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RESEARCH ARTICLE – EARTH SCIENCES

Blast-Induced Caving from Surface over Continuous Miner Panel at a 110 m Cover in an Indian Mine

Arvind K. Mishra · Awadh K. Mishra · Manamohan Rout

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Abstract Continuous mining is a most popular mining method in USA and Europe. Nearly 60 % of coal mining in the West is through continuous mining process in underground mines. Continuous miner technology, which has of late been introduced in India, is able to extract 65–74 % of the coal in a panel depending upon the seam parameters as against 50–60 % by semi-mechanized Bord and Pillar system of mining with SDL/LHDs, which is popular in India. The rate of extraction by the continuous miner being high is able to create larger panels, and thus, reduces the coal loss in barrier pillars. Presently, efforts are being made to introduce this technology on a larger scale in underground coal mines of Coal India Limited in India. When the first depillaring panel started its operation in November 2009 at Jhanjra, the panel was stopped after extraction of nine pillars by the Indian regulatory body (Directorate General of Mines Safety, DGMS, GOI) as the geological conditions of the overlying strata gave trouble of non-caving. The anticipated roof fall did not take place. The exposed roof area reached 10,400 m^2, which was equivalent to 46,800 m^3 of the void volume posing a threat of air blast. Since all the attempts to induce caving by underground blasting failed, an attempt was made from surface to induce the caving by destress blasting. This paper deals with the design of blasting parameters and execution of destress blasting keeping in view the safety and sentiments of nearby villages.

Arvind K. Mishra (✉) · M. Rout
Department of Mining Engineering, ISM, Dhanbad, India
e-mail: arvmishra@yahoo.com

M. Rout
e-mail: manamohanrout@gmail.com

Awadh K. Mishra
Jhanjra Project Colliery, ECL, CIL, Burdwan, India
e-mail: awadh_1234@rediffmail.com

Keywords Induced caving · Rock fracturing · Blasting parameters · Surface caving · Cavability · Destress blasting

الخلاصة

التعدين المستمر هو وسيلة التعدين الأكثر شعبية في الولايات المتحدة وأوروبا، حيث إن ما يقرب من 60٪ من تعدين الفحم في الغرب هو عبارة عن عملية تعدين مستمرة في مناجم تحت الأرض. وتكنولوجيا عامل المنجم المستمرة التي أدخلت في وقت متأخر في الهند لديها قدرة على استخراج 65٪ - 74٪ من الفحم في اللوح. وهذا يتوقف على معاملات التماس مقابل 50٪ إلى 60٪ تعدين بنظام ميكانيكية اللوح والعمود مع SDL / LHDs، الذي يتمتع بشعبية في الهند. ولأن معدل الاستخراج من قبل شركة التعدين المستمر مرتفع فهذا يعطي قدرة على إيجاد لوحات أكبر ، وبالتالي يقلل من فقدان الفحم في أعمدة الحاجز. وتبذل حاليا جهود لإدخال هذه التكنولوجيا على نطاق واسع في مناجم الفحم تحت الأرض لشركة فحم الهند المحدودة في الهند. وحين بدأت أول لوحة بدون عمود عملها في ديسمبر 2009 في جانارجا توقفت اللوحة بعد استخراج تسعة أعمدة من قبل الهيئة التنظيمية الهندية (المديرية العامة للسلامة المناجم، DGMS، GOI)، حيث إن الظروف الجيولوجية للطبقات المغطاة أظهرت عناء في عدم الرضوخ. ولم يحدث سقوط متوقع للسقف. وبلغت مساحة السقف المعرضة 10400 م^2، أي ما يعادل 46800 م^3 من حجم الفراغ مما يشكل تهديدا لانفجار الهواء. وبما أن كل المحاولات للحث على الرضوخ بوساطة التفجير تحت الأرض فشلت ، كانت هناك محاولة من السطح للحث على الرضوخ بوساطة التفجير من غير شد. تتناول هذه الورقة تصميم معاملات التفجير وتنفيذ تفجير من غير شد واضعين نصب أعيننا سلامة القرى المجاورة ومشاعر سكانها.

1 Introduction

1.1 Location of Mine

Jhanjra colliery of Eastern Coalfields Limited (ECL) is situated in the North-Eastern side of Raniganj Coalfield. The Grand Trunk road of National Highway-2 is about 15 km south of the Jhanjra Block and Durgapur Township is about 15 km by road to south-east of the area (Fig. 1). Jhanjra block

Fig. 1 Location map of mine

covering an area of about 11.50 sq. km is surrounded by a number of faults with throws varying from 15 to 110 m. Jhanjra block has eight extractable seams with a total reserve of about 200 million tonnes (Mt). The block has been divided into six sectors namely Sector A, B, C, D, E and F. In sectors 'C' and 'D', located towards the northern half of the Jhanjra fault-bounded block in R-VI seam, Jhanjra Colliery, the development of coal seam has been carried out by continuous miner and associated accessories. Development of the first continuous miner panel (CM1) had already been completed and the development work was yet to begin for the second panel (CM2).

