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Strata Control for Mechanized Stopping Operation in Narrow and Weak Manganese Deposit of MOIL – A Case Study of Balaghat Mine

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Abstract

MOIL is producing annually around 1.13 million tones of manganese ore. It operates 7 underground and 3 opencast mines in the state of Madhya Pradesh and Maharashtra. The underground mines are operated at shallow to moderate depths. Asia's largest and oldest underground manganese mine is being operated by MOIL at Balaghat, Madhya Pradesh. Horizontal cut and fill mining with passive timber square set support has been replaced with advanced technique of pre-mining support by Cable bolting for reinforcement underground structures. The change in technology has enabled to transform the manual post filling of rock to hydraulic sand stowing operation in all the underground mines of the MOIL. With the available underground stopes, the panel working has been started with phased mechanization. The Side discharge loaders (SDL) has been introduced for mechanical handling of ROM in the stope and Load, haul and dump (LHD) machines for drivage development. These rapid mining operations have imposed the need for change in level interval for higher productivity. Presently below 12th level, level interval has been changed from exiting 30m to 45m at Balaghat Mine. Moreover, Single boomer electro hydrostatic crawler mounted drill jumbo has been introduced for drivage development at Balaghat Mine. The paper describes the phased rock mechanics investigations and its application in the underground mining for mechanized stoping operations at Balaghat mine of MOIL for better safety and productivity.

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Keywords: Manganese Ore, Rock Reinforcement, Rock Mass Classification, Stope Design

1. Introduction

Manganese ores in India are being exploited well over the past hundred years. These ore are mainly of secondary origin and are associated with the older Achaean meta-sedimentary [2]. The deposits in India were originally classified as three fold (Fermor, 1909) which were subsequently modified as fourfold (GSI News 1973; Khrishnaswamy 1979).

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They are as mentioned here under:

a. Syngenetic Gonditic deposits associated with highly metamorphosed Sauser series of rocks as in Central and Western India,
b. Syngenetic reef deposits associated with the Khondalite sequences of eastern ghats,
c. Replacement deposits in the Banded Iron Formations as in Singbhum, Karnataka and Goa regions;
d. Lateritic deposits and supergene enrichments associated with all the above three.

The manganese deposits in central India are unique in respect of their formation [9]. The ore body is basically sedimentary deposit subject to various types of geological disturbances. This area has the oldest meta-sedimentary named as Saucer series covering all the mines of MOIL from Ukwa mine to Gumgaon. The exploitation of ore bodies with weak host rock masses as found in the manganese deposit in central India is a difficult and challenging task. All the underground mines of MOIL were using horizontal cut and fill method of mining or it’s variant with various support systems like pack pillar and timber square set with post manual filling of rocks. This system of supporting could not reinforce the strata before mining. Roof support is essential to the safety of every underground mine. It has three primary functions:

a. To prevent major collapses of the mine roof;
b. To prevent miners from small rock falls that can occur from immediate roof skin; and
c. To control deformations so that mine openings remain serviceable for both ventilation as well as access to escape.

For rock reinforcement in underground structures, an engineering analysis begins with evaluation of two fundamental factors [10, 4];

- The strength of the different components of the rock structures and
- The forces that are loading it.

2. Geo-technical investigations

To keep pace with the changing technology in fast growing Indian mining industry, MOIL has undertaken rock mechanics investigation of the various supporting systems and stope design. Due to many constrains of underground mining and limited available space and time for supporting, many accidents in stoping area had taken place due to fall of roof. During the study of timber square set support it was observed that support system provided only passive support to immediate roof and after erecting the square set; it provided support to the immediate 1m top layer of the stope back.

