Exploitation of mica deposits at Nellore mica belt, Andhra Pradesh, India

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India is the leading producer of sheet mica and a major part of this is exported. Nellore mica belt is the largest mica-producing area covering part of Nellore district in Andhra Pradesh, India. As most of the mines are old and privately operated, they are developed and operated purely based on local experiences. In this article, we highlight the problems associated with the present mica-mining practices in the Nellore mica belt, and scientific approaches that have been adopted for fixing different parameters associated with mica extraction. Based on detailed field study, geo-mechanical data and tested rock properties, extensive numerical modelling is done to suggest the best possible method of mining for safe and sustainable mica extraction from the area.

Keywords: Barrier thickness, mica deposits, overhand stoping, pegmatite, pillar design.

DUE to its excellent dielectric strength, insulating properties, low power-loss factor and resistance to high voltage, mica is one of the important minerals used in electrical and electronics industries. For its unique combination of transparency, flexibility and toughness, it is also extensively used in the aircraft industry. Mica is also used in the beauty and personal-care sectors to give the shiny and glittery appearance in products like cosmetics and toothpaste. India is the leading producer of sheet mica and a major part of this is exported. According to the United Nations Framework Classification (UNFC) of Mineral Reserves/Resources, the total resources of mica in the country as on 1 Aril 2015 were estimated at 635,302 t of which 114,433 t was placed under reserves category and 520,869 t under remaining resources category. Andhra Pradesh (AP) leads with 41% share in the country's total resources followed by Rajasthan (28%), Odisha (17%), Maharashtra (13%), Bihar (2%), and the remaining is in Jharkhand and Telangana[1]. As mica is declared as a 'minor mineral' as per Government of India Notification S.O. 423(E), dated 10 February 2015, hence production data after 2014–15 are not available with the Indian Bureau of Mines (IBM). However, IBM gathered some of the production data from individual states. According to these data, production from Andhra Pradesh for the years 2015–16, 2016–17 and 2017–18 was 26,783, 53,630 and 15,217 t respectively. For Rajasthan, it was 5,513, 3,124 and 6,459 t respectively, for the corresponding years[1]. According to 2014–15 IBM data, a 62% decline in mica production was reported compared to the preceding year. Only 31 mines reported mica production during 2014–15 as against 39 in the previous year[2]. Crude mica production was reported only from Andhra Pradesh during that year. There was a gradual decline in mica production reported from the Indian mica belts. As a result, many proponents have started believing in the end of the mica industry. Further, in the recent past, efforts have been made by developed countries to find substitutes for replacing mica and its products, but these are found to be neither perfect nor cost-effective[3]. According to the Transparency Market Research report, the global mica market revenue is expected to increase from US$ 478.1 million in 2015 to US$ 669.3 million by 2024 (ref. 4). The market growth of mica is expected to increase due to its applications across a diverse set of industries, such as electronics, construction, cosmetics, plastic, rubber, paints and coatings. According to IBM records, China alone imported 88,146 t of mica in 2014–15. There are bright prospects for this industry even today. In this context, the present study highlights the geo-technical problems associated with the current mica mining practices in the Nellore mica belt, AP and scientific approaches that have been adopted for fixing different geo-mining issues for safe and sustainable mica mining.

Nellore mica belt

Nellore mica belt is the largest mica-producing area in India, covering part of Nellore district, AP and a total area of 2500 km^2. Mica is extracted in the area either by opencast or underground mining method. The opencast is restricted to a very shallow depth of cover, whereas underground mines are developed generally up to 100 m depth of cover[2]. The mica belt in Nellore district is composed of garnetiferous mica schist, kyanite-bearing chloriteschist and garnetiferous hornblende gneiss intruded by pegmatites and quartz veins. These rocks strike

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northwest and dip in a westerly direction at 60°–70°. A green mica-bearing zone exists in the western part of the area, while in the eastern part, a well-marked ruby mica-bearing zone is found. Pegmatites of the 'ruby mica zone' occur either as lit-par-lit injections in the schist or fillings of existing open spaces with few schistose intercalations. Mica distribution is more regular in the ruby mica zone. In the case of the green mica zone; distribution of mica is neither regular nor even, but reach books are encountered along the fractures. Commonly, a great concentration of mica occurs near the hanging wall and footwall. Sometimes, the hanging wall shows better development of mica than the footwall, or vice versa[5].

Present mining methods and associated problems

All the mica mines in the Nellore mica belt are opened up first as prospecting pits. These trial workings are later developed into opencast workings from 5 to 10 m depth, known as Upper Challa[2]. The nature and quality of the yield decide whether the underground method has to be adopted for the mining of mica. Due to dipping in nature, underground mining of mica is performed by the overhand stoping method with waste backfilling. By driving vertical or inclined shaft, the mica-bearing pegmatite is opened up. Driving and stoping are done only in those areas where mica is confined. The levels are driven in the orebody along the hanging wall or footwall. This enables the vein to be blocked out of suitable size, leaving considerable reserves in the blocks between the drives and winzes. Wall and roof are generally self-supporting[2].

As most of the mines in Nellore district are very old and privately operated, they are developed based on the nature and quality of the mica-bearing pegmatite. The mica mine owners cannot afford to develop the total underground mine without production till stoping. Therefore, the level intervals are generally maintained at 5–10 m. The level interval of 5–10 m also helps to have a close control on the production of pegmatite. It is also found that the mineralogical content of pegmatite is similar to the composition of hard granite used for construction. From field experiences, it is found that pegmatite is so hard that a full round blasting by drilling a 1 m deep hole with blast pattern (1.8 m × 1.8 m × 1 m) is not breakable easily. The powder factor is found to be 1.35. This indirectly proves the hardness of the pegmatite body. Based on the above field experiences by keeping 5–10 m level interval, no underground hazards are anticipated by the mine management from stope stability and strata control point of view.

