-RESEARCH PAPER-

International Journal of Mining and Geo-Engineering

IJMGE

IJMGE 57-2 (2023) 195-203

DOI: 10.22059/IJMGE.2023.346294.594986

# Optimization of powder factor, fragmentation and oversized boulders through subsystem studies in an opencast coal mine

Pijush Pal Roy [a, *], Chhangte Sawmliana [b] and Rakesh Kumar Singh [b]

[a] *CSIR-Central Institute of Mining & Fuel Research, Dhanbad, India.*

[b] *Rock Excavation Engineering Department, CSIR-Central Institute of Mining and Fuel Research, Dhanbad, India.*

Article History:

Received: 25 July 2022.

Revised: 31 December 2022.

Accepted: 24 February 2023.

## ABSTRACT

The massive deposit of medium-grained, white-colored sandstone of about 20 m thick, is located immediately above the coal seam in Quarry No. 2, resulted lesser yield due to lower powder factor ($m^3$/kg) and over-sized boulder formations, specifically from the stemming zones at Chotia Opencast Coal Mine of M/s Prakash Industries Limited, which was operating at a depth of about 30 to 40 m with an average bench height of 5.5 m. The criticality of the problem led to the rectification of the blast design parameters through incorporation of pilot holes and pocket charges, decked charges, air-decking, evolution of static energy distributions, and fragment data analysis for establishing optimized design patterns with available machinery. Several test blasts along with on-site testing of explosive quality, rebound hardness tests of overlying strata, and rearrangements of firing patterns through surface delay connections were considered for adopting the best-suitable blast pattern for the mine. Generalized and perceptible inferences were made to apply the results in other mines with similar kinds of problems.

**Keywords:** *: Fragmentation, Pilot holes, Pocket charge, Air-decking, Static energy distribution.*

## 1. Introduction

In bench blasting, the charge factor usually varies between 0.1 and 0.7 (kg/$m^3$), depending upon the strength of the rock being blasted. If fragmentation is not optimum, handling of oversize boulders becomes a costly affair. It also hampers the blasting cycle time, resulting in a loss of productivity. In a bid to determine the optimum fragmentation cost, Da Gama et al. [1] carried out an investigation on the average production costs in hard rock open-pit mines and revealed that drilling and blasting consumes 30%; loading 17%; crushing 20% and hauling 33% of it.

The selection of improper initiation time periods between rows in a blast is a frequent cause of poor fragmentation as well as backbreak [2]. Konya and Walter [3] have logically classified different pattern constructions and thereby suggested that each blasthole must be analyzed to determine its proper response in the pattern. A group of scientists of the CSIR-CIMFR Regional Centre in Nagpur recommended that the muckpile is quite dependent on the blast design, especially initiation timing, sequence, type of initiation, and cut configuration. They concluded that every blast design leads to a distinct size distribution of fragments, which influences the productivity of the subsequent operations [4]. Nielson [5] used a computer model to obtain an optimum cost from four different mining operations, namely drilling and blasting (D&B), loading, hauling, and crushing. He evaluated the optimum powder factor (kg/$m^3$) in Sydarager's open-pit iron ore mines of Norway by considering its influence on different subsystems of mining. Bhandari [6] recommended different ranges of the charge factor of explosives loaded with ANFO for surface coal and metal mining, considering there is no vibration problem arising due to blasting. Jimeno et al. [7] recommended that the maximum fragment size for the crusher is 80% of the maximum permissible size in the crusher. They also mentioned that the maximum recommended fragment size for loading a bucket is 0.7 times the bucket size. The Precision Blasting Services, USA, developed the blasting model 'BREAKER' which predicts and compares relative differences in fragmentation affected by many normal production blasting variables, such as burden, spacing, bench height, rock properties, and explosive properties. It also takes into account the rock's geologic and structural parameters to calculate a single size distribution based on the inputs from one blast and presents both numerically and graphically [8]. Pal Roy [9] showed through extensive field experiments that the reduction of boulders from the top zone, comprising massive and coarse-grained sandstone, could be possible by putting pilot holes drilled between the production holes along the rows. Singh [10] correlated the Rock Quality Designation (RQD) and the charge factor and showed that such a correlation could be useful for controlling fragmentation.