1.2 Prevailing Condition at the Mine

It had been proposed to depillar the already developed panel (CM1) as well as the developing panel (CM2) using continuous miner. The quantitative and qualitative nature of the abutment loading is highly dependent upon the properties of the overlying rock strata [1–6]. Here, the characteristic of overlying strata is represented by *Cavability Index [7]. However, delay in natural roof caving in the depillaring panel CM1 has been anticipated due to the presence of hard-rock strata within the immediate roof of R-VI seam, which has Cavability Index of more than 5,000. It was presumed that the coal ribs (Rib—The side of a pillar or the wall of an entry.) as designed earlier would eventually collapse (as the designed factor of safety was 0.5) and provide no resistance to the caving. However, the natural caving had been delayed, as the sizes of ribs or stooks left might have caused this. The working had been made unsafe due to overhanging roof and was stopped by the management of mine and Inspector of Mines, Directorate General of Mines Safety, Government of India.

**Cavability Index* The Cavability Index,'I' is directly proportional to the intact rock compressive strength, bed thickness and average core length. This Index can be represented thus:

$$I = \sigma_c \frac{t^{0.5}}{5} L^n,$$

where I is Cavability Index, σ_c is intact rock compressive strength (kg/cm^2), t is bed thickness (m), L is average core length (cm) and n is 1.0 for RQD < 80 and 1.2 for RQD > 80.

The roof categorization can be done on the basis of Cavability Index:

Category of roof	Cavability Index
Category-I : Easily Cavable	Up to 2,000
Category-II : Moderate Cavable roof	Above 2,000–5,000
Category-III : Roof Cavable with difficulty	Above 5,000–10,000
Category-IV : Cavable with substantial difficulty	Above 10,000–14,000
Category-V : Cavable with extreme difficulty	Above 14,000

2 Management of Competent Roof Rock

The caving of a competent overlying roof can be managed by a controlled pre-fracturing or fracturing during workings [8]. The fracturing can be achieved by high pressure water jet or explosives. Both of these approaches are applied to pre-fracture the competent roof strata during long wall mining [9–12].

Long hole drilling and blasting can be used to manage caving of a competent overlying roof of a deep coal seam. Here, all these operations are to be carried out from working horizon of the seam only. A good approach for this purpose is described by Konicek [4,5,10,13]. Here, upward (inclined) drilling of around 100 m long holes in roof, across the massive strata, is done from the gate roads of a long wall face. Charging of these holes is done pneumatically and nearly 3,000–5,000 kg of rock-blasting explosives are blasted at time. However, this is difficult to be practiced in an Indian coal mine as that amount of explosive handling is not permitted in underground.

3 Geo-Mining Parameters of the Depillaring Panel

Within the depillaring area using continuous miner, depth of cover from surface to the R-VI seam varied between 110 and 140 m. The development works had been carried out up to the full seam thickness using continuous miner. The average pillar size was 26 m × 26 m, corner to corner. The height of the gallery varied between 4.0 m and 4.5 m, whereas the average width of the gallery was 6.0 m. The gradient of the seam was 1 in 16. The average width of the panel was 100 m (3 pillars width), whereas the length of the depillaring panel would be sectionalized as per the incubation period, rate of practical retreat, ventilation parameters etc. The average uniaxial compressive strength of coal (R-VI seam) was 25.82 MPa.

The overlying roof rock of R-VI consisted of medium to fine-granied sandstone. As per the report submitted by Roy et al. [14], three beds were identified with likelihood of difficulty in caving during depillaring where the Cavability Index was more than 5,000 in the immediate roof, which was considered up to 20 m from the coal seam. It was assumed that the overlying roof rock will be destressed if the immediate roof rock fails (Table 1).

Bed-I is laying immediately above the R-VI seam and the average thickness of this bed is 1.40 m. The Cavability Index of Bed-I is 5,222.66, which comes under the category of 'roof cavable with difficulty'. Bed-II is located at 4.68 m height from the coal seam roof and the average thickness of this bed is 0.88 m. The Cavability Index of bed-II is 24,466.54,which is under the category of 'roof cavable with extreme difficulty'. Bed-III is located at a height of 7.30 m from the roof of R-VI seam (Fig. 2).

The presence of these three beds might have caused delay of the natural roof fall in the depillaring panel. As per the technical report submitted by Roy et al. [14], the first main fall in the CM1 panel was expected after the area of roof exposure reaching about 6,748 m^2. Therefore, induced caving by blasting would be necessary to reduce the stress abutment in the working areas as well as to mitigate the impact of air blast if regular natural roof fall does not take place during initial phase of depillaring (Fig. 3).