Design of stope has to be analysed based on the depth of cover, stability of stope back and corresponding support design. Some underground mica mines are also developed below or parallel and very close to the opencast mine. No study has been found to evaluate the optimum vertical barrier thickness between the floor of the opencast and underground workings, or safe horizontal distance between them. Here we present scientific approaches that have been adopted for fixing the stope sizes at different depths of cover and support design for the stability of stope backs during development and extraction of the stope for a particular mica mine. Optimum vertical as well as horizontal barrier thicknesses between opencast and underground workings are also evaluated in different geo-mining conditions for the mine.

Approaches adopted for analysing various parameters

Theoretical as well as numerical modelling approaches have been adopted to address the problems and find a feasible solution. The different approaches adopted for stability assessment of various structures with different geomining conditions are discussed below.

Stability assessment of stope

In the normal method of underground mining as practised in Nellore mica belt, the vein is opened up or developed by vertical shafts, drives and winzes. The size of the drives and inclines is generally 2.5 m × 2 m to 2 m × 1.8 m. The levels are driven in the pegmatite along the hanging wall or footwall. This enables the vein to be blocked out of suitable size, leaving considerable reserves in the blocks between the drives and winzes. The size of the blocks usually depends on the distance between the drives. The level intervals vary from 5 to 10 m with rises and winzes at 10 m interval or more, according to the directives of IBM. In the Metalliferous Mines Regulations, India 1961, there is no specification on the size of the pillars or level intervals[6]. It is mentioned that the size of the pillars or level intervals should be maintained depending on the strength of the pillar and roof considering the safety of the underground workers.

As no guidelines are available for block size for different depths of cover, various permutations and combinations of block size are made for different depths of cover in this study considering the required safety factor from the stability point of view. Safety factor is generally selected based on an assessment of pillar performance or statistical analysis of failed and stable cases. A safety factor of 1.5 is sufficient for long-term stability of the block in hard rock[7–10]. Considering old and privately operated mines within the study area, a safety factor of 2.0 is chosen in this study for parameters related to mica extraction. Various empirical equations are available for estimating the pillar strength of hard rock, which is well documented[9–11]. For safety factor calculation of the block in the present case, its strength is estimated using pillar strength equation (eq. 1)[10] and the load on it is assessed by the tributary area method (eq. 2)[7,12–14]. The estimation of pillar strength is done using the following formula[10]

$$S = 0.65 \times \text{UCS} \times \text{LDF} \times (w_e^{0.3}/h^{0.59}), \quad (1)$$

where S is the strength of the pillar (MPa), UCS the uniaxial compressive strength of intact rock (MPa), h the working height (m), w_e the equivalent width of the pillar (m) $= w + (4A/C - w) \times \text{LBR}$ for a rectangular pillar, where w is the smaller width of the rectangular pillar (m), A the area of the rectangular pillar (m^2), C the perimeter of the rectangular pillar (m), LDF the large discontinuity factor and LBR is the length benefit ratio.

LBR is related to the width to height ratio of a block. For a width to height ratio of 1.4 or more, LBR is equal to 1.0. If no large discontinuities are present, LDF will be equal to 1.0. As the study area is free from any large discontinuities, LDF is taken as 1.0 for the pillar strength equation. Load on pillars is estimated using the tributary area method as

$$P = \frac{\gamma H (w_1 + B)(w_2 + B)}{w_1 \cdot w_2}, \quad (2)$$

where P is the load on the pillar (MPa), γ the unit rock pressure (0.028 MPa/m), H the depth of cover (m), w_1 the solid pillar width along the level (m), w_2 the solid pillar width along dip-rise (m) and B is the roadway width (m).

A safety factor of the blocks can be calculated using the following equation

$$\text{Safety factor} = \frac{\begin{array}{c}\text{Strength of the block}\\ \text{calculated using eq. (1)}\end{array}}{\begin{array}{c}\text{Load on the block calculated}\\ \text{using eq. (2)}\end{array}} = \frac{S}{P}. \quad (3)$$

Although the present underground mine depths are restricted up to 100 m, calculations are done up to 450 m considering the future development of the mine. Based on eqs (1)–(3) for an average safety factor of 2.0, Table 1 gives the recommended size of the mica blocks at different depths of cover.

Table 1. Recommended size of mica blocks at different depths of cover for the mine under study

Depth (m)	Block size (centre to centre; m × m)	Gallery size (width × height; m × m)
Up to 60	5 × 9	2 × 2
60–240	7 × 9	2 × 2
240–360	10 × 9	2 × 2
360–450	12 × 9	2 × 2

Method of stoping and support design

The mica mines in the area generally practice the cut and fill method of stoping with waste rockfill. Cut and fill mining can be used in steeply dipping as well as mildly dipping ore bodies with reasonably firm ore. Small as well as large deposits with an irregular outline can be worked; thus, it is a versatile method. Selective mining is also possible. This method is preferred over other mining methods to prevent surface subsidence. As the cut and fill method is being practiced successfully for several decades in the region, the same was found to be suitable for stoping of mica ore. In the cut and fill method of mining, the ore is extracted by drilling and blasting in horizontal slices starting from the bottom of a stope and advancing upwards. A slice has a thickness of not more than 2 m, as all the mica mines are being worked manually.

Figure 1 shows the longitudinal and transverse sections of a block to be extracted by the cut and fill method. In the first stage, stoping operations commenced from the lower level to the upper level by breaking the blocks in slices and advanced longitudinally from inbye end to the outbye end of the block. Ore body is extracted from the hanging wall to the footwall, and the same is filled with waste in steps as shown in Figure 2. The procedure mentioned above is continued until the final stage is reached. In the final stage after leaving 2 m block as ‘crown pillar’, extraction is done from the top level by the underhand method of drilling (Figure 3).