The Chotia opencast coal mine of M/s Prakash Industries Limited (PIL) was operating at a depth of about 30 to 40 m with an average bench height of 5.5 m. The massive formation of medium grained, white-coloured sandstone strata of about 20 m thick, located immediately above the coal seam in Quarry No. 2, posed severe blast fragmentation problems. The oversize boulders generated from such hard and massive sandstone strata were creating loading problems and therefore needed secondary blasting which enhanced the blasting cost and cycle time significantly. To improve blast fragmentation at its desired level, it was imperative to check the explosive characteristics through easily measurable parameters such as velocity of detonation, post-detonation fumes, density, water resistance, and loading density. Knowing it well that the explosive product had a direct bearing on rock fragmentation, such parametric evaluations indirectly helped the explosive manufacturer to improve their product quality. With such improved

* Corresponding author. *Tel/Fax*: +91-9431506000, *E-mail address:* : ppalroy.cimfr@gmail.com (P. Pal Roy).

explosive quality, the desired fragmentation could be achieved through a rigorous scientific study. The main objectives thereby confined to evolve optimum blast design patterns for reduction of boulders vis-à-vis cost optimisation, and reduction of blasting cycle time to enhance productivity [11].

Experimental blasts were conducted by adopting the (i) firing sequence of blastholes using JKSimblast software [12], (ii) pilot holes, (iii) pocket charge, (iv) measurement of the static energy distribution of explosive charge, (v) determination of explosive column vs. top stemming length, (vi) air-decking, etc. to find out the best-applicable design pattern. The in-hole velocity of detonation (VOD) as well as unconfined VOD were also tested using HandiTrap-II VOD Recorder of M/s MREL Group of Companies Limited, Ontario. The Surface hardness of different rocks was determined using the Schmidt hammer testing machine. Overall, the paper indicates how a systematic study, in a simple manner, can help in determining the real causes of unoptimized fragmentation and its remedial measures to achieve the target.

## 2. Rock deposits

The rock deposits of Quarry No. 2 were a matter of concern and therefore thoroughly studied. The rock strata typically formed of massive to medium grained sandstone, with a thickness of 18 to 20 m, located immediately above the coal seam. The colour of the sandstone was white and it contained strong cementing materials. Other than the lamination plane, inherent joint planes were not identified properly (Figures 1 to 3). Softer formations were present in the top bench and the strata were gently dipped with a dip amount of about 1 in 20. Hard and weathered sandstone, in a dark brown colour, was found at a few locations in the top bench near Matin-Di temple site.

Rebound hardness of the rock was determined on the bench faces as well as on the rock boulders using the Schmidt hammer instrument (Figure 4). The rebound hardness values of medium grained white sandstone varied between 24 and 29, whereas for the brown coloured sandstone at the Matin-Di site it varied between 27 and 31.

## 3. Experimental blasts

To achieve the target, the field trials were divided into three parts viz. existing patterns, refined patterns, and innovative patterns. These are explained below.

### *a) Existing Patterns*

Two blasts, both in the Matin-Di area, were conducted using the design patterns followed by the mine management in day-to-day operations. In the 1$^{st}$ Bench, the blasthole diameter was 160 mm, and depth of the holes varied from 3.0 to 6.0 m. The burden and spacing were 4.0 m and 5.0 m, respectively. In some portions of the blasting area, pilot holes of 1.5 to 1.8 m depth were also drilled. The total number of pilot holes was 23, and there were 79 primary holes. The explosive charge per hole varied between 15 and 60 kg. All the pilot holes were charged with 1.56 kg of explosive (1/4$^{th}$ of 125 mm diameter cartridge explosive weighing 6.25kg). Site-Mixed-Emulsion (SME) explosive, Shakti-Bulk 101 of M/s Special Blast Limited was used. Detonating cord was used for in-hole as well as surface hole-to-hole initiation. A diagonal pattern of firing was used using cord relays. In the 2$^{nd}$ Bench, the depth of the holes varied between 5.4 and 6.0 m. Some pilot holes of 1.5 to 1.8 m depth were also drilled. The Burden and spacing were 4.0 m and 5.0 m, respectively. The explosive charge per hole varied between 50 and 60 kg. The pilot holes were charged with 1.56 kg of explosive. The top stemming column varied from 3.0 to 3.2 m and a V-cut pattern of firing was selected using a detonating cord. The threshold of oversize boulders is determined by the available machinery to handle it optimally. As the operative excavators were EX250 & EX300 with a maximum bucket capacity of 1.7 m$^3$, boulders more than 1.19 m$^3$ (70%) sizes were considered as oversized with a maximum acceptable length of the boulder being 1.31 m (nominal diameter of the equivalent sphere).

**Figure 1.** Massive sandstone strata at Quarry 2.

**Figure 2.** Sandstone deposits above the coal seam.

**Figure 3.** Closer view of massive sandstone.

**Figure 4.** Determination of rebound hardness of rock.

### *b) Refined Patterns*

Five trials were conducted with refined blast design patterns compared to the existing patterns. The blasthole diameter was 160 mm, and the hole depth varied between 5.2 and 6.0 m. Two blasts were conducted with decked charges. Another three blasts were conducted using pilot holes (satellite holes) and pocket charges.