For effective induced blasting, it was essential to fracture the main roof rock that delayed natural caving of an

Table 1 Classification of R-VI coal seam roof based on Cavability Index [14]

From the roof level of R-VI seam upward	Average core length (L) in cm	Bed thickness (t) in m	Cavability Index (I)	RQD	Remarks
Bed-I	43.0	1.40	5,222.66	92.14	Roof cavable with difficulty
Bed-II	78.0	0.88	24,466.54	88.64	Cavable with extreme difficulty
Bed-III	37.8	1.94	9,371.16	97.42	Roof cavable with difficulty

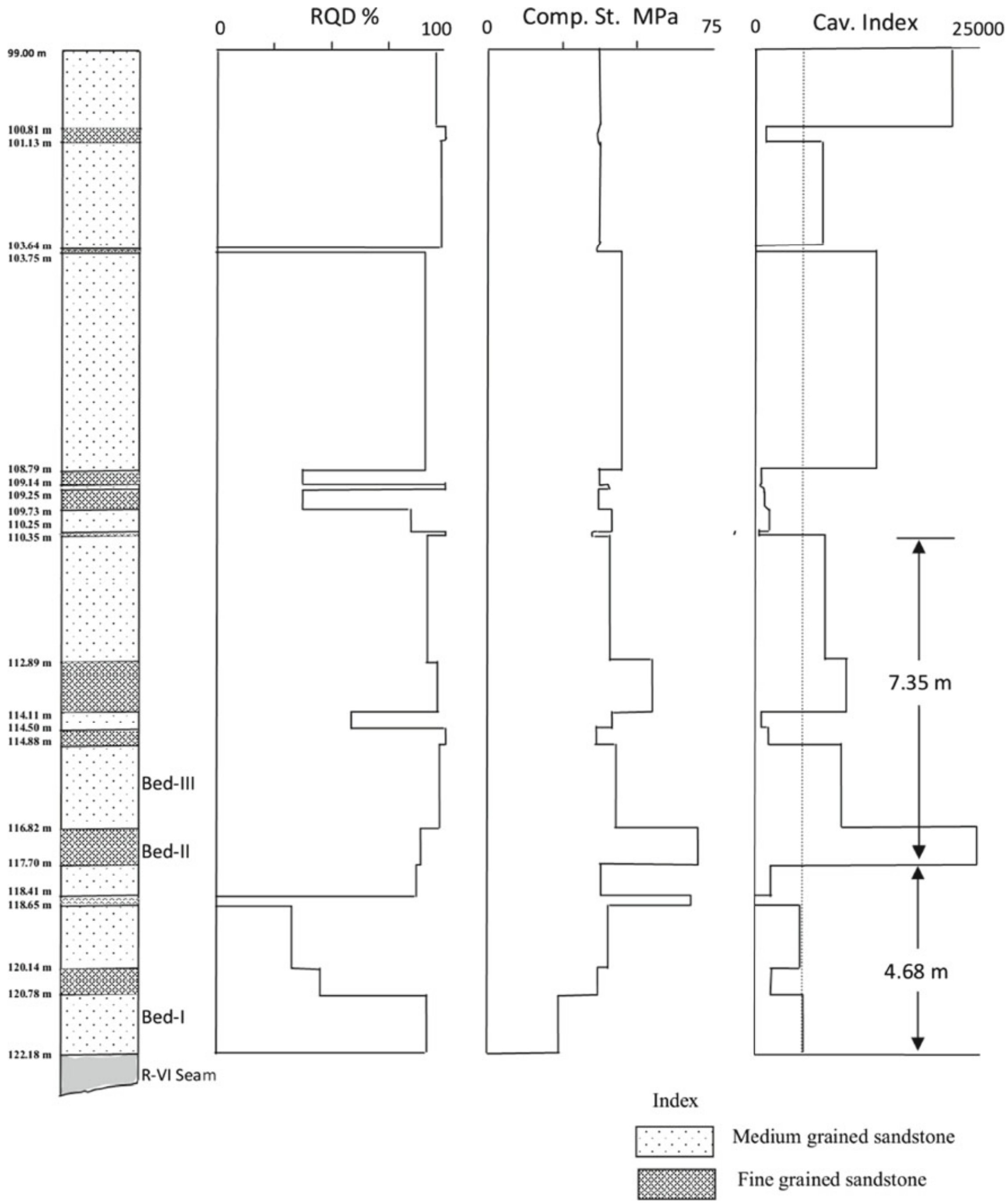

Fig. 2 Strata lithology with RQD, compressive strength & Cavabality Index [14]

overhanging roof. In the depillaring panel at Jhanjra Project with continuous miner (CM1), the three beds which had been found with higher values of Cavability Index were located within the height of 7.30 m from the immediate roof of the coal seam. Therefore, these three beds have to be fractured/broken by destressblasting to facilitate effective roof fall if the overhanging roof does not fall by its own weight after the depillaring. Initially, the induced blasting was tried out from the underground depillaring workings (Fig. 4).

