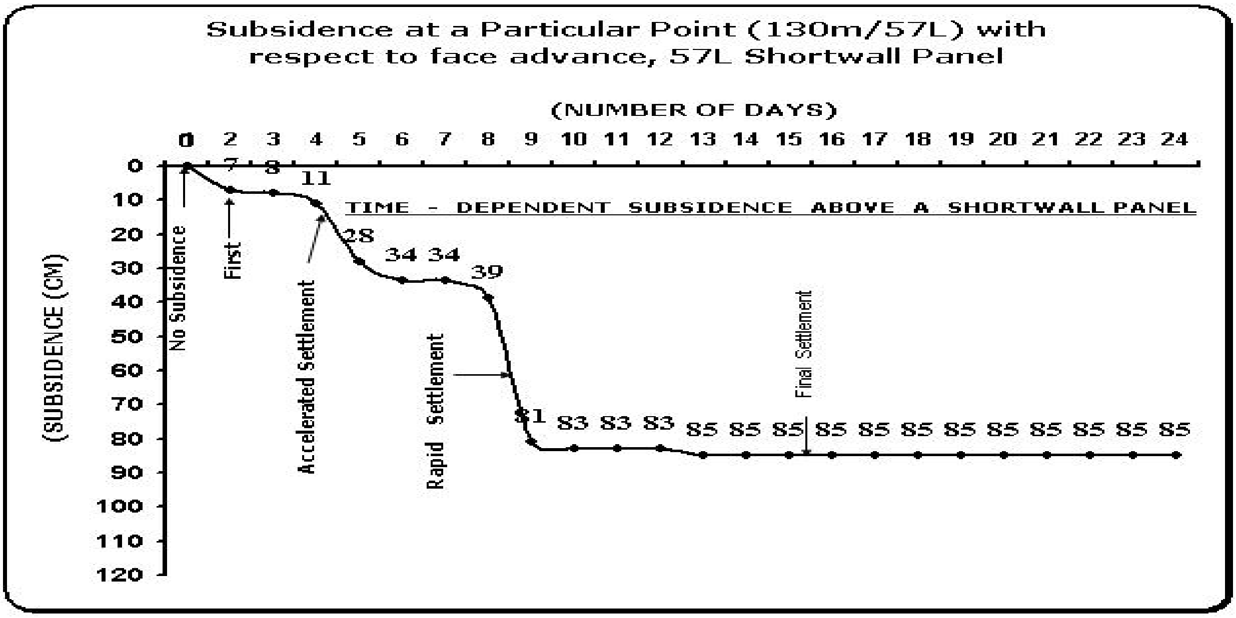


Fig. 1 (d): Subsidence Vs time curve of a particular point (30m/S-7) over the short wall panel.