2.1. Pressure on square set in stope

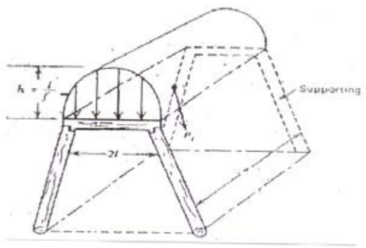


Fig.1. Load on wooden gallery set according to Protodyakonov

The cross section of a stope with timber support is shown in the Fig.1. According to many investigators [5], the pressure on a gallery is in the form of parabolic dome shown in the Fig.1. As the theoretical formula is complicated for practical purpose the approximate values have been taken. The Protodyakonov formula as shown in the Fig.1 is as follows:

$h = l / f$

h = height of the parabola load height comes to (0.4 to 0.53 m)

l = length of the cap on the wooden set (1.6 m)

f = Protodyakonov coefficient of hardness (weak schist 3) or 0.01 of the compressive strength of the rock (about 4)

From the above illustration, it is found that 0.40 to 0.53m top layer has to be supported additionally after erection of timber sq. set support. To reinforce this unstable zone of 0.40 – 0.53m, (for safety purpose considered the zone of 1m), the company made an attempt to change over to new system of support involving pre mining reinforcement of stope back and wall rocks with cable bolting and rock bolting. Before start of Cable Bolting on experimental basis, rock mechanics investigations for support design have been carried out at Balaghat Mine.

At Balaghat Mine, the ore body consisting of manganese oxides and manganiferous quartzite is associated with moderately metamorphosed phyllitic and schistose rock masses. The hang wall rocks are phyllites, phyllitic schist and sericitic schist. The foot wall rocks constitute feldspathic quartzite and quartz schist. The general strike direction of the formation is N25^o E and the amount of dip varies from 50^o and 60^o due West. In the northern section, the dip becomes flatter and in some places the direction has been reversed due to structural complications. The ore body varying in thickness from 4m to 20m is highly jointed with different joint sets developed perhaps due to local structural disturbance. Nevertheless, the trend of the major joint set is parallel to the ore body throughout the mine.

2.2. Rock mass classification

Before implementation of rock reinforcement by full column cement grouted cable bolt, rock mass classification has been carried out in consultation with Central Institute of Mining & Fuel Research (CIMFR) and National Institute of Rock Mechanics (NIRM), [8]. Obtained Q values and RMR with un-supported span of Balaghat Mine is appended below in Table 1 and 2:

Table 1. Q and RMR of ore body of Balaghat Mine

Width of Ore body (m)	RQD	Jn	Jr	Ja	Jw	Q	RMR
4	10	9	1	4	1	0.14	32
	10	12	1	4	0.66	0.07	26
8-14	10	6	1	2	0.66	0.25	39
	10	6	1	1	1	0.83	49
15-20	10	6	1	3	1	0.27	39
	10	12	1	4	1	0.10	30

Table 2. Un-supported Span and Support Pressure – Balaghat Mine

Mean RMR	Max. Span (unsupported) - m	Support Pressure kg/cm²
29	2.73	2.25
	2.07	2.83
44	3.58	1.79
	5.56	1.24
35	3.58	1.79
	2.43	2.47

2.3. Pre reinforcement

The concept of pre reinforcement consists of installing support prior to excavating the adjacent rock. The principal effect is to limit the displacement of the rock mass to small values. This minimizes the shear and dilation along existing geological structures and perseveres their in-situ cohesion and friction such that the rock mass becomes self-supporting (Little John, 1992). Another advantage of pre-reinforcement techniques is that blasting can be done against and already reinforced surface which would reduce the amount of blast damage. Additionally, the effect of sudden changes in the surrounding stress field resulting from mining can be better controlled by a pre reinforced rock mass.

The effective pre reinforcement of the rock mass in excavations such as those for cut and fill mining, longer reinforcement elements are required [10]. Since it is difficult to install long rigid rock bolts, the technique of cable bolting using long cables has been developed. With this scenario, the company has experimented and established the 'Cable Bolting Techniques' for obtaining improved ground conditions.

2.4. Design of cable bolting

Design of cable bolting depends on the mass of rock to be supported and it is essential to identify the probable mode of failure. Then the weight of rock blocks likely to be separated from the competent formations can be estimated by any of the various conventional methods – beam theory for stratified rocks and pressure arch theory for massive formations.