The blast holes were drilled by jumbo drill machine of 42 mm diameter. The holes were drilled in fan pattern from the breaker line of next working pillar, inclined towards the goaf with 30° up from horizontal (Fig. 5). The length of hole was 15 m. The collar spacing and toe-spacing had been kept as 0.5 and 3.5 m, respectively. The blast design parameters are presented in Table 2. Each ring consisted of five holes. The packaged explosives of permitted category P-3 of 32 mm diameter were used. Low grammage (3.6 g

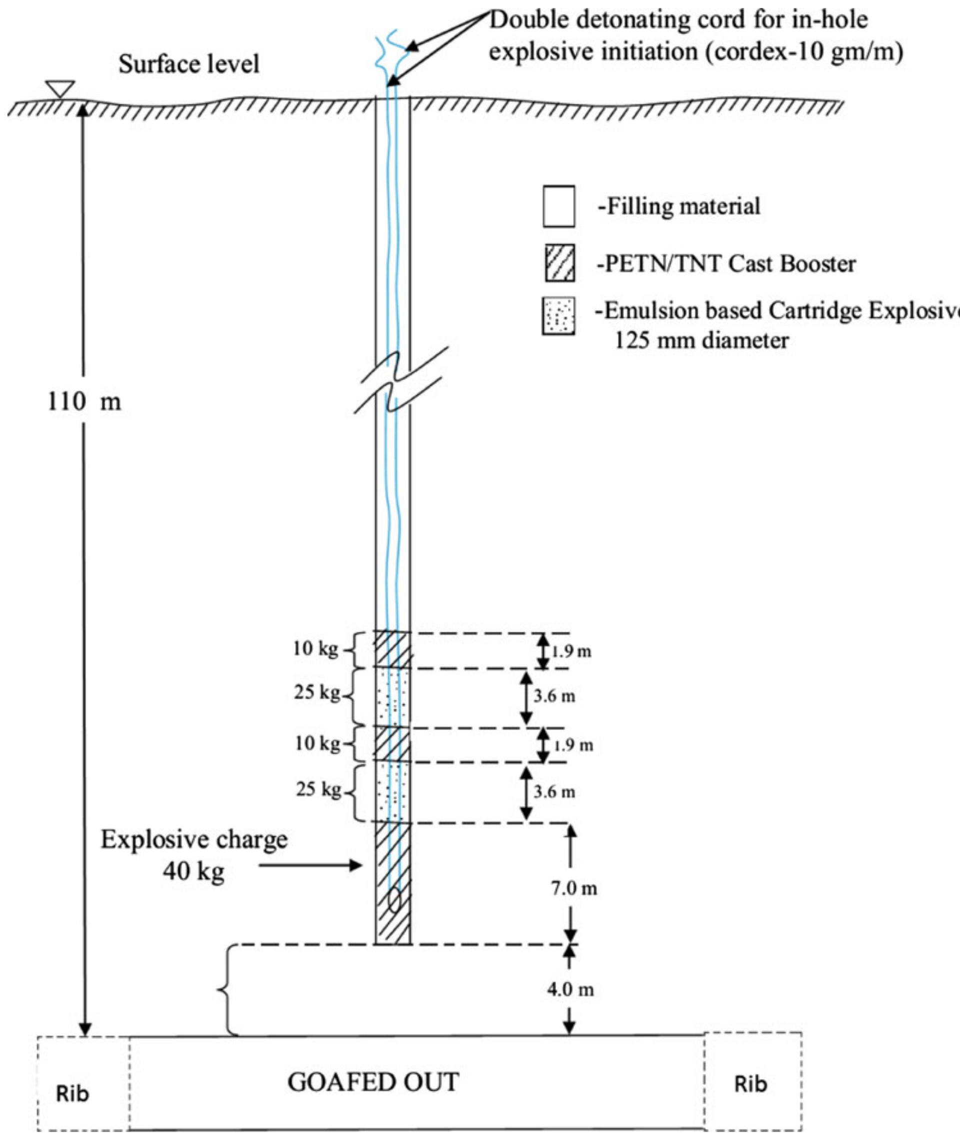

Fig. 3 Section of the blast hole showing charging pattern

of PETN per meter) detonating cord was used for initiating the explosive column inside fire resistant poly vinyl chloride (PVC) pipe used for ring hole blasting in blasting gallery method of mining. The charging pattern of holes is shown in Fig. 6. Copper coated, instantaneous electric detonator was used for each hole to initiate the detonating cord and explosive column.