Support design during stoping

Based on the safety factor contours obtained from numerical modelling, design of support systems for reinforcing the stope back is formulated using the height of an unstable region in the immediate stope back. For this, a two-dimensional numerical model is run for intermediate stoping using FLAC3D software[15]. Physico-mechanical properties of intact rocks as tested in the laboratory are converted into rock mass properties prior to their use in numerical simulation (Table 2). All the geological parameters like layer thickness, structural features, weather ability, rock strength, groundwater seepage rate are taken into consideration during evaluation of rock mass rating (RMR) of the ore body, hanging wall and footwall. RMR as obtained from field study for the ore body, hangwall and footwall is found to be 65, 70 and 70 respectively[16].

Sheorey’s[17] failure criterion for rock mass has been used for numerical modelling. This criterion uses the 1976 version of RMR of Bieniawski for reducing the laboratory strength parameters to give the corresponding rock mass values. This criterion is defined as

$$\sigma_1 = \sigma_{\text{cm}} \left(1 + \frac{\sigma_3}{\sigma_{\text{tm}}}\right)^{b_{\text{m}}}, \quad (4)$$

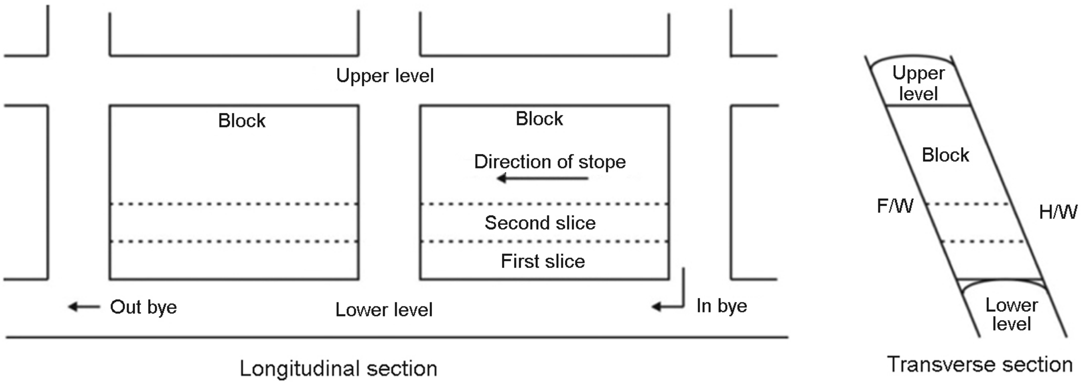

Figure 1. Longitudinal and transverse sections showing the stoping procedure.

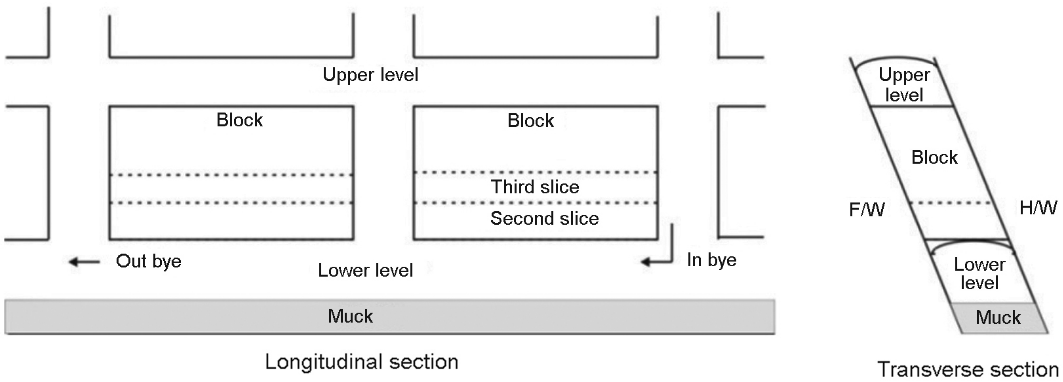

Figure 2. Longitudinal and transverse sections showing the first stage of stoping with filling.

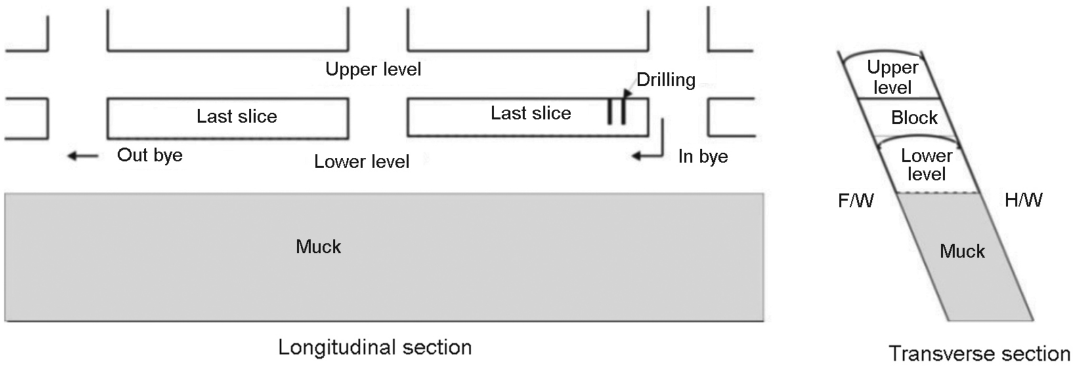

Figure 3. Longitudinal and transverse sections showing the last stage of stoping with filling.