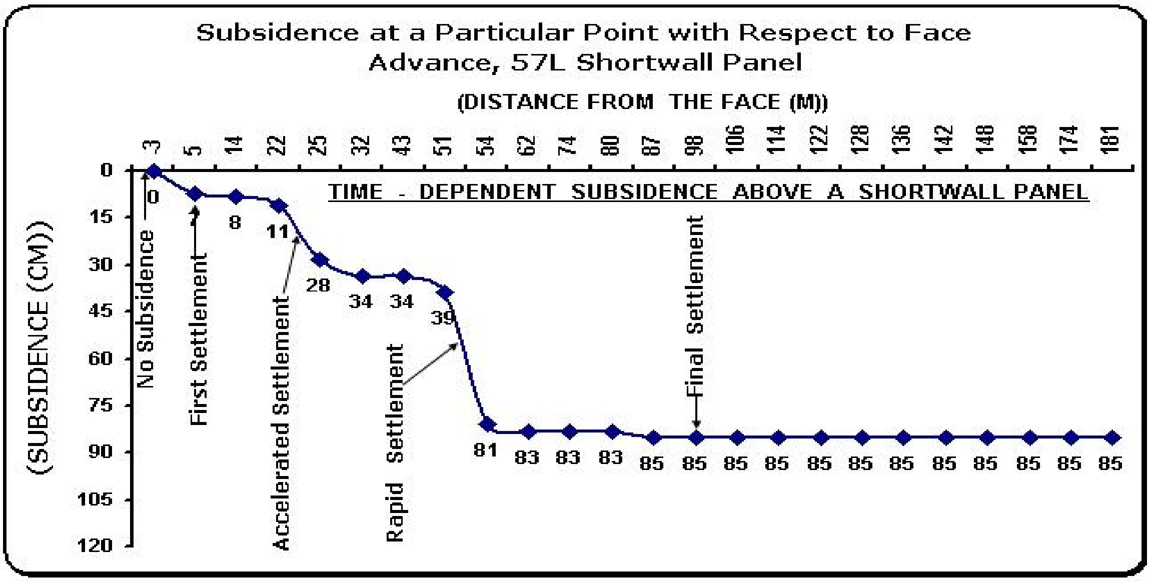


Fig. 1 (e): Subsidence Vs distance from the face curve of a particular point (130m/57L) over the short wall panel

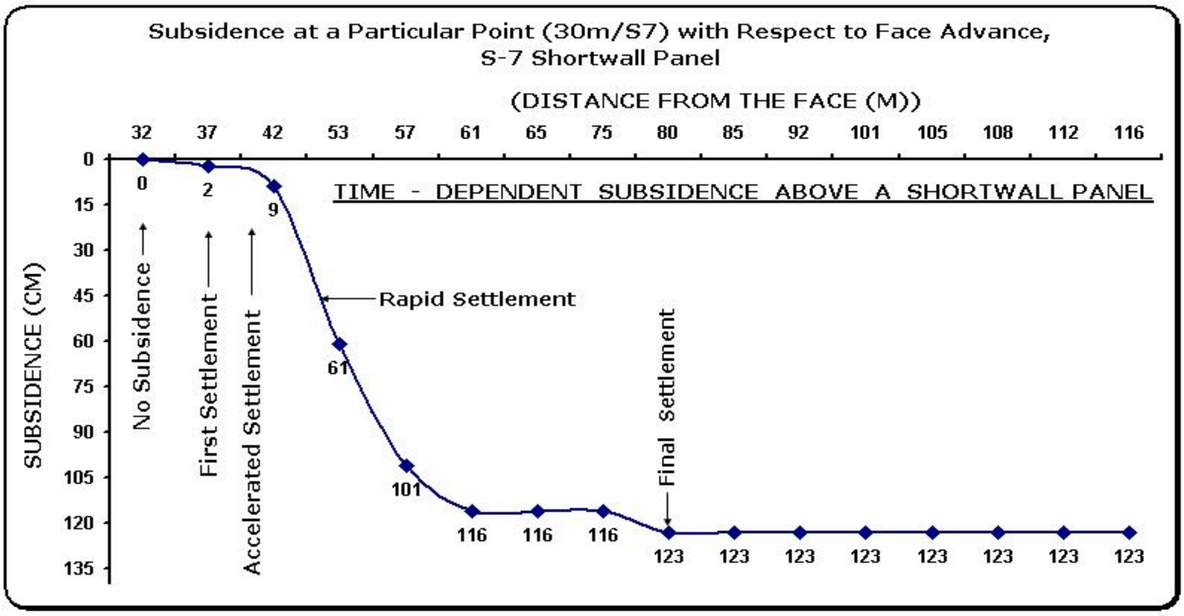


Fig. 1(f): Subsidence Vs distance from the face curve of a particular point (30m/S-7)

CHAPTER 4

GEOMINING PARAMETER

4.1 Geo-mining Details of the Study Site:

Profundity of hard cover of Pasang crease working of Balrampur 10 and 12 mine, SECL in the ascent side were under 15 m at numerous spots having four-path intersections with section measure 4.5 x 2.8m with columns 20m x 20m . Different geo mining parameters of the mine are as per the following:

Table (2): Geo Mining Parameters of Balrampur 10 and 12 mine

Thickness of the seam	1.8 to 2.7 m
Gallery size :	: 45×2.7 m
Height of the exhibition	2.4 to 2.7 m
Pillar size	20 m X 20 m
Depth of working	21 m to 44 m (Hard cover is under 15 m)
Nature of rooftop & floor	Shaley Sandstone
Nature of rooftop & floor	1 in 51
RMR	56.3
Modulus of elasticity	1.8 Gpa
Tensile strength	3.11 Mpa
Compressive quality of coal (M Pa)	27.9
Density of coal(kg/m3)	1400

Fig. 2 shows bore opening segment of BXI 15 of Balrampur, and CBBP-111,112 &113 of Kumda 7&8 Inclines, individually. Top Soil thickness shifted from 2 m to 5 m here while weathered mantle differed from 8 m to 12 m. hard cover in these areas was discovered to be around 6.2 to 15 m.

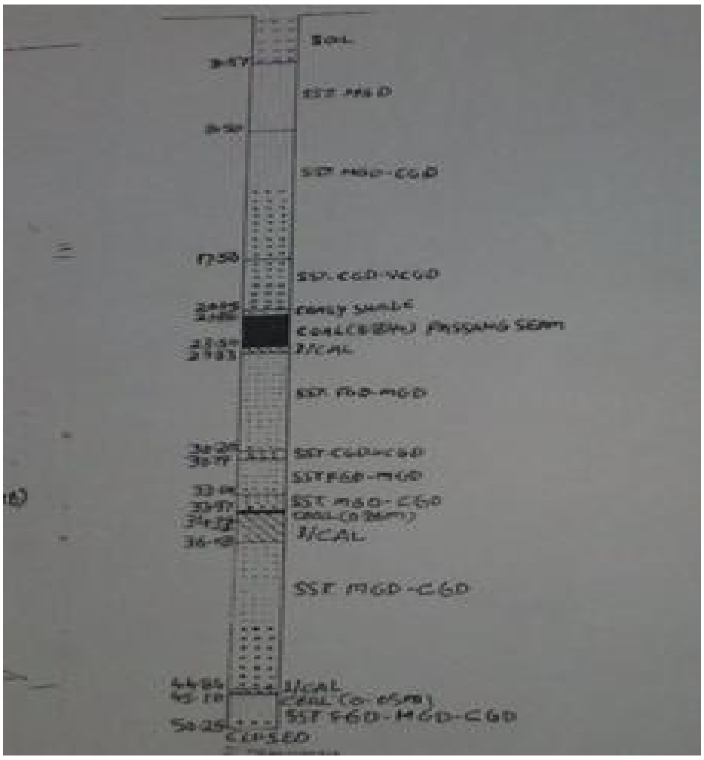


Fig. 2: Typical section of bore hole a Balrampur 10 and 12 mine, SE

Soil and sandstone section assumes a critical part on stacking design of the hidden strata. It for the most part grants dead load on the fundamental beds of the strata. Next fundamental section is sandstone whose lattice changes from coarse grain to fine grained and the other way around. Coarse grained sandstone is by and large of low RQD though fine grained sandstone is of high RQD and quality.

The prompt rooftop over the coal crease comprises of shally coal, carbonaceous shale, and dim shale having thickness of 21 cm to 41 cm. The significant bit of prompt rooftop comprises obviously grained to fine grained sand stone with a compressive quality of around 17.5 MPa. The compressive strength of coal seam is 27.9 MPa

The geo-mining parameters of the proposed study area of the Balrampur and Kumda mines are shown in Table 1. Field observations during February'15 and detailed measurement of width and height of workings and pillar dimensions through offset survey revealed that the gallery width in majority of the cases is within 4.8 m, except at the loading points with about

5.5 m gallery width and height of 2.9 m at an interval of about 2-3 pillars distance in the conveyor roadway. However, there was no spalling of pillar sides and roof disturbances indicating stability of the roadways and pillar in the present condition of no roof support in the developed areas. Travelling roadways were supported by roof bolting.