Unfortunately, destress blasting from underground could not induce the caving. Once again, an attempt of induced caving from underground could not be taken up as the working had been rendered unsafe due to the previous attempt of blasting to induce caving. Therefore, it was decided to cave the roof rock from surface above the depillared panel CM1. Based on the discussions made with the DGMS officials, Sitarampur Area, Mine Management of Jhanjra Colliery, ECL, it was decided that induced blasting would be conducted from surface as the roof could not cave even after extraction of nine pillars in the panel as shown in Fig. 7. The pillars were initially splited along dip rise and then were sliced leaving rib pillars as presented in Fig. 7. The sequence of extraction is shown with pillar numbers in the figure. The extraction started away from fault and proceeded towards fault.

Explosive Type and Initiation System

Explosive type PETN/ TNT based cast booster explosive

In-hole initiation system detonating Cord (10 g of PETN/m)

Hole-to-hole initiation detonating cord with cord relays (42 ms)

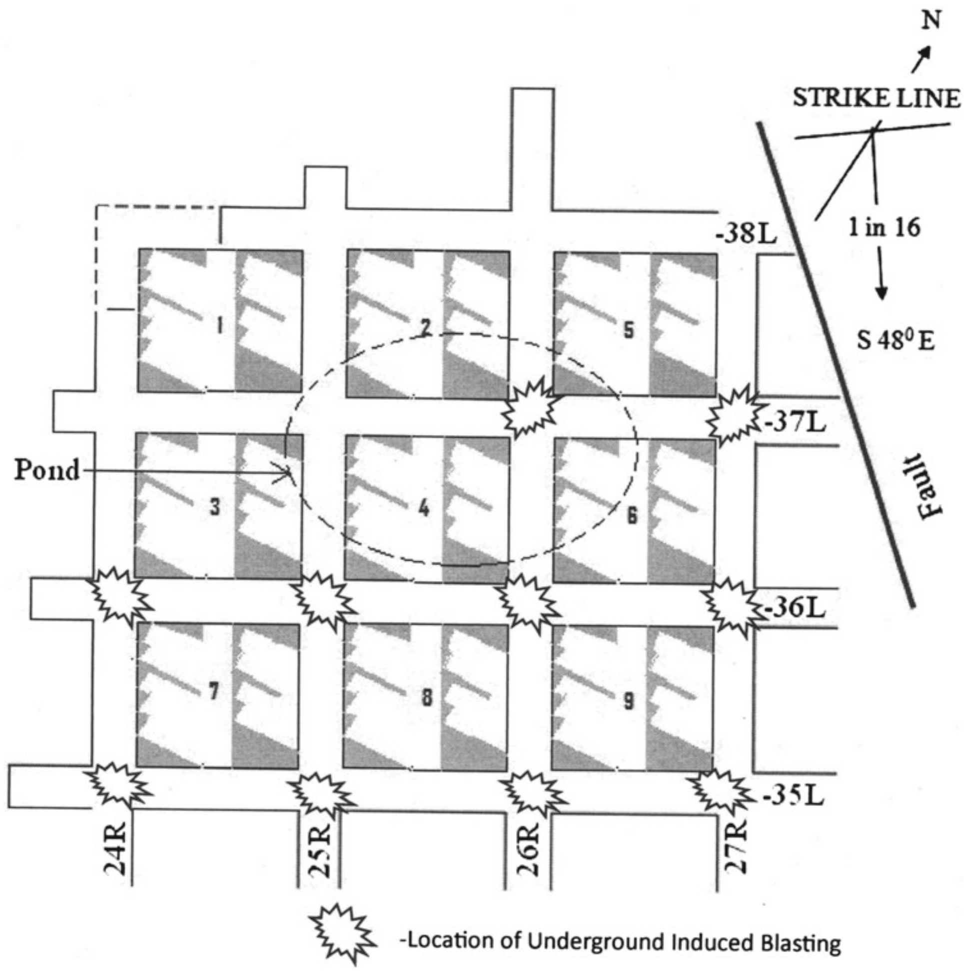

Fig. 4 Plan of panel (CM1) with location of underground induced blasting after extraction of pillars

The drilling and charging pattern of holes for induced blasting from surface at the depillaring panel in Jhanjra Project using continuous miner (CM1) are given in Table 3 and Fig. 8. Holes were drilled in line pattern as shown in Fig. 8 above the panel. The pattern followed a semi-circle as straight line drilling could not be performed due to the presence of pond on the surface.

- The villagers expressed concern on surface blasting which they thought would damage their houses.
- There had been associated risk of blown through shots in underground workings while conducting the destress blasting.
- There was no guideline to handle the misfire if it occurred at a depth of 106 m.

4 Problems Encountered During Execution of Project

The following problems were encountered during execution of the destress blasting at above mentioned site.

- There was a pond on the surface of CM1 panel, which created lot of problem in drilling blast holes on the surface.
- There was a problem of desensitization of emulsion explosive at a depth of 106 m.
- Explosive manufactures refused to provide a guarantee for detonation of emulsion explosive on account of hydrostatic pressure at a depth of 106 m.