### *c) With decked charge*

In order to reduce the stemming column in a blasthole, two blasts were conducted with decked charges. One blast was conducted in the 3$^{rd}$ Bench at the Matin-Di area, and another blast was conducted in the 4$^{th}$

Bench of Quarry No. 2. At the Matin-Di area, the depth of holes varied from 5.1 to 6.1 m with an average depth of 5.8 m. The burden and spacing were 3.5 and 4.5 m, respectively. All the holes were drilled in a square pattern. The total number of holes was 45, and no pilot hole was used. In order to maintain a charge factor of 0.50 kg/m$^3$ (i.e., the powder factor of 2.0 m$^3$/kg), the explosive charge per hole was reduced to about 48 – 50 kg. However, due to some adjustment holes drilled at the front portion of the blast area, the overall charge factor became 0.57 kg/m$^3$ (i.e., the powder factor of 1.75 m$^3$/kg). The bottom of the hole was fitted with 35 kg, and the top had 13 to 15 kg separated by a decking of 1.3 to 1.5 m, as shown in Figure 5. The total explosive charge used in the blasting round was 2,150 kg. The top stemming column was maintained at 2.2 to 2.3 m for all the holes. Detonating cord was used for in-hole as well as hole-to-hole initiation (Figures 5, 6, 8 and 9). A V-pattern was used using cord relays as shown in Figures 6 and 7.

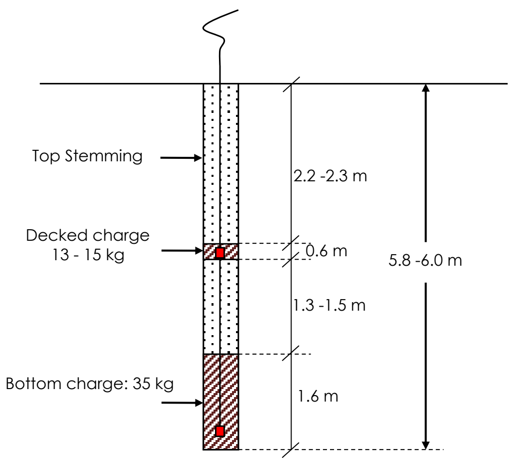


**Figure 5.** The charging pattern of holes using decked charge at the 3rd Bench of Matin-Di area.

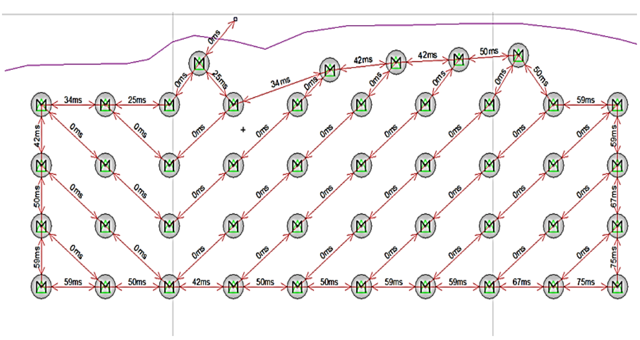

**Figure 6.** Surface firing pattern of the holes for the experimental blast conducted at the 3rd Bench of the Matin-Di area using the decked charge.

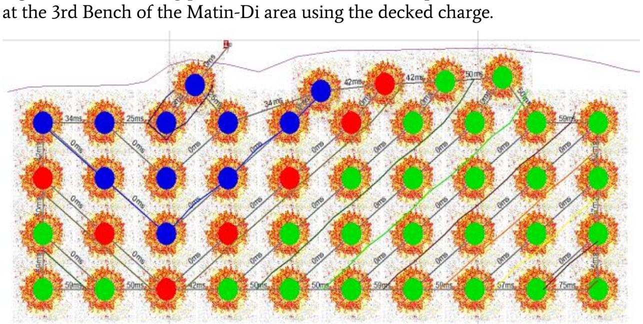

**Figure 7.** Firing sequence of the holes for the experimental blast conducted at 3rd Bench of the Matin-Di area using the decked charge.

At the 4$^{th}$ Bench of Quarry No. 2, the depth of holes varied between 5.3 and 6.2 m with an average depth of 5.9 m. No pilot hole was used. The burden and spacing were 4.0 m and 5.0 m, respectively. The total number of holes was 33, drilled in square pattern. The bottom charge was 40 kg, and top charge was 20 kg. The explosive charge per hole was 60 kg, and the total explosive charge was 1980 kg. The overall charge factor was 0.55 kg/m$^3$ (powder factor: 1.83 m$^3$/kg). The top stemming column was maintained at 2.2 to 2.3 m. The charging and firing pattern of holes are given in Figures 8 to 10.