where

$$\sigma_{cm} = \sigma_c \exp\left(\frac{\text{RMR} - 100}{20}\right), \quad (5)$$

$$\sigma_{tm} = \sigma_t \exp\left(\frac{\text{RMR} - 100}{27}\right), \quad (6)$$

$$b_m = b^{\frac{\text{RMR}}{100}}. \quad (7)$$

σ_c is the intact rock compressive strength (MPa), σ_t the intact rock tensile strength (MPa), σ_{cm} the rock mass compressive strength (MPa), σ_{tm} is the rock mass tensile strength (MPa) and b and b_m are the exponents in the criterion for intact rock and rock mass respectively. The factor of safety (F) is defined as

$$F = \frac{\sigma_1 - \sigma_{3i}}{\sigma_{1i} - \sigma_{3i}}. \quad (8)$$

Table 2. Input parameters used for modelling

Rock type	Young's modulus (GPa)	Poisson ratio	Compressive strength (MPa)	Tensile strength (MPa)	Density (kg/m^3)
Ore body	19.99	0.22	78.77	7.8	2549
Foot wall	11.44	0.17	65.55	6.5	2925
Hanging wall	16.24	0.14	94.64	9.4	3085

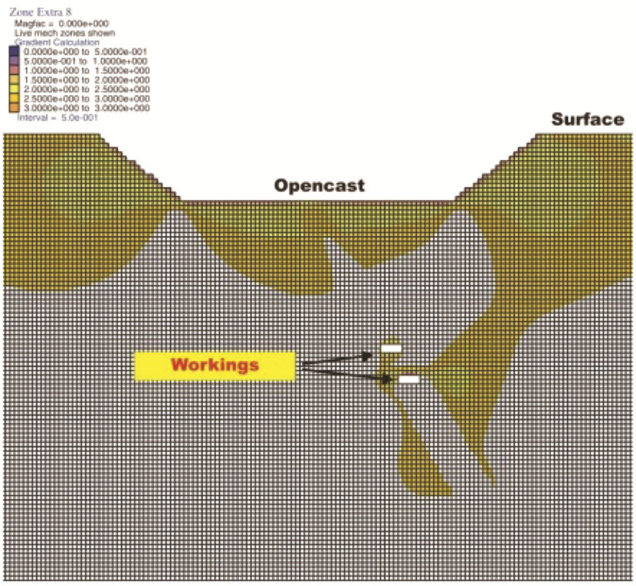

Figure 4. Grid pattern used in 2D numerical model.

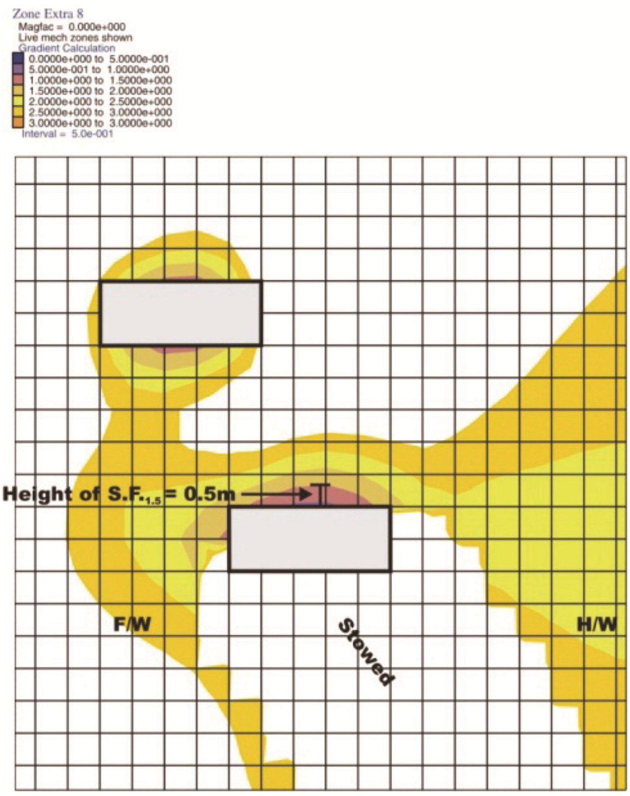

Figure 5. Contour of safety factor showing height of the rock load to be supported.

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

$$F = \frac{\sigma_{tm}}{-\sigma_{3i}}. \qquad (9)$$

In eqs (8) and (9), σ_{1i} and σ_{3i} are the major and minor induced stresses from the numerical model output respectively. The sign convention followed here is negative for tensile stress and positive for compressive stress. The boundary conditions for the model consist of roller boundary conditions at the four sides, fixed boundary conditions at the bottom and truncated load is applied at the top of the model depending on the depth of cover. Extraction of mica blocks/pillars has been simulated in the numerical model in stages. In the first stage, the virgin 2D model (Figure 4) is run using the rock-mass properties given in Table 2 and in the second stage, extraction of mica blocks with filling is carried out up to an intermediate level.

The factor of safety is estimated using the principal stresses obtained from numerical modelling and rock-mass strength estimated by Sheorey's failure criterion reveals an unstable region in the stope back. As stopes are temporary in nature, a safety factor of 1.5 is sufficient for stability of the stope back. Using the safety factor value of 1.5, the average height of the unstable zone in the stopeback is measured (Figure 5) and used for estimating the required rock load (R_L) density for reinforcing the stopeback as

$$R_L = \rho \times h_{1.5}, \qquad (10)$$

where R_L is the required rock load density (t/m^2), ρ the rock density, 3.0 t/m^3 of the host rock, and $h_{1.5}$ is the height of the unstable region (m, i.e. the height of factor of safety 1.5).