2.4.1. Empirical design

Numerous methods for the design of roof bolts have been proposed over the years. The oldest simplest and probably still most widely used equation for bolt design is dead-weight suspension (Obvert and Duvall 1967):

$$P = \frac{U * t * W_e * R}{n + 1} SF$$

Where, P = required bolt capacity;
U = unit weight of the rock;
t = thickness of suspended rock;
n = number of bolts per row;
We = entry width;
R = row spacing;
SF = safety factor.

2.4.2. Rock load height (Capacity/Pattern):

The rock load height concept is a slightly more sophisticated version of the dead weight theory. Originally proposed by Terzaghi (1946), the theory predicts the load on the support based on the rock quality and the span. Unal (1984) defined the rock load height taking RMR indicated by Benawaski. The rock load height concept (after Unal, 1984) is shown in Fig.2.

$$Ht = B\frac{(100 - RMR)}{100}$$

where, B is a width of opening/ Roof span

The guidelines for cable selection were based on empirical methods provided by Barton, et al (1977) [3]. For large openings, the recommended cable length, L’, in m is:

L’ = 0.4 B/ ESR for roof and
L’ = 0.35 H/ESR for walls
Here, B = opening width, H = height of the opening (stope)
ESR = Excavation support ratio, ESR values are 1.6 for permanent opening and 3 to 5 for temporary openings.

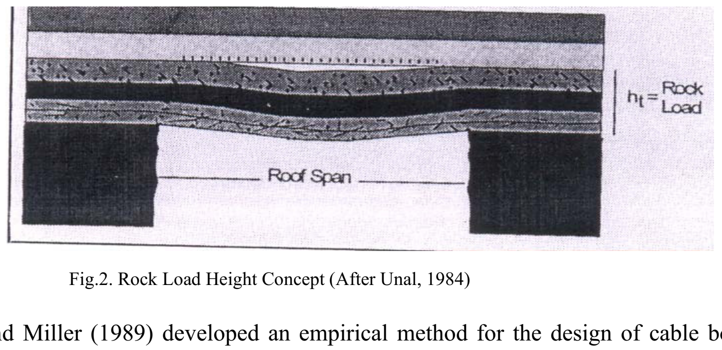
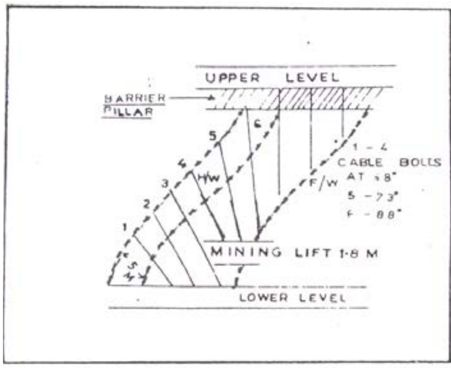


Fig.2. Rock Load Height Concept (After Unal, 1984)

Potvin, Hudyma and Miller (1989) developed an empirical method for the design of cable bolts. The rock mass conditions are first assessed using NGI rock mass approach (Barton and others, 1974) and adjustments are made to the Q values to obtain a modified stability number, N’ (Potvin and Miller, 1992) [11].

3. Cable Bolting in MOIL Mines

Roof deterioration was observed earlier in the square set stopes up to 1m height. Rock blocks of about 0.4 to 1m (for additional safety measures max 1.5m is considered) thick used to get separated from the stope block and fall down. Therefore, considering a probable weak zone height of 1.5 m and the density of the ore body a maximum of 4 T/cu.m., the load exerted by the loose strata is 1.5 x 4 = 6 T/sq. m.