5 Methodology Adopted to Overcome Issues

Single hole firing of cartridge emulsion explosive was tried and continuous in-hole velocity of detonation (VOD) was measured using VOD Mate instrument of Instantel Inc., Canada. The sensing cable was low resistance coaxial cable with data acquisition system of 2 MHz sampling rate. (Velocity of detonation is the rate at which the steady state detonation wave travels through an explosive column). It was found that explosive could not attain steady state VOD, may be due to desensitization of explosive at such a depth and hydrostatic pressure. Looking at the above fact, it was decided to use PETN/TNT-based explosive to induce the caving.

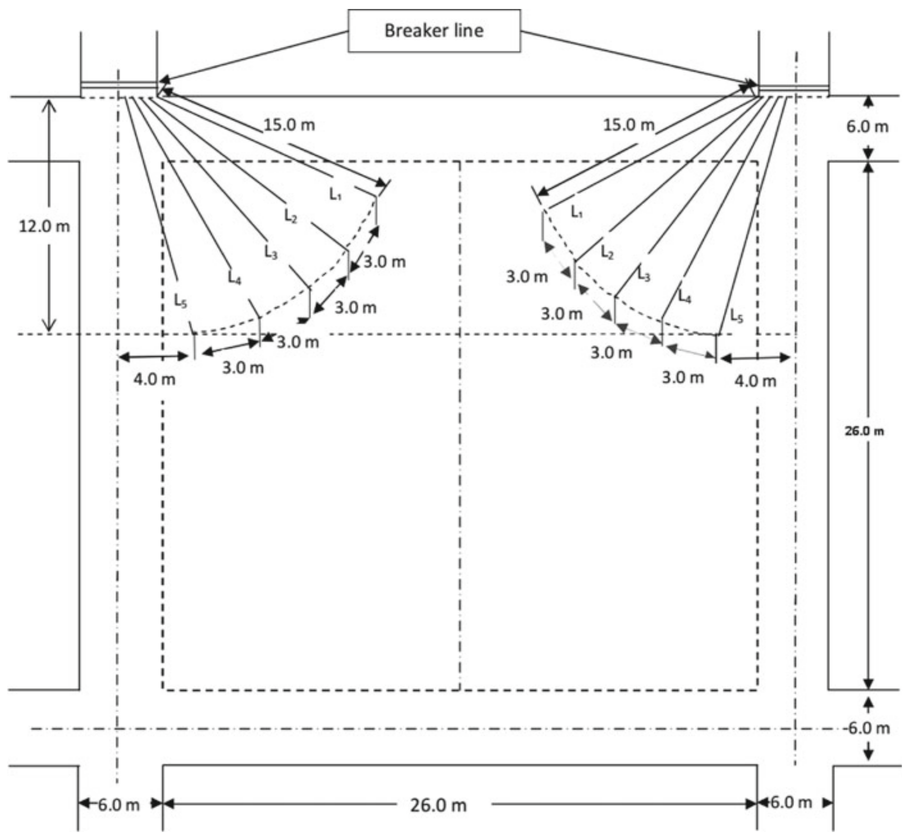

Fig. 5 Drilling pattern for induced blasting from underground

Table 2 Blast design parameters for induced blasting from underground workings

Blast design parameters	
Blast hole diameter	42 mm
Number of holes per ring	5
Depth of hole	15 m
Swing angle (measured from the vertical plane along the axis of the gallery)	38°, 48°, 58°, 68° and 78°
Burden	
Front-burden	1.35–1.75 m
Toe-burden	7.00 m
Spacing of holes	
Collar spacing	0.50 m
Toe-burden	3.00 m
Explosive charge length	8.00 and 10.00 m
Explosive charge per hole	7.40 and 9.25 kg
Total explosive charge per ring	40.70 kg

Cast boosters (Pentolite of PETN/TNT based) of 100 and 250 g were tied and a train of cartridge was formed to cause the fracturing of overlaying strata at CM1 (Fig. 9). Down the hole and surface trunk line initiation was planned with detonating cord of 10 g per meter. Down the hole delay could not be provided due to detonating cord being used as down line, only surface delays were provided.

There are three villages at a distance of 300, 315 and 335 m, respectively, from the blast site towards West, South-West and South. The houses of the villages are made of mud and brick. Therefore it was decided to keep the vibration level within 5 mm/s. The seismometers of Instantel Inc., Minimate and Minimate Plus were used with standard tri-axial transducers and microphone. Three seismometers were installed at the nearest edge of each village at 280, 290 and 305 m, respectively, from the blast site towards West, South-West and South (Fig. 10).

The theory of pre-splitting was used, but due to restriction of holes on account of blast-induced ground vibrations, only two holes at a time were initiated so that a clear fracture plane could be created for caving of the strata. A person was deployed near the exit of mine to monitor the coal dust cloud and other gasses.

