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Presplit blasting for stabilization of highwall slopes and productivity enhancement in a trench-mining project

Stabilization of highwall slope by implementing presplit blasting and productivity enhancement by improving the powder factor using controlled blasting at Sharda project of South Eastern Coalfields Ltd. (SECL) was the endeavour of the research team of CSIR-CIMFR in the first trench highwall mining of India. Good fragmentation was achieved through improved rock-explosive interactions and redefined design parameters in the trench which had restricted width and lesser initial powder factor (in m^3/kg) due to varying rock-geologic and rock-explosive characteristics.

Through systemic analysis of blast-results and implementation of scientific theories combined with the usage of rock parameters and its physico-mechanical properties, it could possible to achieve around 2.0 m^3/kg powder factor instead of 1.3-1.4 m^3/kg maintaining stable highwall benches with 70^0or more slope angle. The work led to huge cost benefits in explosive consumption for the company i.e. M/s Cuprum Bagrodia Ltd. and also markedly reduced the environmental implications from production blastsduring day-to-day operations.

Keywords: *presplit blasting;trench highwall mining;slope angle; controlled blasting*

2.0 Introduction

At Sharda Highwall Mining Project of Sohagpur Area, SECL, thin coal seams were extracted using a "Trench Highwall Mining" method. It was necessitated to optimise the trench dimensions to minimise the land degradation and volume of trench excavation and at the same time maintaining safety during the entire highwall mining operations. A parametric study on slope stability had been done for finding the effect of slope angle and ground water conditions on trench slope stability[1]. Using numerical modelling, a parametric study on slope stability had been conducted for sequential multiple seam trenching and highwall extractions and it was obtained that the slope angle of 70° was safe from stability standpoint (Fig.1).

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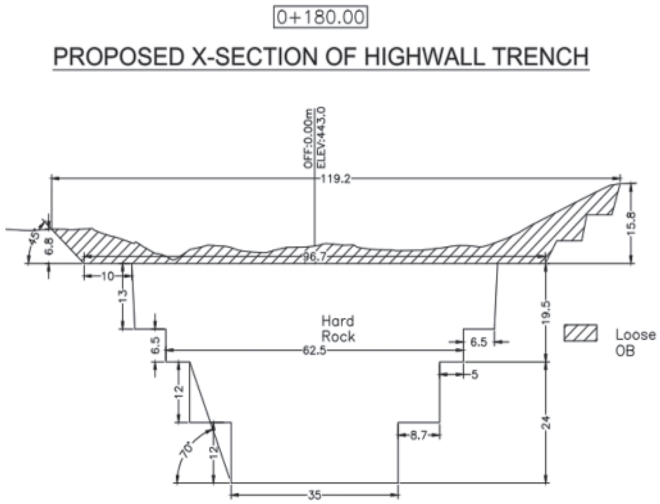

Fig.1 Cross-section of highwall trench at Sharda OCP with 70^0 slope angle

Consequent upon slope-stability study, the outsourcing company viz. Cuprum Bagrodia Limited awarded another scientific study to CSIR-CIMFR for improvement of blasting efficiency and presplit blast design for the formation of highwall benches [2]. After studying the nature and type of rock strata present at Trench No.3 (T-3), smooth blasting technique was implemented for improvement of powder factor. Rock samples were also collected from -6 m to -12 m bench of T-3 area for determination of physico-mechanical properties. Based on the nature of rock deposits and their physico-mechanical properties, blast design patterns for presplit blasting were evolved and tried in-field conditions for stable and smooth final wall.

3.0 Type and nature of rock deposits

The type of rocks present at T-3 area mainly consisted of sandstone with massive formation. They were classified as:

(1) Very coarse-grained sandstone (VCgsst)

(2) Coarse-grained sandstone (Cgsst)

(3) Medium-grained sandstone (Mgsst)

(4) Intercalation of shale and sandstone (I/C-Shale-sst)

(5) Shale and coal

Experiments conducted in Trench-3 (T-3) area consisted of hard rock wherein 0 to -12 m were only exposed. The first bench (i.e. 0 to -6 m) comprised shaley sandstone and massive formation of sandstone. The shaley sandstone was present in the top portion of the bench and the thickness varied from 1.5 to 2.4 m. Apart from the lamination (bedding plane), two major joint sets were observed in the top i.e. shaley-sandstone portion. One joint plane was found nearly perpendicular to the highwall slope (Fig.2) and the other joint set was nearly parallel to the highwall slope (Fig.3).