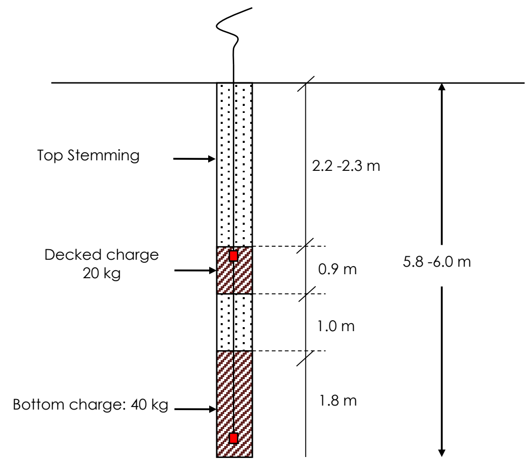


**Figure 8.** Charging pattern of the holes using the decked charge at the 4th Bench, coal-touch.

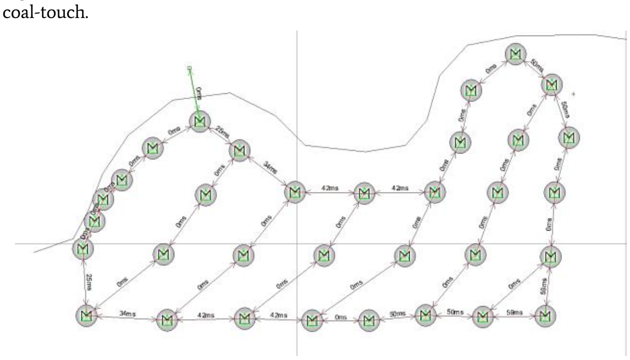

**Figure 9.** Surface firing pattern of the holes for the experimental blast conducted at the 4th Bench coal-touch using the decked charge.

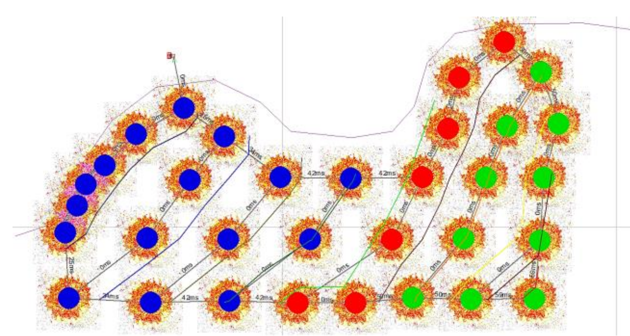

**Figure 10.** Firing sequences of the holes for the experimental blast conducted at the 4th Bench coal-touch using the decked charge.

### *d) With pilot holes & pocket charges together*

Three trial blasts were conducted using pilot holes as well as pocket charges with various design patterns. One blast was conducted at the 1$^{st}$ Bench (Top Bench) of the Matin-Di area on June 8th, 2013. The depth of the primary holes ranged between 5.7 and 6.2 m with an average depth of 5.9 m. Pilot holes were drilled between the primary holes along the middle of the main rows of holes. The depth of the pilot holes varied from 2.0 to 2.3 m. The burden between rows was 4.0 m and, the spacing of the main holes was kept at 5.0 m, drilled in square patterns. The total number of main holes was 49, and there were 30 pilot holes. The explosive charge per hole in the main rows varied between 50 and 60 kg.

In order to minimize the overbreak, a lesser explosive charge was used for all the last row of holes. In order to reduce the generation of boulders from the top stemming zones bearing harder strata, pocket charges along with pilot holes in between production holes as depicted in Figure-11 were incorporated. A pocket charge of around 2.0 kg was used in all the primary holes (nearly 1/3rd of 6.25 kg cartridge explosive). The top stemming column was maintained at 1.5 m for all the primary holes. All the pilot holes were charged with 4.0 kg (nearly 2/3rd of 6.25 kg cartridge explosive). The charge factor was 0.56 kg/m$^3$. The drilling, charging, and firing patterns of the holes are given in Figures 11, 12, and 13. The total explosive charge fired in the blasting round was 3004.90 kg.