Figure 5 shows that the contour of the factor of safety 1.5 is confined within the height of 0.5 m in the immediate roof of the stope. Therefore, the required rock load density for reinforcing the stope back is $R_{Ls} = 0.50$ m $\times$ 3.0 t/m^3 = 1.5 t/m^2. As all the mica mines are manually operated and are habituated with timber supports, it is, proposed to support the stope back with 2 m long timber prop having load-bearing capacity of the wooden prop as 6 t. Figure 6 shows the plan view of the proposed support pattern for reinforcing the stope back. If props are placed in a grid pattern of 1.5 m $\times$ 1.5 m, applied support load (A_L) density in the stopeback can be estimated as

$$A_{\mathrm{L}} = \frac{n \times b_{\mathrm{c}}}{w \times s_{\mathrm{p}}} \tag{11}$$

where n is the number of props in a row (four numbers), b_c the load-bearing capacity for a 2 m long wooden prop (6 t), w the width of the stope (5 m according to the model) and s_p is the spacing between consecutive two rows (1.5 m). The applied support load in the stope back, $A_{Ls} = (4 \times 6)/(5 \times 1.5) = 3.2$ t/m^2. Accordingly, the safety factor = 3.2/1.5 = 2.14, which is more than that required for short-term stability (1.5).

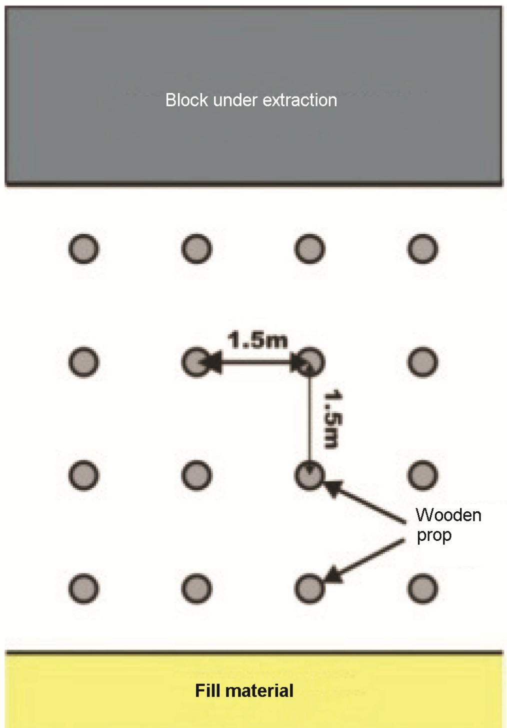

Figure 6. Plan view of the proposed support pattern for reinforcing the stope back.

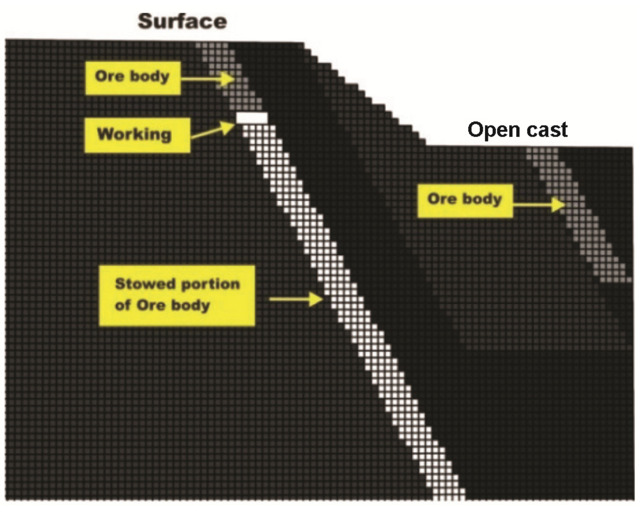

Figure 7. Two-dimensional grid used for modelling to determine the optimum horizontal barrier thickness between opencast and underground workings.

Optimum horizontal barrier thickness between opencast and underground workings

Two types of ground conditions prevail in the study area of the particular mine. In the first condition, two parallel pegmatites are extracted, one by the open cast and another by the underground method. In the second condition, the thick pegmatite ore body is extracted by the open cast as well as underground methods along the strike. In both situations, optimum barrier thickness between opencast and underground workings is evaluated.

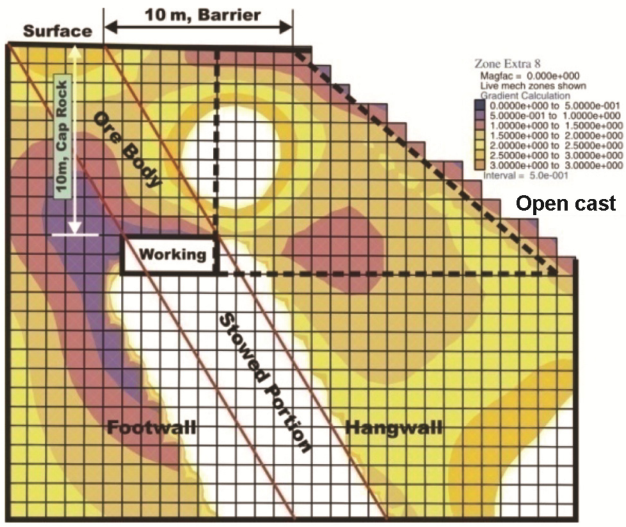

Figure 8. Safety factor contour of a representative model showing horizontal barrier between opencast and underground workings.

Table 3. Average safety factor of horizontal barrier between opencast and underground workings for different thicknesses of the ore body

Thickness of orebody (m)	Dry	Filled with over burden rock	Filled with water
For 5 m thick horizontal barrier			
3	2.18	2.20	2.20
4	2.10	2.10	2.12
5	2.01	2.01	2.03
6	1.94	1.94	1.96
7	1.88	1.88	1.90
For 10 m thick horizontal barrier			
3	2.67	2.68	2.69
4	2.60	2.60	2.61
5	2.53	2.52	2.54
6	2.47	2.46	2.48
7	2.41	2.40	2.41

For determining the optimum horizontal barrier thickness between opencast and underground workings, when two parallel pegmatites are extracted one by the open cast and another by the underground method, 2D numerical modelling is performed by varying the barrier thickness. It is found that there is variation in the horizontal width of the ore body with a dipping of about 60°. To simulate all the possible variations, the horizontal width of the ore body has been altered from 3 to 7 m. Models are also run in three conditions, keeping the open cast space filled with water; filled with overburden dump, and also in empty and dry conditions to simulate all the possibilities. Models are made assuming that the ore body has been extended in the strike direction up to sufficient length. The horizontal barrier thickness values of 5 and 10 m are considered for all the models. Figure 7 shows the 2D grid used for modelling. Thirty different models are run according to the conditions mentioned above. Using Sheorey's failure criterion and the in-built FISH program of FLAC3D, safety factor of the horizontal barrier between opencast and underground workings (dash portion, Figure 8) is calculated (Table 3). Figure 8 shows a safety factor contour of the representative model. An average safety factor of 2.0 in numerical models is considered for long-term stability. From Table 3 and the safety factor contours plotted in Figure 8, it can be concluded that 10 m is the optimum horizontal barrier thickness between opencast and underground workings, which gives a safety factor more than 2.0 in all the cases with varying ore body thicknesses for this particular case.