4.2 Physio-Mechanical Properties Of Over Laying Strata

Various boreholes had been drilled over concerned panels i.e. 18L of Balrampur 10 & 12 Inclines mine and 29L district of Kumda 7&8 inclines mine. The hard cover along with the depth and thickness of the seam are given in Table 2.

Table 3: Geo-mining Parameters of developed workings under 15 m depth cover

Height of gallery:	2.4 m- 2.7 m
Depth of concerned developed panels:	21 to 44 m
Hard cover	6.2 m to 15 m
Existing overlying/underlying workout areas	Nil
Existing mining pattern	Developed on bord & pillar workings
Pillar size:	20 m x 20 m (center to center)
Gallery width:	4 m - 4.8 m

The average hard cover in the concerned area is less than 15m and the depths of the seam were from 20m- 33m. Based on engineering judgment and giving a higher weightage to the borehole lithology, estimated RQD and the intact average compressive and tensile strengths of different beds tested in the laboratory are given in the Table 4.

Table 4: The borehole details lying over developed workings under 15 m depth cover at the Balrampur mine

Borehole no.	Depth of cover, (m)	Seam thickness, (m)	Hard cover, m
BIX-15 (18L district)	21.23	2.27	6.20
CBBP 113(29L)	44.00	1.10	15.00

4.3 Lithology of the Mine

Table 5: Representative lithology above the Pasang seam along with their intact properties

Bed No.	Run up wards, m	Rock types	Thick-ness, m	RQD %	Compressive Strength, Mpa	Tensile strength, MPa
		Coal	2.4	--	23.8	2.5
Bed-I	0.0-5.51	Medium grained sandstone, laminated with shale	5.51	40	10.96	1.5
Bed-II	5.51-12.17	Coarse grained to medium grained sand stone	6.66	78	17.1	1.4
Bed-III	12.17-16.02	Very coarse grained sand stone	3.85	43	13.92	1.4
Bed-IV	16.02-30.00	Medium grained sand stone	13.98	75	14.5	2.2
Bed-V	30.00-41.00	Weathered rock	11.00	--	--	--
Bed-VI	41.00-50.00	Sandy soil	9.00	--	--	--

The thickness and RQD of the six major overlying beds for these four boreholes are given in Table-5. Similar to the observations of the previous boreholes in this mine, it is observed that Bed-2 and Bed-4 are the massive beds with high RQD and are responsible for the weighting intensity at the faces.

As per the numerical modeling studies carried out earlier (CIMFR Report No. GC/MT/47/99-2000), for a face length of 84m with a depth of cover of 50m consisting of 30m hard cover and 20m weathered rock, the main fall was predicted to occur between 110 to 120m of the face advance. The actual observed main fall span was 120m.

The borehole BIX-117 is at the start of the 58L short wall panel where the hard cover was 25.17m and the depth of cover was 41.97m. As per CIMFR Report No. GC/MT/47/99-2000, for a face length of 84m, cover depth of 40m consisting of 20m of hard cover and 20m of weathered rock/alluvial soil, the main fall was predicted to occur between 100 m to 110m of face advance. The actual observed main fall span was 96m.

Thickness of hard cover and soil+ weathered rock decreases. From the observations of numerical models (Refer CIMFR Report No. GC/MT/47/99-2000), the main fall span increases with the increase in the hard cover thickness, decreases with the increase in the thickness of soil+ weathered rock Based on the above analysis it is observed that as the depth of the cover decreases the and also decreases with the decrease in the cover depth. Further, as the face span increases from 84 m to 104 m, the main fall span for depth of cover of 50m consisting of 30m hard cover and 20m weathered rock decreases from 110-120m to 90-100m which correspond to a decrease of 16% to 17%.

Overburden Dump of 8.2 to 12 m height along with Arshota nalah is available over N2 panel of Kumda 7 & 8 m under 37 m depth cover. From the surface contours it is evident that the developed workings are at a depth of 27 m under the nalah.

CHAPTER 5

EMPIRICAL MODEL

5.0 EMPIRICAL MODEL

Stability of the pillars was evaluated with the conditions of 4 locations; 18 L district, S1 panel of Balrampur mine, 29 Level districts and N2 panel at Kumda mine of the study sites. The safety factor of pillars is calculated as the ratio of the pillar strength and the average pillar stress. The strength of the pillar is calculated using CMRI pillar strength – empirical equation.