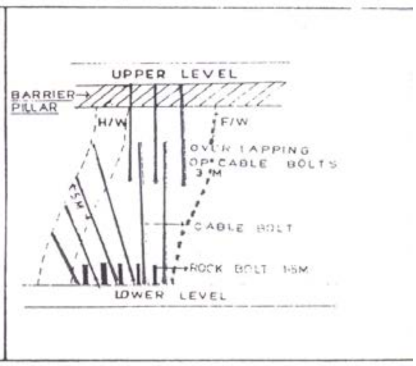


Fig.3. Fixing of Cable bolts in various angles (dip is 50°-65°) Fig. 4. Over lapping of Cable bolts by 3 m (dip is more than 70°)

Since the cable bolt capacity is 25 T, a spacing of 2m x 2m would give a support load of 25/4 = 6.25 T/sq.m. For additional safety, a rock bolt in between the cable bolt is also provided that gives an additional support loads of 10 T/4 sq.m. = 2.5 T/sq.m. Using the design charts of Potvin et al. also the support density comes to 0.25 cables per sq.m or 1 cable per sq. m. (that is a spacing of 2 m x 2 m); and the cable length should be 12-14 m. However, the cable length is dependent more on the ground requirements and is selected in such a way that the cable is anchored for about 5 m in the sound portion of hangwall rock. Accordingly cable bolts of length varying from 6m to 12m are used for support of the stope back in the cut and fill stopes of MOIL's underground mines. Illustration is given below in Fig. 3 and 4 for fixing of cables in various angles in the ore body. Seven wires, 16mm diameter pre stressed un-galvanized steel wire rope are fully grouted in 57 mm holes. To increase the compressive strength of the cement grout super plasticizer 'Sikament FF' is used along with grout @ 200 ml per bag of 50 kg cement. Second set of cables are installed in the stope back with a minimum overlap of 3m.

3.1. Monitoring of cable bolting

The monitoring work of the efficacy of cable bolt supports has been carried out with Multipoint borehole extensometers (MPBXs), Tape Extensometers and Load Cells. 5 MPBXs were installed to monitor the deformation of the cable bolted hang wall and the overlying ore body, in addition to MPBX, 8 load cells was also installed under the chocks in the cable bolted section to monitor the loosening load of the overlying ore. The installation details of the MPBX and load cells are given below in Table 3 & 4.

Table 3. Details of multipoint borehole extensometers

Sr. No.	MPBX No.	Anchor No.	Depth (m)
1.	MP 1	1,2,3 & 4	16.8,12.0,8.0 & 4.0
2.	MP 2	1,2,3 & 4	11.5,9.0,6.0 & 3.0
3.	MP 3	1,2,3,4 & 5	18.0,14.0,10.0,6.0 & 2.0
4.	MP 4	1,2,3 & 4	12.0,10.0,6.0 & 4.0
5.	MP 5	1,2,3 & 4	10.0,6.0,4.0 & 2.0

3.1.1. Observations of MPBX

a. The effect of the mining operations is likely to be noticed by MP1 and subsequently by MP2.
b. The relative displacement of anchors MP1 & MP2 reaches max. of 0.8 mm. It indicates that the cable bolted overlying ore and hangwall is stable and that the cable bolts are effective.

c. In MP3 shows that the max. Displacement is 0.53 mm. This also confirms that the support is adequate.
d. The relative displacements of anchors of MP4 & MP5 are less than 0.1 mm which is negligible.

From the above analysis it is confirmed that the cable bolt are adequate.

3.1.2. Observations of load cells

Mechanical Load Cells of 10 T capacity was installed under the timber chocks in the cable bolted stopes. Each load cell experienced the load varying between 0.4 to 1 T. The spacing of the chocks being 3m center to center, the support pressure works out in the range of 0.03 to 0.044 kg/cm². This pressure is negligible and it is confirmed that the immediate roof is not affected by mining and cable bolts are adequate.

Table 4. Details of load cell installed in cable bolted station

S.No.	Load Cell	Location	Load T
1.	L1,L2, L3 & L4	6th level;	2.96 T
2	L5	6th level	4 T

The following conclusions have been resulted from this experimental study. Principally,

a. The loosening of the ore is prevented by cable bolts and stoping with advance support by cable bolts has been effective and safe in the experimental panel
b. The technique aims at suitable reinforcement of weaker rock mass to withstand induced stresses during heavy blasting and allowing faster mining cycle resulting in quantum jump in productivity and safer mining.