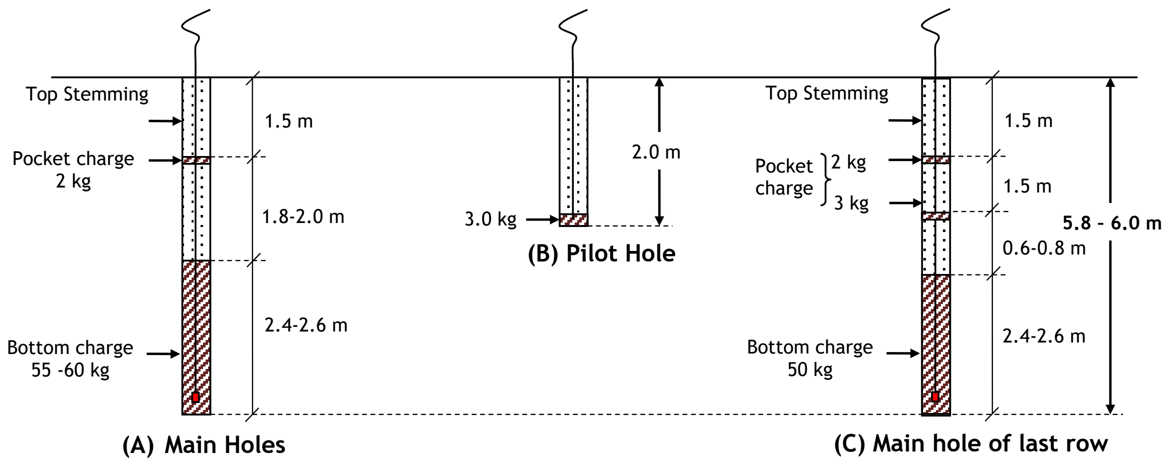


**Figure 11.** Charging pattern of the holes using the pilot holes and pocket charges at the 1st Bench of the Matin-Di area.

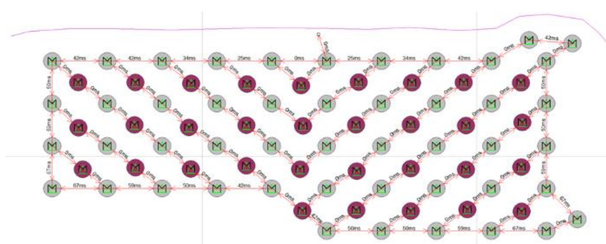

**Figure 12.** Surface firing pattern of the holes for the experimental blast conducted at the 1st Bench of the Matin-Di area using the pilot holes & pocket charges.

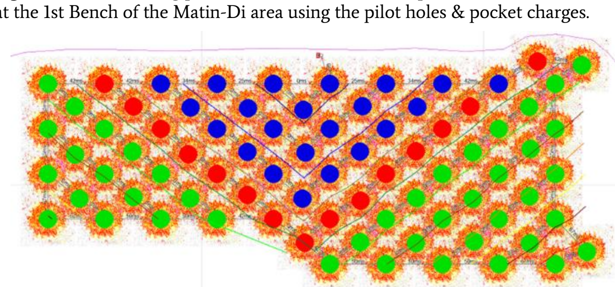

**Figure 13.** Firing sequence of the holes for the experimental blast conducted at the 1st Bench of the Matin-Di area using the pilot holes & pocket charges.

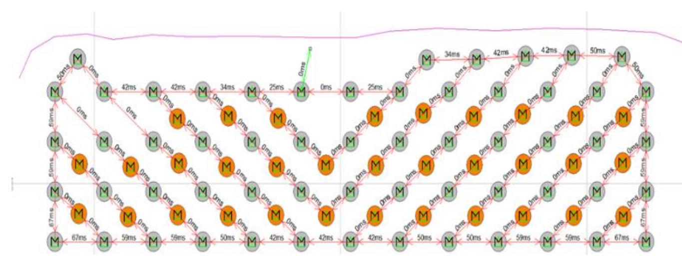

**Figure 14.** Surface firing pattern of the holes for the experimental blast conducted at the 4th Bench of the Quarry No. 2 using the pilot holes & pocket charges.

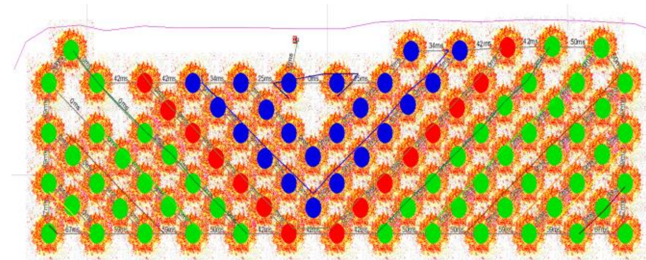

**Figure 15.** Firing sequences of the holes for the experimental blast conducted at the 4th Bench of the Quarry No. 2 using the pilot holes & pocket charges

The same blasthole geometry and pilot hole patterns were used for the trial blasts conducted at the 4th Bench of Quarry No. 2 and the top bench of the Matin-Di area.

In the 4th bench of Quarry No. 2, the total number of primary holes was 61, whereas the number of pilot holes was 30. The hole depths varied from 5.0 to 6.0 m with an average depth of 5.66 m. The depth of pilot holes was 2.0 m. The quantity of pocket charge for the primary holes as well as pilot holes was kept at 3.125 kg (half-cartridge). The total explosive charge was 3293.60 kg. The calculated charge factor was 0.56 kg/m$^3$. The firing pattern and sequence of holes are given in Figures 14 and 15.