The lower portion of about 3.6 to 4.5 m consisted of white coloured, coarse to medium-grained sandstone. No prominent joint planes were observed except the lamination plane. The optimum slope angle of the highwall in hard rock was taken as 70^{o} or more from the horizon (Fig.1). It was to ensure that the highwall should be made stable during extraction of all the seams (top downwards) under the site-specific geomining conditions.

The second bench (i.e. -6 to -12 m) consisted of hard and massive coarse to medium-grained white coloured sandstone

Fig.2 Joint planes perpendicular to highwall slope in T-3 (Bench: 0 to –6 m)

Fig.3 Joint planes parallel to highwall slope in T-3 (Bench: 0 to -6 m)

(Fig.4). No prominent joints were visible in the exposed highwall. Rebound hardness values measured on the massive sandstone varied from 22 to 32. It was observed that hard and massive formation of white coloured, coarse to medium-grained sandstone constituted the major portion of the strata (Fig.5). However, shale and shaley-sandstone were found immediately above and bottom of the coal seams which occasionally created problem during presplit blasting.

Fig.4 Massive sandstone formation in 0 to –12 m in T-3

Fig.5 View of different rock strata along the Highwall in T-1

2.0 Physico-mechanical properties of rocks

The rock mass properties of T-3 area were determined at the Rock Testing Laboratoryof CSIR-CIMFR. The minimum, maximum and the average values of uniaxial compressive strength, tensile strength, Young's modulus, Poisson's ratio, shear modulus, bulk modulus, apparent cohesion and angle of internal friction for different rocks are given in Table 1. Primary wave velocity (P-wave) of medium-grained sandstone varied between 2615 and 2564 m/s and the shear wave velocity (S-wave)varied between 1442 and 1453 m/s. The average density of medium-grained sandstone was 2.19 g/cc.

3.0 Presplit blast design considerations

There are many theories on the mechanism of presplit blasting. In this technique, a fracture plane is created in the

TABLE 1: PHYSICO-MECHANICAL PROPERTIES OF DIFFERENT ROCK TYPES AT SHARDA PROJECT

Type of Rock	Properties	Minimum value	Maximum value	Average/value
Very coarse-grained sandstone	Compressive strength (MPa)	2.02	18.59	9.02
	Tensile strength (MPa)	0.36	2.80	1.19
	Young's modulus (GPa)	0.58	5.10	2.92
	Poisson's ratio	0.11	0.30	0.24
	Shear modulus (GPa)	0.29	2.30	1.18
	Bulk modulus (GPa)	0.19	4.17	1.82
	Apparent cohesion (MPa)	0.82	3.90	2.63
	Angle of internal friction (°)	42.49	57.37	48.72
Coarse grained sandstone	Compressive strength (MPa)	4.00	29.16	12.92
	Tensile strength (MPa)	0.70	3.37	1.52
	Young's modulus (GPa)	0.65	8.70	3.43
	Poisson's ratio	0.05	0.29	0.19
	Shear modulus (GPa)	0.29	3.51	1.48
	Bulk modulus (GPa)	0.30	5.58	1.97
	Apparent cohesion (MPa)	1.35	6.37	3.52
	Angle of internal friction (°)	38.55	56.00	46.63
Medium grained sandstone	Compressive strength (MPa)	7.64	35.61	15.77
	Tensile strength (MPa)	0.84	3.99	2.23
	Young's modulus (GPa)	1.18	4.98	3.34
	Poisson's ratio	0.04	0.28	0.15
	Shear modulus (GPa)	0.53	2.39	1.44
	Bulk modulus (GPa)	0.52	3.01	1.63
	Apparent cohesion (MPa)	2.33	5.94	3.94
	Angle of internal friction (°)	30.04	58.14	47.23
Intercalation of shale and sandstone	Compressive strength (MPa)	11.39	44.68	21.78
	Tensile strength (MPa)	1.37	5.83	3.65
	Young's modulus (GPa)	1.33	5.88	3.47
	Poisson's ratio	0.02	0.31	0.15
	Shear modulus (GPa)	0.54	2.67	1.52
	Bulk modulus (GPa)	0.69	3.67	1.78
	Apparent cohesion (MPa)	3.61	6.55	4.58
	Angle of internal friction (°)	39.50	55.82	47.54
Shale	Compressive strength (MPa)	16.02	45.34	24.16
	Tensile strength (MPa)	0.90	5.50	3.53
	Young's modulus (GPa)	2.09	5.14	3.82
	Poisson's ratio	0.06	0.25	0.17
	Shear modulus (GPa)	0.98	2.12	1.63
	Bulk modulus (GPa)	0.81	3.06	2.09
	Apparent cohesion (MPa)	-	-	-
	Angle of internal friction (°)	-	-	-
Coal	Compressive strength (MPa)	17.75	43.43	31.01
	Tensile strength (MPa)	0.36	4.58	2.27
	Young's modulus (GPa)	1.52	2.06	1.75
	Poisson's ratio	0.04	0.18	0.12
	Shear modulus (GPa)	0.66	0.87	0.78
	Bulk modulus (GPa)	0.58	1.07	0.78
	Apparent cohesion (MPa)	-	-	-
	Angle of internal friction (°)	-	-	-