To determine the optimum horizontal barrier thickness between opencast and underground workings when the same pegmatite is extracted by both the methods in the strike direction of the pegmatite, three-dimensional numerical modelling is done with a 10 m barrier thickness as established above. Figure 9 shows a schematic vertical section along strike. Figure 10 shows the grid used for numerical modelling. Fifteen different models are run according to the conditions mentioned as above using Sheorey's failure criterion. The average safety factor of the horizontal barrier between opencast and underground workings is calculated (Table 4). From Table 4, it is obvious that 10 m is the optimum horizontal barrier thickness between opencast and underground workings,

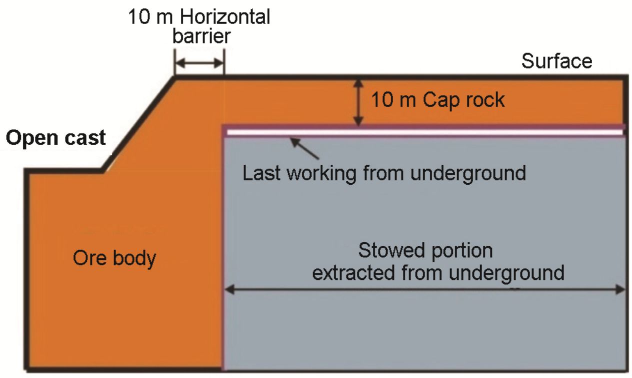

Figure 9. Schematic vertical section along strike showing the horizontal barrier to be left between opencast and underground workings when the same pegmatite is extracted using both the methods.

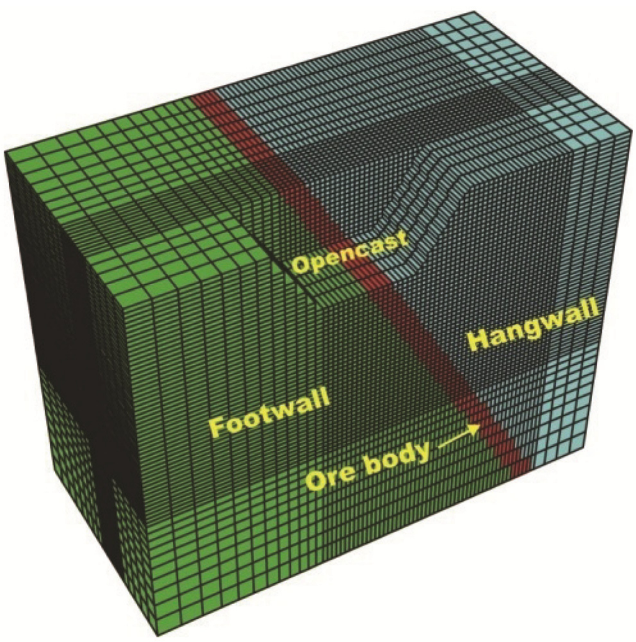

Figure 10. Three-dimensional grid used for modelling.

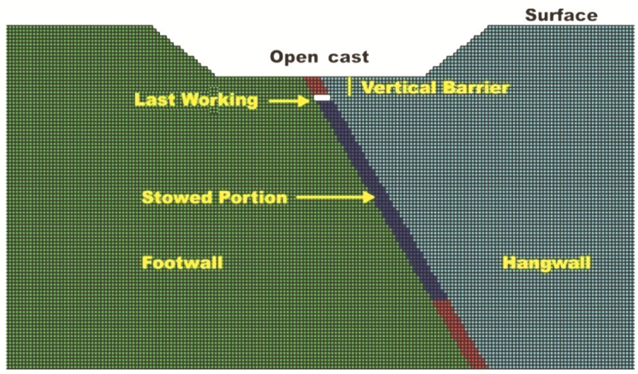

Figure 11. Grid used for modelling to determine the optimum vertical barrier between opencast and underground workings.

Table 4. Average safety factor of 10 m horizontal barrier between opencast and underground workings when the same pegmatite is extracted by both methods

Thickness of ore body (m)	Dry	Filled with over burden rock	Filled with water
3	2.52	2.55	2.63
4	2.67	2.70	2.76
5	2.77	2.79	2.85
6	2.84	2.86	2.91
7	2.88	2.90	2.96