In the experimental blast conducted at the top bench of the Matin-Di area, the depth of the pilot holes was reduced to 1.75 m from 2.0 m. The burden and spacing for the primary holes were kept at 4.0 m and 5.0 m, respectively. The depth of the holes for the primary blastholes varied from 5.5 to 6.0 m with an average depth of 5.90 m. The total number of the primary blastholes was 30 and the number of pilot holes was 15. All the pilot holes were charged with 4.0 kg of explosive (2/3rd of the 6.25 cartridge explosive). The pocket charge was used for all the main holes with 3.125 kg of explosive (1/2 of the 6.25 kg cartridge explosive). The firing pattern and sequence of holes are given in Figures 16 and 17.

## 4. QUALITY ASSESSMENT OF EXPLOSIVE

The quality of bulk explosive was assessed through the conventional methods of measuring the velocity of detonation (VOD) and density at the time of charging holes. For testing VOD in an unconfined condition, the D'Autriche Method was adopted, whereas for testing in-hole VOD (i.e., in a confined condition), the HandiTrap VOD Instrument was used (Figures 18 & 19). The density of the explosive before and after gassing was measured during hole charging (Figure 20). The density of the explosive before gassing varied from 1.35 to 1.37 g/cc and after gassing, it reduced to 1.20 - 1.25 g/cc.

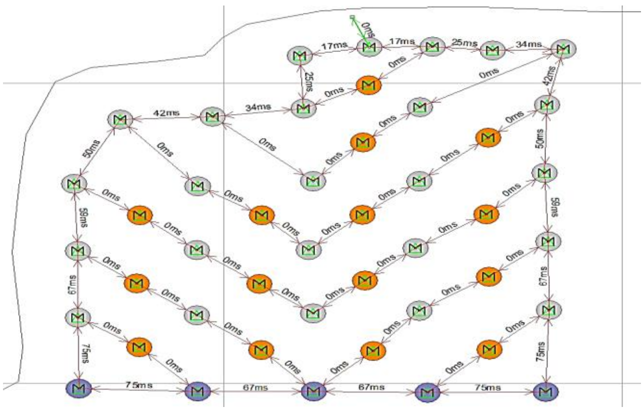

**Figure 16.** Surface firing pattern of holes for the experimental blast conducted at the 1st Bench of the Matin-Di area using the pilot holes & pocket charges.

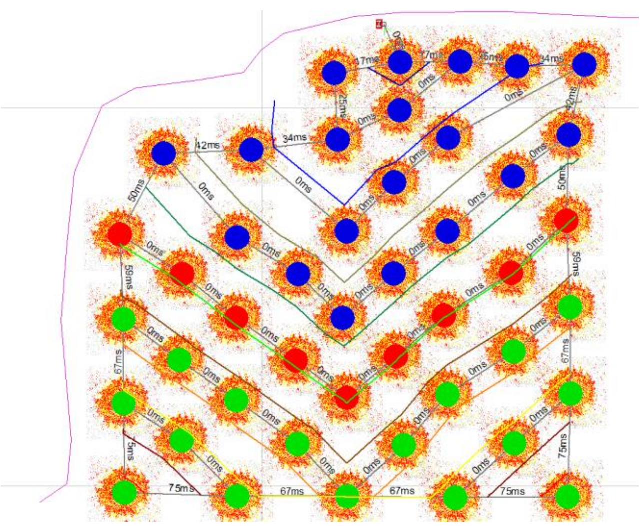

**Figure 17.** Firing sequence of the holes for the experimental blast conducted at the 1st Bench of the Matin-Di area using the pilot holes & pocket charges.

**Figure 18.** VOD measurement by the D'Autriche.

In the D'Autriche method, a plastic pipe with an inner diameter of 100 mm with a thickness of 2 mm, and a length of 600 mm, as well as an aluminum plate of 30 cm length, 4 cm in width, and 0.5 cm in thickness were used. Bulk explosive (Shakti-Bulk 101) was poured inside the plastic pipe as shown in Figure 14. About 3.0 m length of a detonating fuse of known VOD was cut, and the middle point was marked. The detonating fuse was then placed along the aluminum plate in such a way that the center of the detonating fuse coincided with the reference line marked on the aluminum plate.

**Figure 19.** VOD measurement using the HandiTrap-II.

**Figure 20**. Explosive density measurement.

For the determination of in-hole VOD, The HandiTrap-II VOD Recorder of M/s MREL Canada was used. The instrument uses the proven continuous resistance wire technique to monitoring of VOD. An MREL-manufactured probecable-HT (or proberod) of known linear resistance was radially placed on the explosive column. As the detonation front of the explosive consumed the probecable-HT, the resistance of the circuit decreased in proportion to the reduction in the length of the probecable-HT. HandiTrap-II recorded the consequent resulting decrease in voltage across the probecable-HT versus time.