rock mass before firing the production blast by means of a row of blastholes with decoupled explosive charges. During detonation of such decoupled charges, a rapid decomposition takes place and the explosive releases tremendous amount of heat and gas. The stable end products are gases that are compressed, under elevated temperature, to very high pressures. The sudden rise in temperature and pressure from ambient conditions in all decoupled charged-holes that results in shock waves which collide during propagation between the boreholes places the web in tension and causes cracking that produce a shear zone between the blastholes as shown in Fig.6.

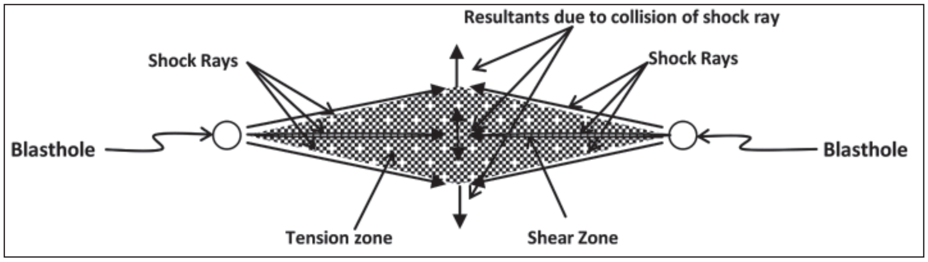

Fig.6 Pre-spitting theory illustration [4]

Once the cracking between boreholes takes place, the expanding gases of the explosive is subsequently vented and widened by the expanding gases based on the three factors viz. (1) properties and conditions of the rock; (2) spacing between blastholes; and (3) amount and type of explosive in the holes. This split or crack in the rock forms a discontinuous zone which minimizes overbreak from the subsequent primary blast and produces a smooth finish rock wall [3].

When rock has numerous joints between blastholes and those joints intersect the blasting face at less than 15° angle, it will be impossible to form a good and smooth face even with controlled blasting techniques [5]. In heavily jointed rock mass, presplitting may not yield good result whereas smooth blasting may provide better result.

Blasthole diameter for presplit blasting mainly depends on the availability of drilling machines at the project sites. The blasthole diameter used by different persons for presplit blasting varied widely from 30 to 250 mm. However, blasthole diameter ranging between 51 and 115 mm are commonly used for presplit blasting in underground and surface excavations. ISEE [4] recommended 51 to 89 mm whereas Olofson et al. [6] recommended a hole diameter of 30 to 64 mm. Jimeno et al. [7] recommended hole diameter ranging from 35 to 75 mm. Hagan and Mercer [8] recommended hole diameter ranging from 75 to 250 mm.