3.2. Advantages of cable bolting

There are many advantages of cable bolting in cut and fill method of mining in poor to fair rock mass of hard rock mining. The two main advantages for strata reinforcement are; they can be installed in opening with low headroom and accommodate relatively larger amounts of strain without failure. They provide the advantages of pre placed support under difficult ground conditions, of secondary support to supplement rock bolting and also of permanent support [1, 6]. The cable bolts provide reinforcement of the rock mass by building a considerably thick beam and also by tying up the potential rock blocks to a higher stable horizon. Moreover, the cable bolts can be placed at any angle [7]. The observed improvements by Cable Bolting at Balaghat Mine is given in Table 5.

Table 5. Improvement in productivity by cable bolting at Balaghat Mine

Sr. No.	Description	Old Supporting method	Cable bolted stopes
1.	Face – OMS in T	1.19	2.94 – 3.5
2.	ROM from the stope/year	5327 T	10082 T
3.	Life of stope	6 – 7 years	3 – 3.5 years

Other advantage of this system is that it creates more head room for mechanization in stopes as well in drives also. Furthermore with the satisfactory outcome in safety standards of underground mines, the company has introduced panel working of stopes. The high value timber used for supporting has been eliminated.

4. Introduction of Mechanized Stope Design

With success of ‘Cable Bolting’ cycle, it was then decided to increase the stope length from 30m to 60m for panel working of 3 stopes, illustrated below in Fig.5, along with mechanical handling of ROM in stopes by Side Discharge Loader.

- A stope block of 60 m length with insitu rib and post pillars
- A panel of 3 such stopes for 180 m strike length
- Sequence, 1 stope for SDL for mechanical handling of ROM in stope with production drilling, blasting and ROM handling
- 2 stopes for preparatory operations like Hydraulic sand stowing and supporting – cable bolting

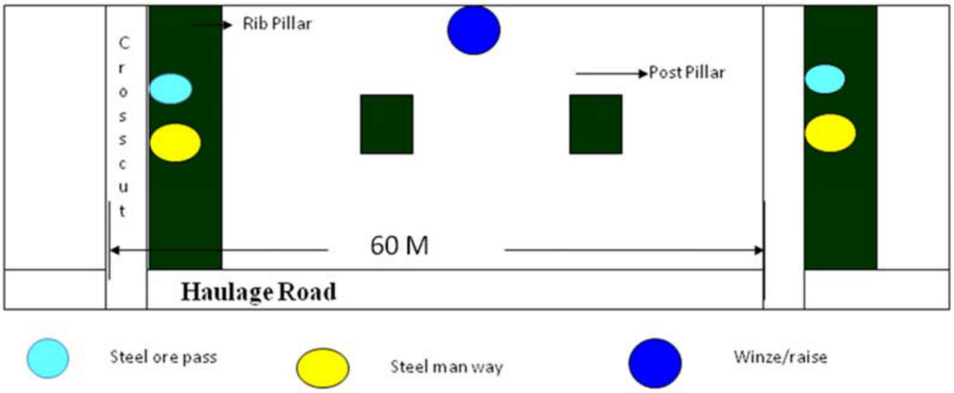


Fig.5. Conceptual Plan of 60m Stope at Balaghat Mine

4.1. Optimization of the size of Rib Pillar and Post Pillars

The overall Rock mass strength (RMS) of the Balaghat mine is 34.4 MPa and hence the post pillars of 5m x 5m size placed at 15m center to center along the strike and about 10-12m across that assures FOS 0.8. Similarly, numerical modeling has also been carried out.

After confirmation regarding the safety standards of these panel workings electrically operated Side Discharge loaders has been introduced in the stope for mechanical handling of ROM. The ROM is then loaded through the ore passes in 5T mine car. In this system, exposure of the employees under freshly exposed roof is minimized and improved through the productivity from the stopes. Improvement in productivity at Balaghat Mine by introduction of SDL for mechanical handling of ROM in stope is given below in Table 6.