Uni-axial compressive strength tests were conducted and for estimation of pillar strength the compressive strength of 25.4 mm cube sample σ_c was found to be **27.9** MPa. Mean density of the overlying strata was estimated considering the variation in density of hard cover, soil, weathered mantle and also in a typical site the 10 m high overburden dump. Pillar strength, Stress on pillar and safety factor was estimated for three locations with depth cover of 21.23 m, 23.9 m, and 44 m considering the concerned borehole sections and the field observations:

Location 1: 18 L District:

Depth of cover H: 21.23m, Height of gallery w: 2.7m, Width of gallery h: 4.8m,

Strength of pillar S $= 0.27 \times 27.9 \times 2.7^{-0.36} + (\frac{21.23}{250} + 1) \times (\frac{4.8}{2.7} - 1)$

= 6.1093 MPa

Stress on pillar $= r \times z(1 + \frac{Wo}{Wp})^2$ where, r = unit weight of rock per meter,

z= depth of cover (m)

$=2.6 \times 10 \times 10^3 \times 21.13\ (1 + \frac{4.8}{15.2})^2$

= 0.955 MPa

Factor of Safety: $\frac{Strength}{Stress} = \frac{6.1093}{0.955} = \mathbf{6.397}$

Location 2: S1 Panel:

Depth of cover H: 23.9m, Height of gallery w: 2.7m, Width of gallery h: 4.8m

Strength of pillar S $= 0.27 \times 27.9 \times 2.7^{-0.36} + (\frac{23.9}{250}+1) \times (\frac{4.8}{2.7}-1)$

$= 6.116$ MPa

Stress on pillar $= 2.6 \times 10 \times 10^3 \times 23.9\ (1+\frac{4.8}{15.2})^2$

$= 1.076$ MPa

Factor of safety: Strength/stress

= 6.116/1.076 = **5.684**

Location 3: 29 L District: Depth of cover H: 44m, Height of gallery w: 2.7m, Width of gallery h: 4.8m

Strength of pillar S $= 0.27 \times 27.9 \times 2.7^{-0.36} + (\frac{44}{250}+1) \times (\frac{4.8}{2.7}-1)$

$= 6.179$ MPa

Stress on pillar: $= 2.6 \times 10 \times 10^3 \times 44\ (1 + \frac{4.8}{15.2})^2$

$= 1.981$ MPa

Factor of safety: Strength/stress

= 6.1795/1.981 = **3.1193**

Location 4: N2 panel: Depth of cover H: 44m, Height of gallery w: 2.7m, Width of gallery h: 4.8m

Strength of pillar S $= 0.27 \times 27.9 \text{x} 2.7^{-0.36} + (\frac{44}{250}+1) \times (\frac{4.8}{2.7}-1)$

$= 6.179$ MPa

Stress on pillar: $= 2.6 \times 10 \times 10^3 \times 44\ (1 + \frac{4.8}{15.2})^2$

$= 1.981$MPa

Factor of safety: Strength/stress = 6.1795/1.981= **3.1193**

Based on the above analysis, the factor of safety estimated through empirical models is in the range of 3.1 to 6.4 indicating long term stability.

CHAPTER 6

NUMERICAL MODELLING

6.1 Numerical Modeling Design

The elements in the panel considered are small. The elements are of size 1 m vertically and 1 m horizontally in the pillars. The dimensions of mesh elements increase geometrically from the inner model to the outer boundary. This is done because accurate reading over the seam is only required. Varying mesh size also reduces the simulation and computation time of model as the elements at the boundary were of greater dimension. The development model is then modified into an excavation model. Mohr Coulomb criteria and plain strain condition are used for simulation of the model. The sandstone element was used as the depth covers and the floor material. Table 6 shows Parameters used in the Numerical Modeling.