Spacing between presplit holes is generally determined based on the dynamic tensile strength of the rock and borehole pressure of the explosive generated by decoupled charge. Spacing between presplit holes as given by Calder and Jackson [9] is:

$$S \le \frac{D_h \times (\sigma_t + P_{de})}{\sigma_t} \quad \text{... (1)}$$

Where,

S = Hole spacing (m)

D_h = Hole diameter (m)

σ_t = Tensile strength of the rock (MPa)

P_{de} = De-coupled borehole pressure of the explosive charge

Borehole pressures generated by fully-coupled and de-coupled explosive charge as given by Jimeno et al. [7] are:

$$P_{bh} = 228 \times 10^{-6} \times \rho_e \times \frac{VOD^2}{(1+0.8\rho_e)}$$

(Fully coupled charge) ...2

$$P_{de} = P_{bh} \times \left[\frac{V_e}{V_b}\right]^{1.2} = P_{bh} \times \left[\frac{D_e}{D_h}\sqrt{C_l}\right]^{2.4}$$

(De-coupled charge) ...3

Where,

P_{bh} = Borehole pressure of fully coupled charge (MPa)

P_{de} = Borehole pressure of de-coupled charge (MPa)

V_e and V_b=Volumes of explosive and blasthole respectively

ρ_e = Density of explosives (g/cc)

VOD = Detonation velocity of explosive (m/s)

D_e and D_h = Diameters of explosive and blasthole respectively

C_l = Percentage of explosive column that is loaded

Chiappetta [10] used the following equation (4) for determination of borehole pressure generated by de-coupled explosive charge:

$$P_b = 1.25 \times 10^{-4} \times \rho \times (VOD)^2 \left[\frac{r_e}{r_h}\right]^{2.6} \quad \text{... 4}$$

Where,

P_b= Borehole pressure of de-coupled charge (MPa)

ρ_e = Density of explosives (g/cc)

VOD = Detonation velocity of explosive (m/s)

r_e and r_h = Radius of explosive and blasthole respectively

The explosive charge concentration required for presplit blasting is generally determined based on the blasthole diameter. The empirical equations developed by different researchers [7, 11] for the linear charge concentrations are:

$Q_l = 90 \times d^2$ ... 5

$Q_l = 8.5 \times 10^{-5} \times D^2$ ... 6

Where,

Q_l = linear charge concentration (kg/m)

d = blasthole diameter (m)

D = blasthole diameter (mm)

The blast design parameters recommended by different researchers for presplit blasting are given in Table 2.

TABLE 2: BLAST DESIGN PARAMETERS FOR PRESPLIT GIVEN BY DIFFERENT PERSONS

	Blasthole diameter (mm)	Spacing (m)	Linear charge concentration
Blaster Handbook [4]	102	0.6-1.2	0.38-1.12
Hagan & Mercer [8]	115	1.2	1.10
Person et al. [11]	80	0.6-0.8	0.57
Gustafsson [12]	64	0.6-0.8	0.46

4.0 Proposed blast design patterns of presplit blasting

Based on the geo-mechanical properties of rocks and nature of deposit, the proposed blast design patterns for presplit blasting at T-3 area of Sharda highwall mining project were made. The blasthole diameter used for production blast at the project site was 160 mm diameter. However, smaller blasthole diameter was preferred for presplit blasting. Hence blast design patterns for both 115 mm and 160 mm diameter were proposed for the experimental blasts. The drill machine was chosen so as to drill with lesser hole deviation up to the height of at least 14 to 15 m. For better positioning of the drilling rig, crawler mounted drill machine was recommended.

4.1 PRESPLIT BLASTING PATTERNS WITH 115 MM BLASTHOLE DIAMETER

For presplit blasting with 115 mm blasthole diameter, cartridged explosive of 25 mm diameter, 125 g weight having detonation velocity less than 4000 m/s was recommended. As an alternate, due to easy-accessibility, permitted explosives of 32 mm diameter was also suggested for carrying out presplit blasting.

Using equation (4) of Chiappetta [10], the borehole pressures to be exerted on the walls of 115 mm blasthole diameter while detonating 25 mm (dia.) non-permitted explosive, 32 mm (dia.) permitted (P-5 type) emulsion explosive and 32 mm (dia) slurry explosive are shown in Table 3.

Based on the study of borehole No. CMSBK, coarse to very coarse-grained sandstone of white colour constituted majority of the rock mass. The uniaxial compressive strength of coarse grained sandstone varied from 4.0 to 29.16 MPa and very coarse-grained sandstone varied from 2.02 to 18.59 MPa. The mean values of compressive strength for coarse-grained and very coarse-grained sandstones became 9.02 MPa and 12.92 MPa respectively. Considering the dynamic in-situ compressive strength of rock as two times its static strength, dynamic compressive strength of coarse-grained sandstone varied from 8 to 58 MPa with mean value of 26 MPa. Similarly, the dynamic compressive strength of very coarse grained sandstone varied from 4 to 37 MPa with a mean value of 18 MPa.