All mine people had been withdrawn on surface. Villagers and their representatives were briefed about the operation and consequences along with civil administration to avoid rumors. Mine Inspector, officials of Joy Mining Company and Golder RMT (Rock Mechanics Technology Ltd.), London was also briefed about the project as they were also the stake holders in the operation of the mine.

The overlaying strata caved after 15 min of firing that shot for induced caving as the coal dust cloud could be observed at the exit of the mine. It was confirmed with a visit to underground workings after 6 h of the blast.

Fig. 6 Charging pattern for induced blasting from underground [14]

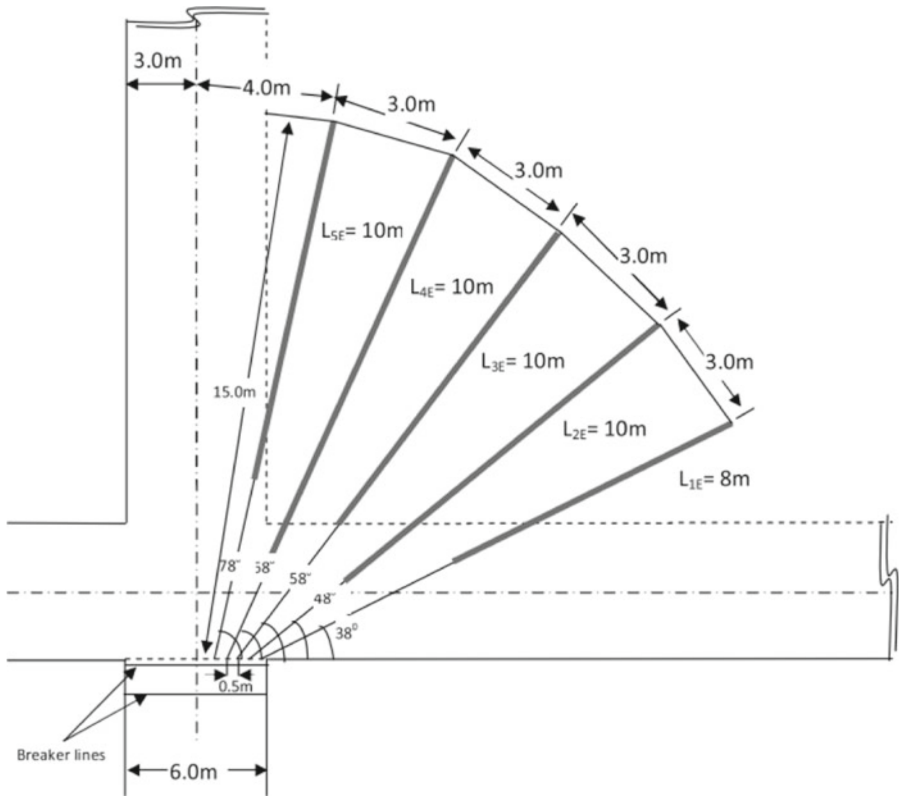

Fig. 7 Plan of panel (CM1) after extraction of pillars

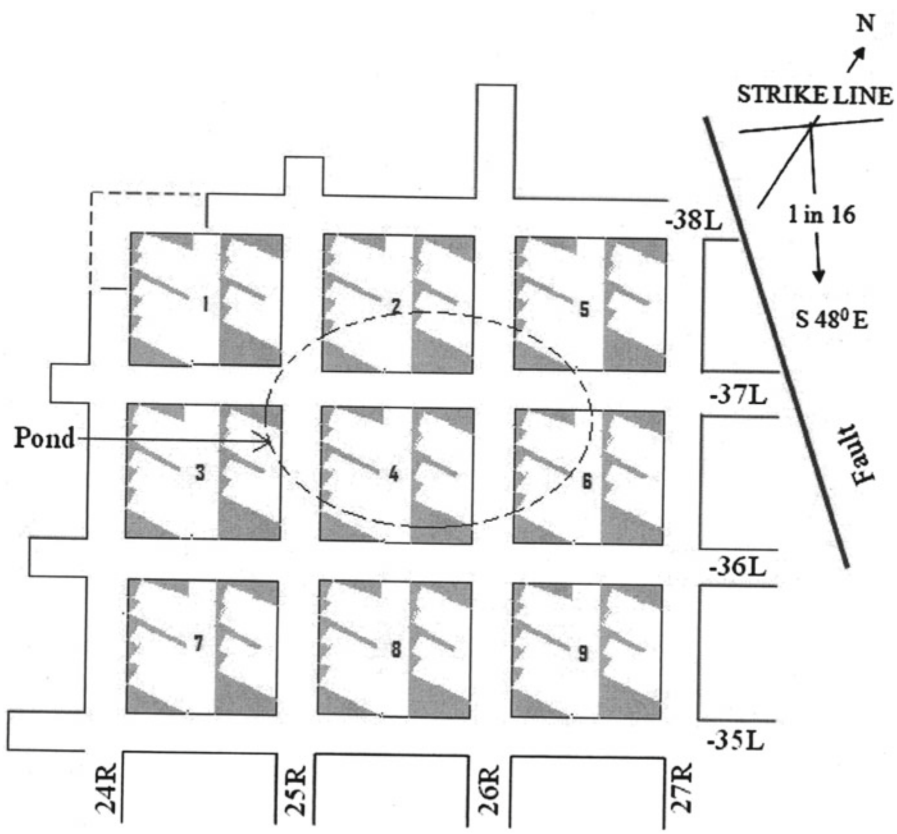

Table 3 Summary of blast design

Sl. no.	Design parameters	Values
1	Hole diameter	150 mm (6 in.)
2	Total hole depth (from surface)	110–111 m
3	Parting between hole bottom and roof of the gallery	4.0 m
4	Spacing of holes	6.5 m
5	Bottom explosive charge length	7.0 m
6	Total explosive charge length	18.0m
7	Quantity of explosive for bottom charge	40 kg
8	Total explosive charge per hole	110 kg
9	Total no. of holes	16
10	Total explosive charge	1,760 kg
11	Maximum charge per delay	220 kg

Fig. 9 Preparation of explosive before charging

6 Conclusion