Table 6: Parameters used in the Numerical Modeling

Property	Coal	Sandstone	Clay Band
Bulk Modulus	3.67 GPa	6.67 GPa	2 GPa
Shear Modulus	2.2 GPa	4.0 GPa	1.4 GPa
Density	1480 kg/m^3	2100 kg/m^3	1650kg/m^3
Tensile Strength	1.86 MPa	9.0 MPa	6000 Pa
Cohesion	1.85 MPa	6.75 MPa	5000 Pa
Friction Angle	30^0	45^0	17^0

The top edge of the model is unconstrained and allowed to move in any direction. The side edges of the model are constrained to move in x direction and left free to move in y direction. The bottom edge of the model is constrained in moving in y direction that is vertically. The in-situ vertical and horizontal stresses were calculated:

Vertical stress = ρ x H

Horizontal stress = 3.75 + 0.015 H

Where, ρ = specific gravity of the rock mass cover and H = depth of cover.

The model is simulated to generate the in-situ stresses, before adding the mine openings or galleries to the model. Then the mine opening or galleries required are added to the model. After this the simulation is re-simulated to give the final stress distribution.

The 3 pillars have been modeled using FLAC5.0 with 4 galleries. The galleries are simulated to plot their vertical displacement and vertical stress contours over pillars. It shows different stages of a depillaring process. The different stages include division of pillars, splitting, and extraction of the stooks so formed leaving just ribs in the goaf. The sequence of Numerical modeling includes the following stages Fig 3 illustrates Grid Generation for Galleries and pillars are developed in the seams.

In the present numerical model studies, geo mining conditions of Kumda 7&8 inclines and Balrampur 10 & 12 inclines were simulated to understand the stability of pillars for hard cover varying from 12m to 15 m.

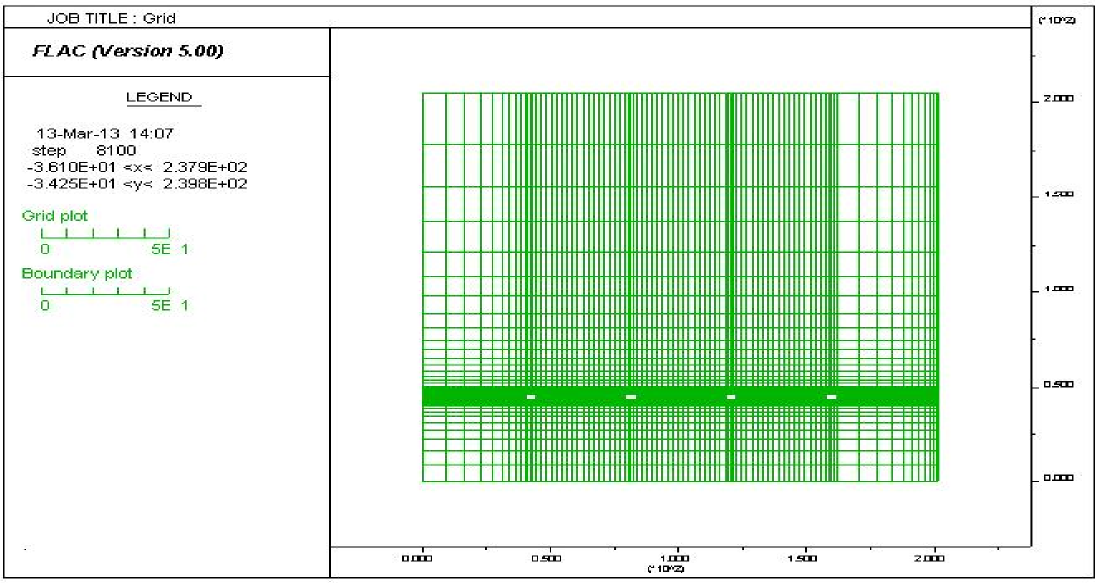


Fig 3: Grid Generation for Development of Galleries and Pillars in the Seams

6.2 Stress Distribution over Pillars

Cumulative stress over the pillars and stooks for FLAC simulation of numerical modeling for different stages is shown. The model was simulated with roof support, roof and side support and without support to comprehend the stress distribution over the pillars and stooks.