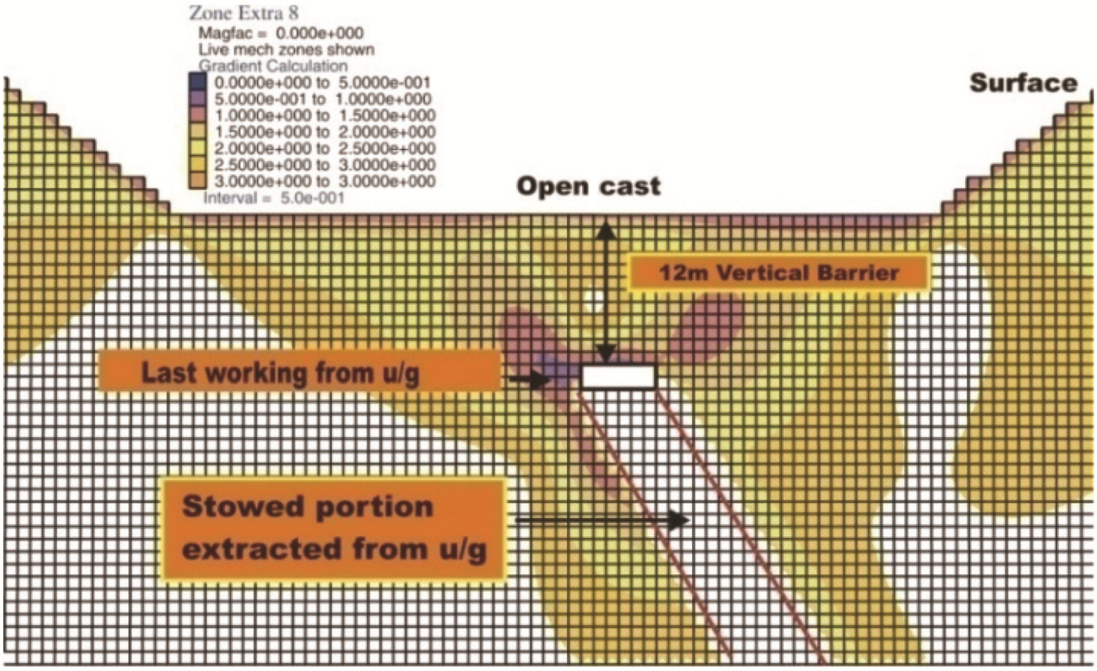

Figure 12. Safety factor contour of the representative model used to determine the optimum vertical barrier between opencast and underground (u/g) workings.

Table 5. Average safety factor of different thicknesses of vertical barrier and ore body between opencast and underground workings

Thickness of vertical barrier (m)	Dry	Filled with over burden rock	Filled with water
For 3 m thick ore body			
5	1.43	1.59	1.47
8	1.77	1.99	1.89
9	1.88	2.09	1.93
10	1.98	2.22	2.04
12	2.17	2.43	2.23
For 4 m thick ore body			
5	1.36	1.52	1.40
8	1.70	1.91	1.75
9	1.82	2.02	1.86
10	1.92	2.16	1.98
12	2.11	2.37	2.17
For 5 m thick ore body			
5	1.33	1.48	1.37
8	1.67	1.87	1.71
9	1.78	1.97	1.82
10	1.89	2.11	1.93
12	2.08	2.33	2.14
For 6 m thick ore body			
5	1.31	1.45	1.35
8	1.64	1.84	1.69
9	1.75	1.94	1.80
10	1.86	2.08	1.91
12	2.05	2.30	2.11
For 7 m thick ore body			
5	1.30	1.44	1.33
8	1.63	1.83	1.68
9	1.74	1.93	1.77
10	1.84	2.06	1.89
12	2.03	2.27	2.09

which gives a safety factor more than 2.0 in all the cases with varying ore body thickness under different conditions of the open pit.

Optimum vertical barrier thickness between opencast and underground workings

To determine the optimum vertical barrier thickness between opencast and underground workings, numerical modelling is performed by varying the barrier thickness. Figure 11 shows the grid used for numerical modelling. From the stability point of view, the average safety factor of the vertical barrier must be more than 2.0 as well as strain developed on the floor of the opencast working due to underground extraction must be less than the maximum permissible limit of 3.0 mm/m. The calculation of strain of the floor of the opencast is done in anticipation of cracks generated due to the underground mining, if any, through which accumulated water in the opencast may enter into the underground workings. Forty-five different models are run and analysed using Sheorey's failure criterion. The average safety factor of the vertical barrier between opencast and underground workings is calculated (Table 5). Figure 12 shows a safety factor contour of the representative model. From Table 5 and the safety factor contours plotted in Figure 12, it is clear that 12 m is the optimum vertical barrier thickness between opencast and underground workings, which gives a safety factor more than 2.0 in all the cases with varying ore body thicknesses for this case study.

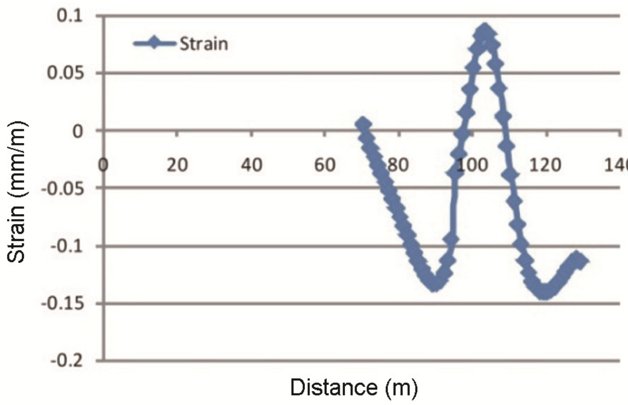

Figure 13. A representative plot showing strain developed in open pit due to extraction of 3 m thick ore body by underground workings for a 12 m thick vertical barrier.