Three rock beds had been identified in the immediate roof of the proposed depillaring panel (CM1) with Cavability Index more than 5,000. The delay in natural roof fall might have been due to the presence of these hard rock strata within the immediate roof of R-VI Seam. Problem on natural roof fall has also been anticipated due to increased size and number of rib pillars left during each pillar extraction.

The technical know-how and blast design for destress blasting to induce caving over first ever panel (CM1) at a depth of 110 m from surface has been successful from the surface after unsuccessful attempts from underground. The situation was more critical due to the increased awareness of the nearby villagers and their irrelevant worries. Recorded blast-induced ground vibration was less than 2 mm/s at the nearest village boundary. The size of rib pillars may be reduced looking at the cavability problem of the mine.

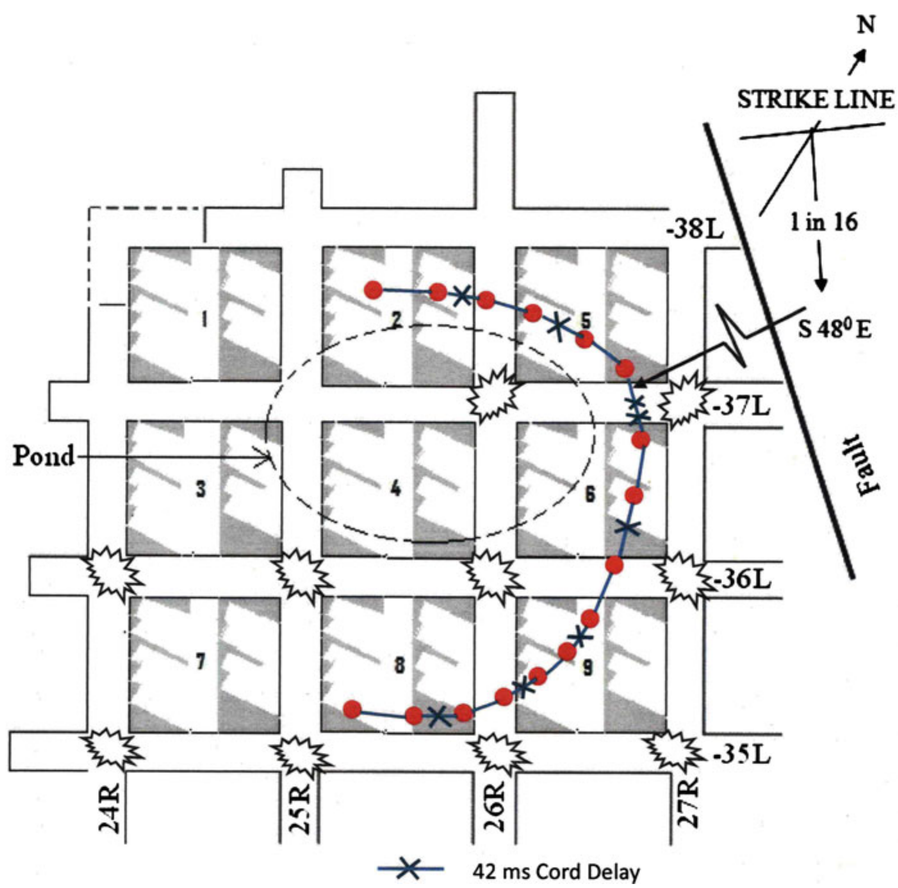

Fig. 8 Plan showing the drilling pattern and firing sequence above panel CM1 OFDMA uplink system

Fig. 10 Location of seismometer near village

Although surface blasting to induce caving has been difficult and troublesome but a successful implementation could restart the mine and the revenue flow, thereby giving relief to the management.

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