Maximum stress of 9 MPa is experienced by the stook present next to the fourth gallery after excavation of 5 stooks. The maximum over the pillar remains more or less same for supported and unsupported roof because the rock load remains constant. But the stress distribution profile changes showing more stress enforcement at the side of the pillars for supported roof and sides. Fig. 4 illustrates Stress Distribution over Pillar and stooks at different stages of depillaring.

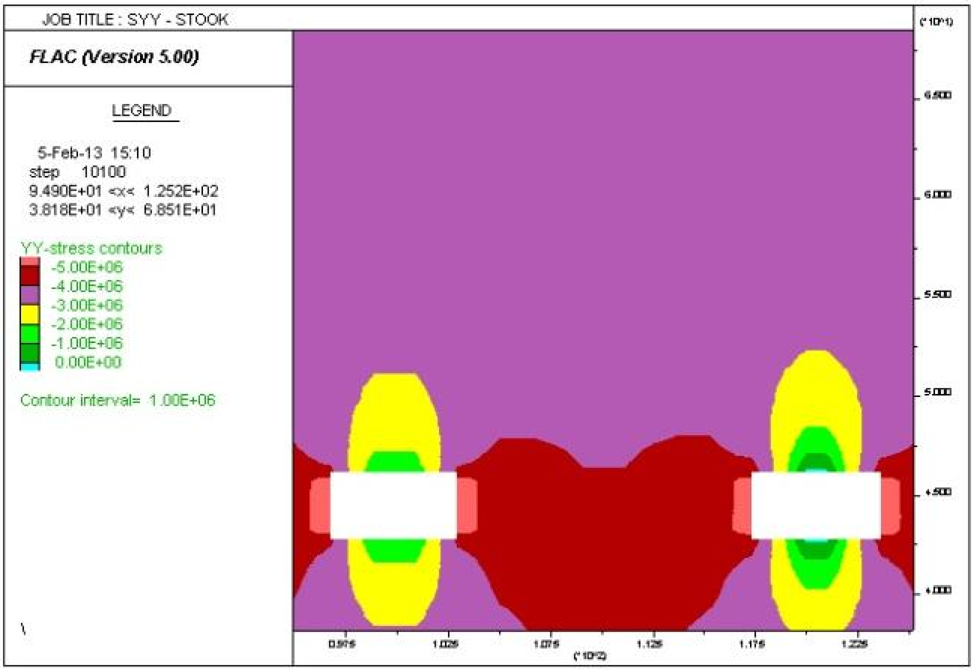


Fig. 4: Stress distribution over the fourth gallery

Stress distribution over the fourth gallery is shown in the Fig 16. The X-axis represents the stress in MPa and Y-axis represents the goaf edge distance in meters. The maximum stress distribution over the pillar/stook shows increasing trend because of load on the pillar/stook due to extraction of adjoining stooks. The maximum stress observed from modeling was 9

MPa. Critical spans for first breakage of different beds are computed on the basis of the empirical-cum-statistical norms developed for long wall. Maximum span for first breakage of the beds without presence of any ribs in the goaf has been found to be about 49 m. Roof caving span was also ascertained for different depillaring panels in Indian geo mining conditions on the basis of Q system of rock mass classification through the parameters: α= factor depending on rectangularity b/a of the plate, a= smaller plate dimension, and b = larger plate dimension. When b/a <3, the equivalent face advance a_{eq} can be found as $a_{eq} = \alpha$ a for an infinitely long face when fall would occur.