The borehole pressures produced by emulsion and slurry based permitted explosives (P-5 type) of 32 mm diameter in 115 mm blasthole diameter were 69.61 MPa and 60.64 MPa respectively. However, these values were much higher than the dynamic compressive strength of rocks. The borehole pressure generated on 115 mm blasthole diameter by 25 mm diameter of non-permitted explosive was 42.92 MPa. That value was less than the borehole pressure produced by 32 mm permitted explosives. Hence, 25 mm diameter was preferred for presplit blasting. However, the mean dynamic compressive strengths of coarse-grained sandstone and very coarse-grained sandstone are still less than the borehole pressure. Hence, linear charge concentration was reduced in order to prevent excessive crushing.

For determination of spacing between presplit holes, the mean dynamic tensile strength of coarse-grained sandstone was taken. The mean value of tensile strength was 1.52 MPa. The dynamic tensile strength was considered as 4.56 MPa. The spacing of holes for presplit line was calculated as:

$$S \leq \frac{D_h \times (\sigma_t + \mathrm{P}_{de})}{\sigma_t} = \frac{115 \times (4.56 + 42.92)}{4.56} = 1.2\mathrm{m} \qquad \text{... 7}$$

Therefore, the spacing between presplit holes became1.2 m. The linear charge concentration based on the empirical equations given by Jimeno et al. [7] and Person et al. [11] for 115 mm blasthole diameter were calculated as:

$$Q_l = 90 \times 0.115^2 = 1.2 \text{ kg/m} \qquad \text{... 8}$$

$$Q_l = 8.5 \times 10^{-5} \times 115^2 = 1.10 \text{ kg/m} \qquad \text{... 9}$$

Based on the above calculations, the linear charge concentration for 115 mm blasthole diameter became 1.1 to 1.2 kg/m. However, the borehole pressure generated by continuous explosive charge of 25 mm diameter was higher than the mean dynamic compressive strength of rocks. Therefore, linear charge concentration of 0.85 to 1.0 kg/m wasused during trial blasts. For charging of explosives in presplit holes, detonating fuse of core charge (PETN) 10 g/m of length was used. The proposed blast design pattern for 115 mm blasthole diameter is given in Fig.7. Based on the

TABLE 3 BOREHOLE PRESSURE PRODUCED BY DIFFERENT EXPLOSIVE TYPES IN 115 MM BLASTHOLE DIAMETER

Explosive type	Diameter (mm)	Density (g/cc)	VOD (m/s)	Borehole Pressure (MPa)
Non-Permitted	25	1.1	3800 – 4000	42.92
P-5 (Emulsion)	32	1.1	3500 – 4000	69.61
P-5 (Slurry)	32	1.1	3400 – 3800	60.64

requirement, blasthole depth varied between 13 and 14 m.

4.2 Presplit Blasting Patterns with 160 mm Blasthole Diameter

The borehole pressure generated by different types of explosives on 160 mm blasthole diameter is given in Table 4. Due to enhanced decoupling ratio, the borehole pressure generated in 160 mm blasthole diameter is lowered than that of 115 mm blasthole diameter. Therefore, 25 and 32 mm diameter of non-permitted explosives or 32 mm diameter of permitted explosive (P-5) was applicable for 160 mm blasthole diameter.

Considering dynamic tensile strength of rock as 4.56 MPa, the spacing of presplit holes for 160 mm blasthole diameter calculated as is:

$$S \le \frac{D_h \times (\sigma_t + P_{de})}{\sigma_t} = \frac{160 \times (4.56 + 17.63)}{4.56} = 0.78\text{m} \approx 0.8\text{m} \quad \text{... 10}$$

(for 25 mm diameter non-permitted explosive)

$$S \le \frac{D_h \times (\sigma_t + P_{de})}{\sigma_t} = \frac{160 \times (4.56 + 25.65)}{4.56} = 1.06\text{m} \approx 1.1\text{m} \quad \text{... 11}$$

(for 32 mm diameter slurry P-5 type explosive)

The required linear charge concentration based on empirical equations prescribed by Jimeno et al. [7] and Person et al. [11] for 160 mm blasthole diameter calculated as-

$$Q_l = 90 \times 0.160^2 = 2.3 \text{ kg/m} \quad \text{... 12}$$

$$Q_l = 8.5 \times 10^{-5} \times 160^2 = 2.18 \text{ kg/m} \quad \text{... 13}$$

The required linear charge concentration is higher in case of 160 mm than that of 115 mm blasthole diameter. Therefore, 32 mm diameter cartridged explosives was preferred for charging of holes in presplit blasting using 160 mm blasthole diameter. The design pattern of presplit blasting using 160 mm blasthole diameter is depicted in Fig.8. The view of smooth and final wall obtained in 0 to -6 m bench of T-3 area is depicted in Fig.9.