Table 6. Productivity improvement by Mechanical Handling of ROM in the stopes

Sr. No.	Description	Without SDL	With SDL
1.	Face – OMS in T	2.94-3.5	6.90 – 9.0
2.	ROM from the stope / year	10082 T	23662 T
3.	Life of stope	3 – 3.5 years	2.5 – 3 years
4.	Stope length	30 – 60 meters	60 meters
5.	Panel length	Max. 60 meters	180 m – Panel of 3 stopes

4.2. Support design for haulage road and cross cut

After confirmation on 'Cable Bolting Techniques' for stope back reinforcement, numerical modeling for roadway support design has been carried out for the deeper levels at Balaghat Mine. The geotechnical assessment of rock mass was made through physico-mechanical properties and Bieniawski's rock mass rating (RMR) [7]. The RMR values obtained were in the ranges of 37-56. Details are given below in Table 7:

Table 7. Summarized geotechnical properties and RMR

Compressive Strength (MPa)	Tensile Strength (MPa)	Unit Weight (t/m^3)	RMR	Poisson's Ratio (v)
39.5-52.8	6.7-14.0	2.7-3.75	37-56	0.25

The support requirements were evaluated considering a safety factor of 1.5. For this purpose, load bearing capacity of 8 T was assumed for fully grouted rock bolts of 1.5 m length. The rock mechanics study was carried out in respect of support design for haulage roads and cross cuts in ore and wall rocks. The support design for roadways found out after numerical modeling is given in Table 8.

Table 8. Support design for roadway of Balaghat Mine

Name of drivage	Roadway width	Rock load	No. of bolts across	Spacing between bolts

	(m)	(t/m2)	the drivage	along drivage (m)
Haulage Road	3.6	3.40	2	1.0
Cross cut	2.4	2.16	1	1.2

4.3. Increase level interval from 30m to 45m

Present working is confined to 13.5th level at Balaghat Mine. L-section showing the operating levels at the Balaghat mine is placed in Fig.6. The reserves persistency at Balaghat mine is proved up to 23L (650 mtrs) below the surface and stoping rate has increased with introduction of SDL for mechanical handling of ROM in the stopes of 60 m length with panel of 3 stopes for mining, supporting and filling. It was decided to increase the level interval from existing 30 m to 45 m below 12th Level. The numerical modeling and in-situ stress estimation has been carried out for better safety.

It was found that ore body width is varying from 9 m to 20 m and it is only 3 m wide at southern side of 11th Level. Hangwall contact is of phylite and its thickness varies from 0.2 to 10 m. It is thicker at northern zone of the ore body. In the stope, an opening of 3 m is maintained between stope back and fill material (sand is being hydraulic filled in the stope). Cable bolt of 12 m length at grid of 2mx 2m is used to stabilize the roof back along with rock bolts of 1.5 m length at grid of 2m x 2m (in the center of 4 cable bolts) is used to stabilize the immediate roof. Stope distance from the haulage roadway is generally 30 m. present stope height is 30 m with a crown of 5 m height above the 12th Level.

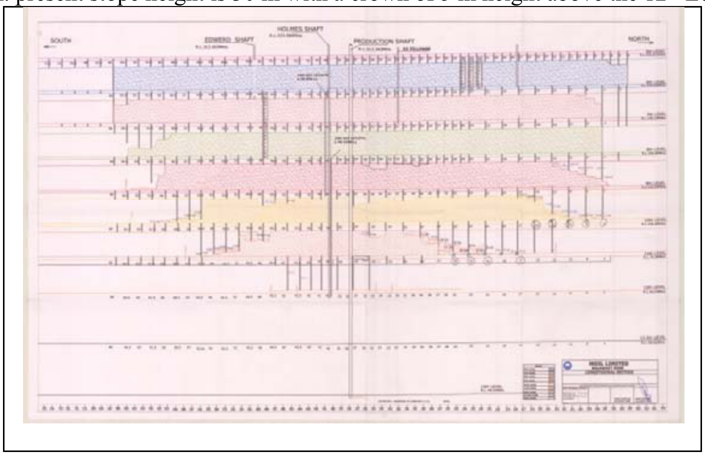


Fig. 6. Present L-Section of Balaghat Mine

4.4. Geo-mining conditions considered for modelling

The following parameters have been considered for modelling [8];