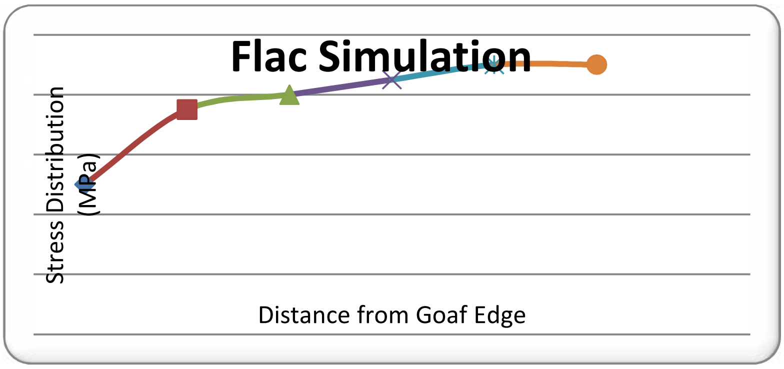


Fig. 5: Stress Distribution over Pillar/Stook

In the light of the plate theory and considering the rectangularity of the depillaring panel, the corresponding span for first breakage of the roof is ascertained to be 58 m. therefore, considering the above two approaches, first major fall without presence of any ribs can be anticipated at 50 to 60 m clear span in the goaf. Numerical analysis was also conducted to assess the influence of ribs which are generally left in the goaf. To study the influence of maximum possible size of ribs in the proposed extraction on goaf settlement, stability of 2.0 m wide ribs was analyzed under the condition of 45 m and 60 m span of the roof in the goaf.

Maximum vertical stress of about 8 to 9 MPa (Table 4.1) was indicated in the numerical model at the side of the gallery located in the middle of the panel, while the roof is under tension. Such situation may lead to tensile failure of the roof and need proper support to reinforce the roof and spalling of sides in the middle of the panel. After exposure of about 40 m clear span in the goaf, the 2.0 m wide rib is standing at safety factor of about 0.8 which may not fail instantaneously. Whereas, after exposure of about 60 m clear span in the roof, safety factor of the rib may be reduced to 0.5, which would crush without causing any adverse conditions to the advance workings.

CHAPTER 7

ANAYLISIS OF RESULTS

RESULTS

Subsidence movement, stress, strength and safety factor investigations conducted over short wall led to the following results:

1. The subsidence monitoring in shallow depth workings where the effect of subsidence is more severe and quickly felt was essential to estimate the overhang in goaf behind the face and consequent transference of strata pressure or abutment pressure to the face and to the immediate advanced gallery of the short wall panels.
2. The subsidence was largely depends on the geometry and configurations of the panel, nature and matrix of the overlying strata, geo-technical properties of the strata and induced caving from surface.
3. The width-depth ratios were 1.84 to 3.25 and new were from 1.84 to 2.4 in the extracted short wall panels.
4. The maximum observed subsidence was 1.58m and the subsidence factor was 0.42 to 0.63 for all the short wall panel.

   Subsidence was not reaching surface instantaneously. The subsidence was occurred in stages at particular point. The major surface depression was achieved in about 6-9 days. The magnitude of the residual subsidence after one year from the completion of extraction in the panels varies from 4.08% to 6.45% of the total subsidence.
5. The maximum slope, maximum compressive strain and maximum tensile strain were 109mm/m, 69mm/m and 67mm/m respectively on the ground.
6. The maximum and minimum safety factor was 5.68 and 3.11 respectively.
7. The width of the subsidence induced cracks ranged from few mm to 350mm.
8. The maximum stress calculate by the Flac-5.0 software was 5MPa.

CHAPTER 8

CONCLUSION

CONCLUSIONS

Based on laboratory testing of coal samples, empirical and numerical modelling studies related to the stability of the pillar and analysis of observation data on subsidence profiles for underground coal mine, the following conclusions are drawn:

1) Field observations at Kumda underground mine indicated stability of the pillars without any perceptible side spilling and crushing.

2) Uniaxial compressive strength, triaxial testing of coal samples indicated, UCS, cohesion(C) and internal friction angle(∅) of 27.9MPa, 1.85 MPa and 30^0 ,respectively.

3) Empirical and numerical modelling using two dimensional continuum analysis indicated the maximum stress of 5 MPa over the pillar and safety factor exceeding 2 confirming to the qualitative observation of stability of the pillar.

4) Maximum subsidence of 1.58 m was observed over the extracted panels with a subsidence factor of 0.63 for the depillaring panels with the width and depth of 64 m-14 m, 30 m-45.5 m respectively.

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