5.0 Production blasts

The blasthole diameter for the production blast was 160 mm and holes were drilled with truck-mounted hydraulic drill. Depth of holes was maintained between 6.0 and 6.2 m. Burden and spacing were 3.0 m each and the front burden was kept as 2.0 m. In general, three rows were drilled in squared-pattern and the total number of holes in a blasting round varied between 45 and 60. Cartridged explosives of 125 mm diameter and 6.25 kg weight were used and all the holes were charged with Nonel system of initiation containing 300 ms DTH. For surface hole-to-hole initiation in a row, 17 ms TLD was used for the first two rows. However, for the last row, 42 ms TLD was used. For row-to-row initiation, 42 ms DTH was used. The average explosive charge per hole was 37.50 kg and the powder factor varied between 1.4-1.5 m^3/kg. Deck charge was used for the last two holes. The top stemming column without decked charge varied from 3.2 to 3.4 m. Depending upon the number of holes in a row, the total number of holes fired within 8 ms time-frame varied from 3 to 4. The primer and column charge ratio of 1:4 was generally maintained.

5.1 Experimental blasts

Experimental blasts were conducted in 0 to – 6 m bench using increased burden and spacing values in order to enhance powder factor (measured in m^3/kg). Drilled in squared pattern, the burden and spacing were 3.5 m and 4.5 m respectively. As a case example, as shown in Fig.9, the satellite holes (or pilot holes) were drilled in-between the production holes. The depth of production holes varied between 5.8 and 6.2 m whereas the depths of pilot holes were maintained as 1.6 to 1.9 m. The total number of holes including adjustment holes for the front burden was 55 (45 main and 10 adjustment holes) and the total number of pilot holes was 27.