The in-hole velocity of detonation (VOD) of SME Shakti Bulk 101 of M/s Special Blast Limited measured by HandiTrap-II was 4360.8 m/s, whereas the unconfined VOD determined through the D'Autriche method was 4276 m/s. The trend of the graph depicted in Figure-21 verified the uniformity of the explosive composition.

## 5. Results and observations

No significant improvement in blast fragmentation was observed with the decked charge though the top stemming column was reduced to 2.2 - 2.3 m by using decked charge. It was observed that the oversize boulders still generated from the top portions of the blastholes (Figures 22 & 23). However, a good fragmentation was observed in the bottom portion of the blastholes, and the backbreak was significantly controlled (Figure 24). Due to the use of 160 mm blasthole diameter, the lineal charge concentration of explosive was high, and therefore the explosive column could not be increased by maintaining the charge factor of 0.54-0.58 kg/m$^3$. In some blasts, it was further observed that a few boulders were generated due to the shearing action of the detonating cord.

The experimental blasts conducted using pilot holes and pocket charges showed improvements in blast fragmentation (Figures 25-27). Better blast fragmentation was obtained in comparison to the existing blasting patterns as well as decked charges. A better distribution of explosive charge in the top portion of the holes could be achieved using pilot holes and pocket charges. The fragment size analyses in Figures 22-25 are shown in Figure 28 depicting the usefulness of pilot holes and pocket charges. A lesser value of the uniformity index (n) obtained in Figure 25 indicated that the fragmentation was well-graded. The static energy distributions of explosives in the case of blasting conducted without pilot holes and with pilot holes and pocket charges combined are given in Figures 29 & 30, respectively. It is clear from these figures

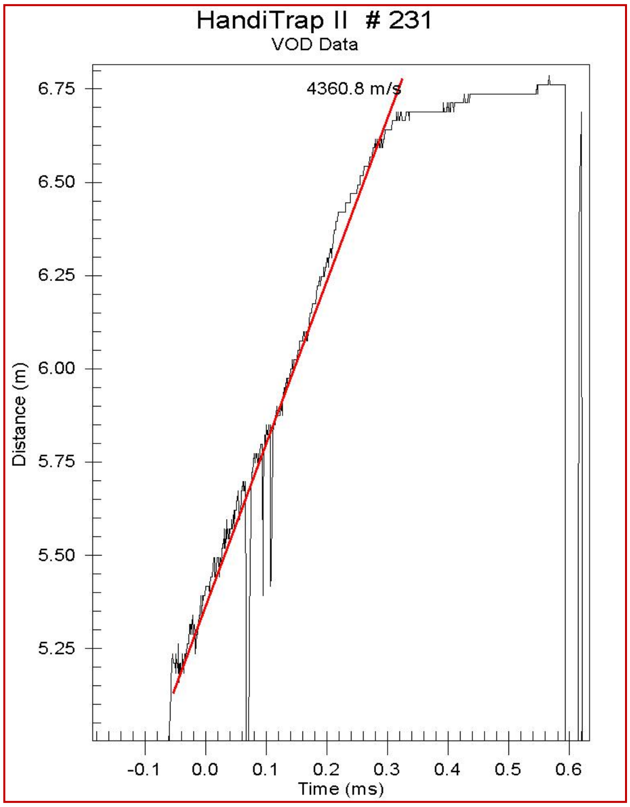


**Figure 21.** Graphical output of the HandiTrap-II showing the VOD value.


**Figure 22.** Oversize boulders generated from the top portion of the holes in the trial blast conducted with the decked charge at the 3rd Bench coal-touch (Matin-Di area).


**Figure 23.** Oversize boulders generated from the top portion of the holes in the trial blast conducted with the decked charge at the 4th Bench, Quarry No. 2.