Stope height	=	45m
Stope length	=	60 m centre to centre of rib pillar
Rib pillar	=	6 m
Crown pillar	=	5 m
Orebody dip	=	60^o, 25^o
Orebody thickness	=	9 m to 20 m
Depth of cover at 12th Level	=	320 m
Height of opening between filled material and stope back	=	3 m

With this rock mechanics study, stope back stability and support design for 45 mtr level interval has been derived. The ore body thickness varies from 9 m to 20 m. Therefore, analysis was performed for stope spans of 9 m, 12 m, 15 m, 18 m and 20 m. after extraction of half of the stope leaving an opening of 3 m and sand filling in rest of the stope along with installation of cable bolts of 12 m in stope back, safety factor analysis is performed. A factor of safety of 1.0 or less is representative of the unstable zone.

The numerical modelling concludes that

a. Height of failure zone in stope back does not vary much from the previous levels where 30 m stope height was followed. Thus cable bolts of 12 m length with rock bolts of 1.5 m length as the earlier practice is adequate and does not require any changes.
b. Increasing the level interval to 45 m has no significant impact on the stope back stability. Thus, the stope back support system would remain same as the current practice

The rock mechanics study reveals that the increase in stope height from existing 30m to 45m has no significant effect on the stope back and placement of haulage roadway. For assessing the wall rock stability, strains bars are installed at 11th Level given in Fig 7 and 8.

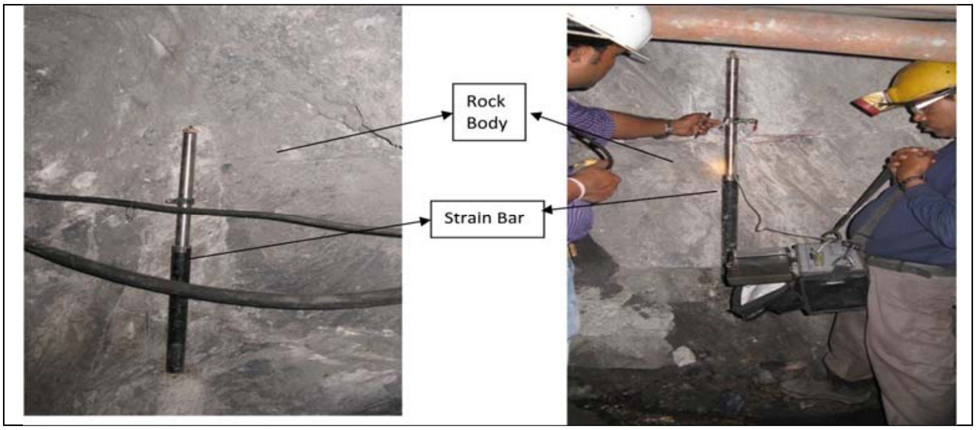


Fig. 7. Strain bar installed at 11th L

Fig. 8. Reading of Strain Bars

5. Conclusions

The above rock mechanics studies have been helped to improve safety and productivity of Balaghat Mine by phased mechanization of underground which is listed below;

- Face productivity has been improved from 2.4 T to 3.5 T in cable bolted stopes.
- Mechanical handling of ROM has been successfully introduced and improved the face productivity up to 9 T.
- Single boomer crawler mounted electro- hydrostatic drill jumbo has been introduced at 13.5th Level along with LHD for drive development. Gallery dimensions have been increased.
- Level interval has been increased from 30m to 45m below 12th Level
- Stope geometry has been changed and panel operation has been successfully implemented.
- Valuable timber required for square set has been eliminated and it is helping to preserve the environment.

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