Table 6. Tensile and compressive strains due to the extraction of varying widths of the ore body

Thickness of vertical barrier (m)	Tensile strain (mm/m)	Compressive strain (mm/m)
For 3 m thick ore body		
5	(+) 0.145	(–) 0.462
8	(+) 0.118	(–) 0.249
10	(+) 0.103	(–) 0.181
12	(+) 0.089	(–) 0.142
For 4 m thick ore body		
5	(+) 0.150	(–) 0.523
8	(+) 0.130	(–) 0.283
10	(+) 0.114	(–) 0.205
12	(+) 0.095	(–) 0.168
For 5 m thick ore body		
5	(+) 0.128	(–) 0.562
8	(+) 0.136	(–) 0.305
10	(+) 0.118	(–) 0.221
12	(+) 0.102	(–) 0.188
For 6 m thick ore body		
5	(+) 0.139	(–) 0.589
8	(+) 0.122	(–) 0.321
10	(+) 0.116	(–) 0.240
12	(+) 0.108	(–) 0.205
For 7 m thick ore body		
5	(+) 0.149	(–) 0.609
8	(+) 0.113	(–) 0.333
10	(+) 0.111	(–) 0.256
12	(+) 0.100	(–) 0.218

Strain analysis

Underground extraction may result in some surface movements. These may occur over a period of time during and after the underground stoping operations. Ground movements result in horizontal displacement on the surface. These horizontal displacements result in horizontal strains of compressive (–ve) as well as tensile (+ve) nature. This can be measured by the following method

$$S_{x_{i/i+1}} = \frac{dx_i - dx_{i+1}}{D}, \qquad (12)$$

where $S_{x_{i/i+1}}$ is the horizontal strain between two adjacent surface grid points dx_i and dx_{i+1} respectively, and D is the distance between the points. The tensile and compressive strains due to the extraction of varying widths of the ore body are calculated (Table 6). Figure 13 is a representative plot showing strain developed in the open pit due to extraction of underground workings. From Table 6 and the representative plot in Figure 13, it is clear that the strain developed on the floor of the open pit due to extraction of varying widths of the ore body is well within the maximum permissible limit of 3.0 mm/m. So, there is no danger of inundation of underground workings even if water gets accumulated in the open pit area in this study. In future, after completion of underground extraction even within this 12 m barrier thickness, mica ore can be extracted by an opencast method with proper safety precautions.

Conclusion

Nellore mica belt is the largest mica-producing area in India. This study highlights the problems associated with the present mica mining practices in the Nellore mica belt, and scientific approaches that have been adopted for fixing different parameters associated with mica extraction. Based on the detailed field investigation, geomechanical data and tested rock properties, extensive numerical modelling is done to suggest the best possible method of mining for safe and sustainable mica extraction from the area. The findings of this study will be helpful in better understanding the issues and designing safe geo-mining parameters associated with the exploitation of mica from the region. As most of the near-surface mica-bearing pegmatite has been exhausted, this study will help in exploiting the deep-seated mica-bearing pegmatite.

1. IBM, *Indian Minerals Yearbook 2018* (*Part-III: Mineral Review*) *57th Edition, Minor Minerals 30.16 MICA* (*Final Release*), Indian

Bureau of Mines, Ministry of Mines, Government of India (GoI), June 2019; https://ibm.gov.in/writereaddata/files/06292019105720Mica.pdf (accessed on 11 July 2019).
2. IBM, *Indian Minerals Yearbook 2016 (Part-III: Mineral Review) 54th Edition, MICA* (*Final Release*), Indian Bureau of Mines, Ministry of Mines, GoI, 2017; http://ibm.gov.in/writereaddata/files/08092017100711Mica2015(Final).pdf (accessed on 12 February 2019).
3. Ganesh, P., Mishra, S. B. and Yadav, S. P., Issues and challenges of Indian mica industry. *Inventi Impact – Tech. Res. Rev.*, 2015, **4**, 187–190.
4. https://www.bizvibe.com/blog/mica-industry-india/ (accessed on 16 May 2018).
5. Madhukar, B. B. L. and Srivastava, S. N. P., *Mica and Mica Industry*, Taylor and Francis, 1995.
6. The Metalliferous Mines Regulations, India, 1961; http://www.dgms.net/mmr.pdf (accessed on 10 June 2018).
7. Salamon, M. D. G. and Munro, A. H., A study of the strength of coal pillars. *J. S. Afr. Inst. Min. Metal.*, 1967, **68**, 55–67.
8. Mark, C., Analysis of longwall pillar stability (ALPS): an update. In Proceedings of the Workshop on Coal Pillar Mechanics and Design, US Bureau of Mines, Information Circular 9315, 1993, pp. 238–249.
9. Hoek, E., Kaiser, P. K. and Bawden, W. F., Support of underground excavations in hard rock. 2000, pp. 01-225; https://www.mirarco.org/wp-content/uploads/mirarco_books/Support_of_Underground_Excavations_in_Hard_Rock.pdf
10. Esterhuizen, G. S., Dolinar, D. R. and Ellenberger, J. L., Pillar strength and design methodology for stone mines. In Proceedings of the 27th International Conference on Ground Control in Mining, Morgantown, West Virginia, USA, 29–31 July 2008.
11. Bieniawski, Z. T., *Rock Mechanics Design in Mining and Tunnelling*, A. A. Balkema, Rotterdam, The Netherlands, 1984.
12. Martina, C. D. and Maybee, W. G., The strength of hard-rock pillars. *Int. J. Rock Mech. Min. Sci.*, 2000, **37**, 1239–1246.
13. Babcock, C., Morgan, T. and Haramy, K., Review of pillar design equations including the effects of constraint. In *Proceeding of the 1st Annual Conference on Ground Control in Mining* (ed. Peng, S. S.), West Virginia University, Morgantown, USA, 1981, pp. 23–34.
14. Hedley, D. G. F. and Grant, F., Stope-pillar designs for the Elliot Lake uranium mine. *Bull. Can. Inst. Min. Metal.*, 1972, **65**, 37–44.
15. FLAC3D, Fast Lagrangian Analysis of Continua in 3 Dimensions, version 3.01, Itasca Consulting Group, Inc., Minneapolis, USA, 2010.
16. Bieniawski, Z. T. (ed.), Rock mass classification in rock engineering. In Proceeding Exploration of Rock Engineering, Balkema, Rotterdam, The Netherlands, 1976.
17. Sheorey, P. R., *Empirical Rock Failure Criteria*, Balkema, Rotterdam, The Netherlands, 1997.

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