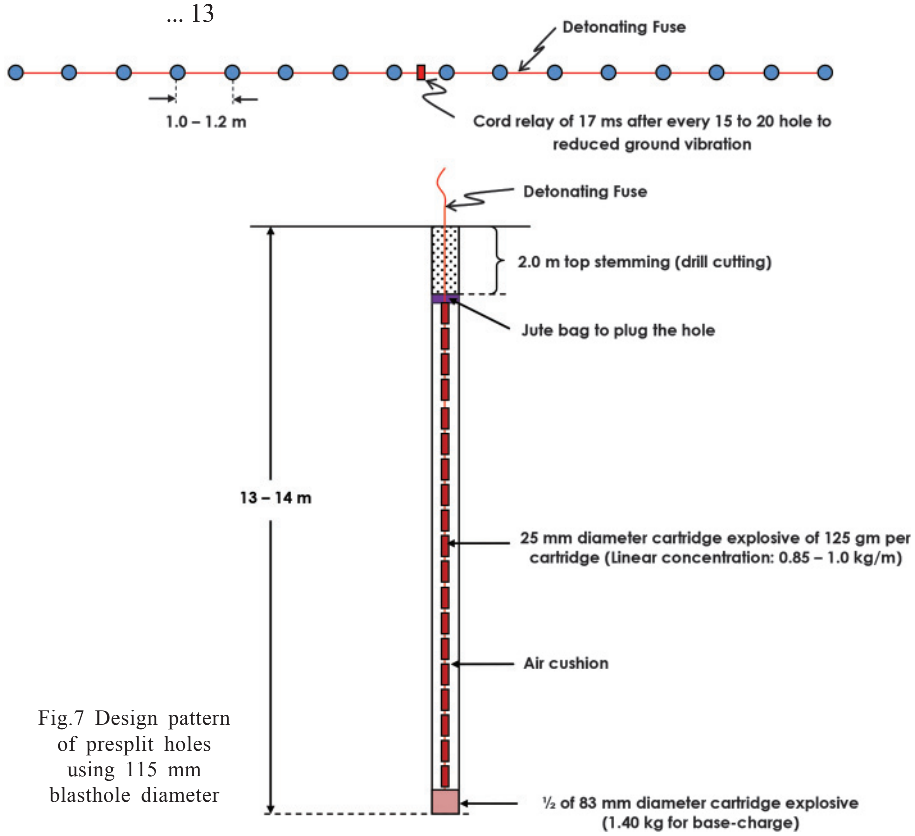

Fig.7 Design pattern of presplit holes using 115 mm blasthole diameter

Table 4: Borehole pressure produced by different explosive types in 160 mm blasthole diameter

Explosive type	Diameter (mm)	Density (g/cc)	VOD (m/s)	Borehole Pressure (MPa)
Non-Permitted	25	1.1	3800 – 4000	17.63
P-5 (Emulsion)	32	1.1	3500 – 4000	29.44
P-5 (Slurry)	32	1.1	3400 – 3800	25.65

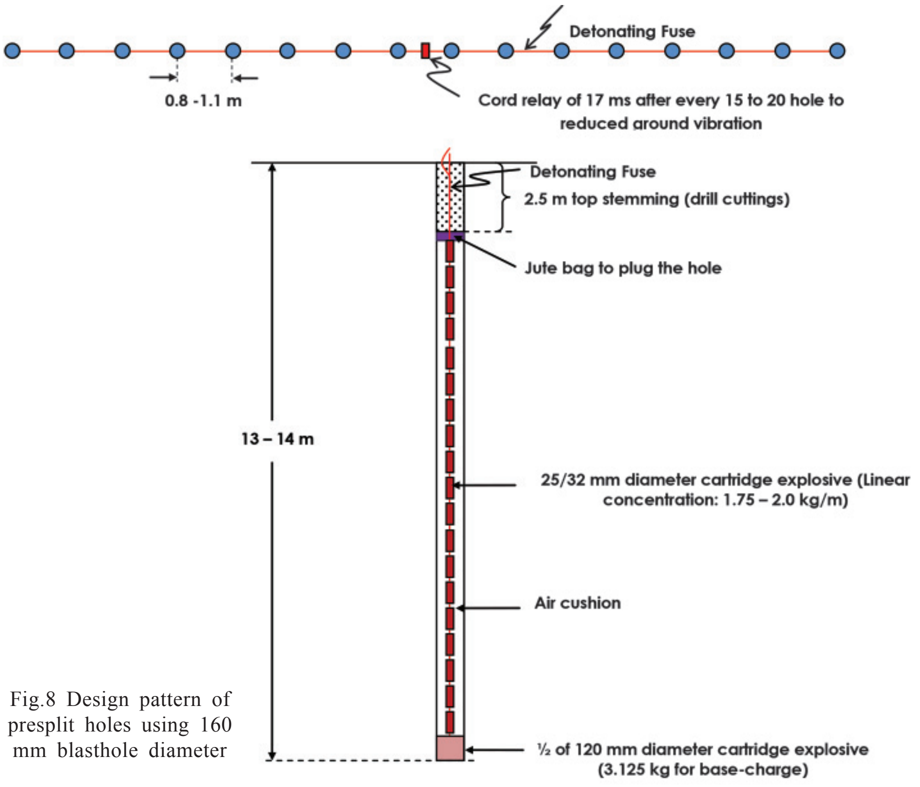

Fig.8 Design pattern of presplit holes using 160 mm blasthole diameter

All the main holes were charged with 50.00 kg of explosive each and satellite holes were charged with 3.125 kg of explosive. In order to minimize the overbreak from the last row of holes, decked charge was used in all the holes in the last row using detonating cord. The pilot holes were initiated with the nearest hole of the main holes as shown in Fig.9. The calculated powder factor including pilot holes was 1.78 m^3/kg and the overall powder factor was 1.8 m^3/kg (Since the number of pilot holes was only 27).

5.2 Analysis of blast results

Good fragmentation was obtained with increased burden and spacing using pilot holes. The overall powder factor was enhanced to 1.8 m^3/kg. In 0 to-6 m bench, nearly 2.5 m thick shaley sandstone was present at the top of the bench. Therefore, good fragmentation was obtained from the stemming portions as well as in the bottom portion of the holes where massive and hard rock was predominant. On recurring trials, it was possible to achieve the improved powder factor (m^3/kg) while maintaining the desired fragmentation so as to make reasonable profit to the outsourcing company.

6.0 Conclusions and recommendations

Considering the optimum slope angle of the highwall in hard rock as 70^0 (or more) from the horizon, the design patterns for presplit blasting were recommended for both 115 and 160 mm blasthole diameters. The parameters which were primarily considered during establishing such design patterns included

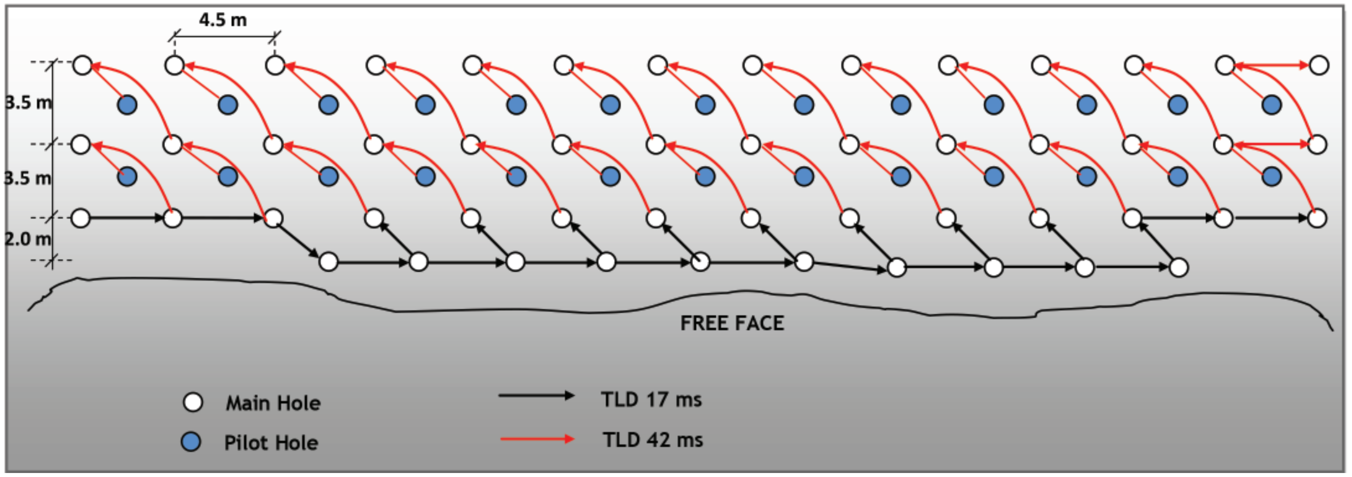

Fig.9 Design pattern of main and pilot/satellite holes for achieving higher powder factor

(i) Physico-mechanical properties of the rock; (ii) Spacing of presplit holes; (ii) Linear charge concentration of explosives; (iii) Quantity of base charge; (iv) Top stemming length; (v) Distance between buffer holes and presplit holes; (vi) Ground vibration and noise generated from presplit blasts. The aim was to achieve effective presplitting for smooth and stable final wall so that during extraction of coal through trench mining, no failure occurs. While evolving the appropriate design parameters for presplit blasting, one trim blasting with 160 mm hole diameter was carried out in the hard rock(in bench: 0 to -6 m) during dressing of the final wall. In that blasting, spacing varied between 0.75 to 1.0 m and holes were charged with detonating fuse using distributed cartridged explosives of 125 mm diameter. Results of that blast helped in establishing the drilling and charging patterns for presplit blasting during regular operations.

Improvement of powder factor from 1.4 - 1.5 m^3/kg to about 1.8 m^3/kg was achieved in normal hard-rock blasting by opting inflated burden and spacing added with pilot holes as shown in Fig.9. The fragmentation with such enhanced powder factor was good and economically beneficial to the company. As the parting between 6 seam top and bottom varied from 18 to 20 m and the parting between 6 seam bottom and 4 seam varied from 26 to 28 m, it was relatable to propose longer blastholes up to 8.0 m, for speedy excavation. The effect of such increased blastholes vis-a-vis charge per hole and total charge on ground vibration was a matter of concern and therefore it was advised to the management to carry out further study to ascertain the impacts of ground vibration and consequent remedial measures.

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