**Figure 24.** Good cut with minimum backbreak obtained with the decked charge.


**Figure 25.** Fragmentation obtained using the pilot holes and pocket charges at the 1st Bench (Matin-Di area).


**Figure 26.** Fragmentation obtained using the pilot holes and pocket charges at the 4th Bench, Quarry No. 2

**Figure 27.** Fragmentation obtained using the pilot holes and pocket charges at the 1st Bench, Matin-Di area.

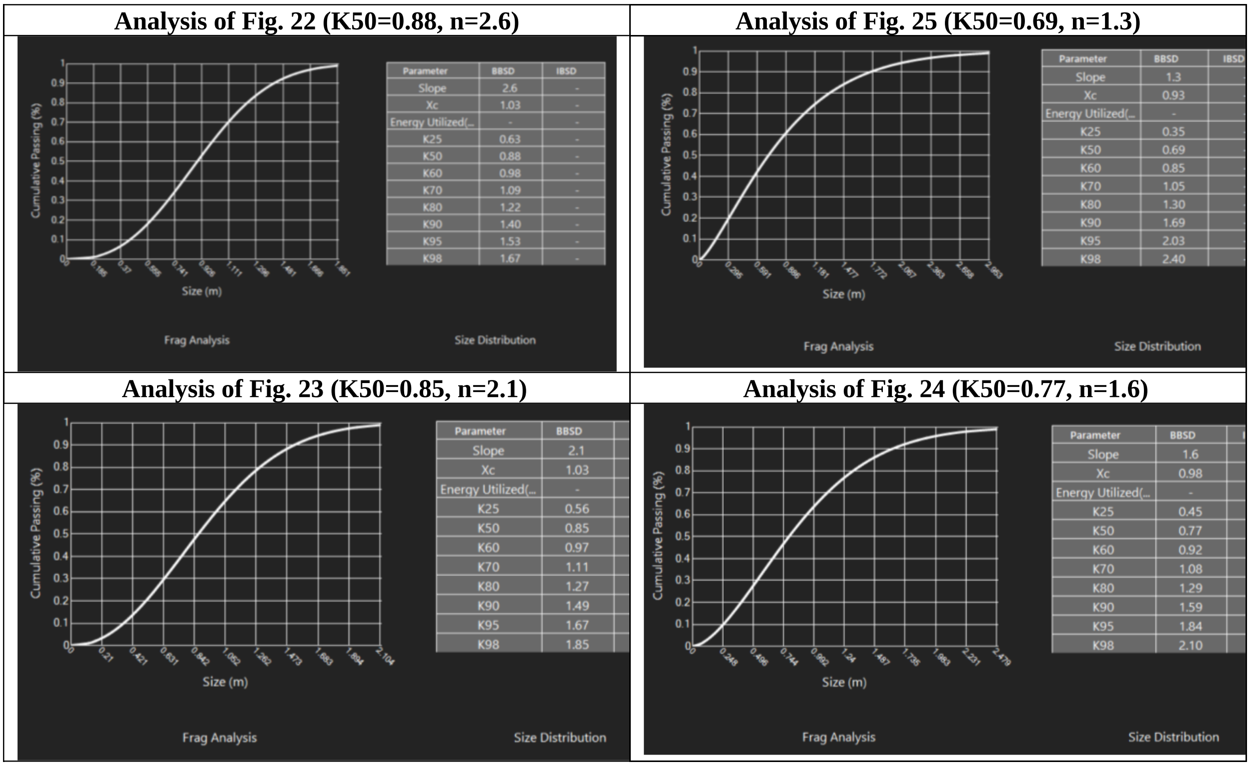


**Figure 28.** Fragment size analyses of figures 22-25 before and after optimisation.

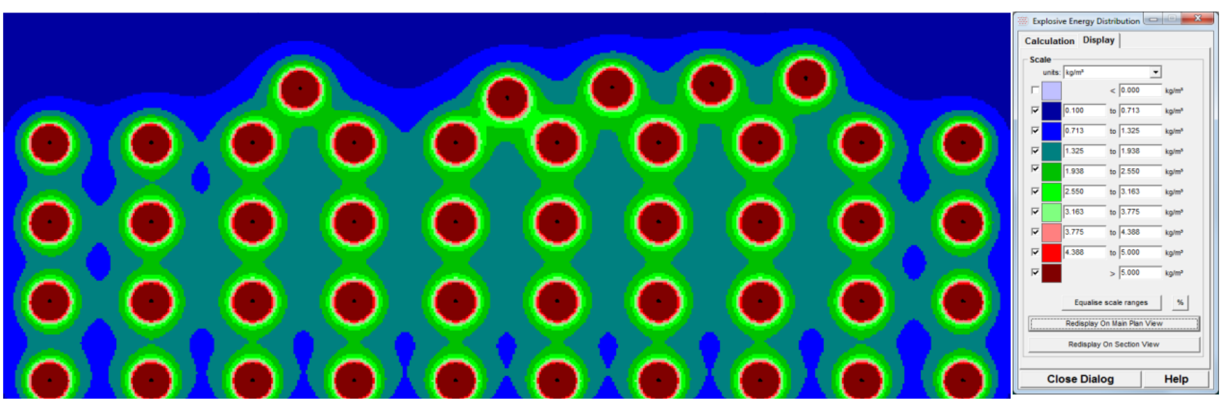


**Figure 29.** Static energy distribution of explosive charge for the experimental blast conducted with the decked charge, but without pilot holes at the 3rd Bench, Matin-Di area.

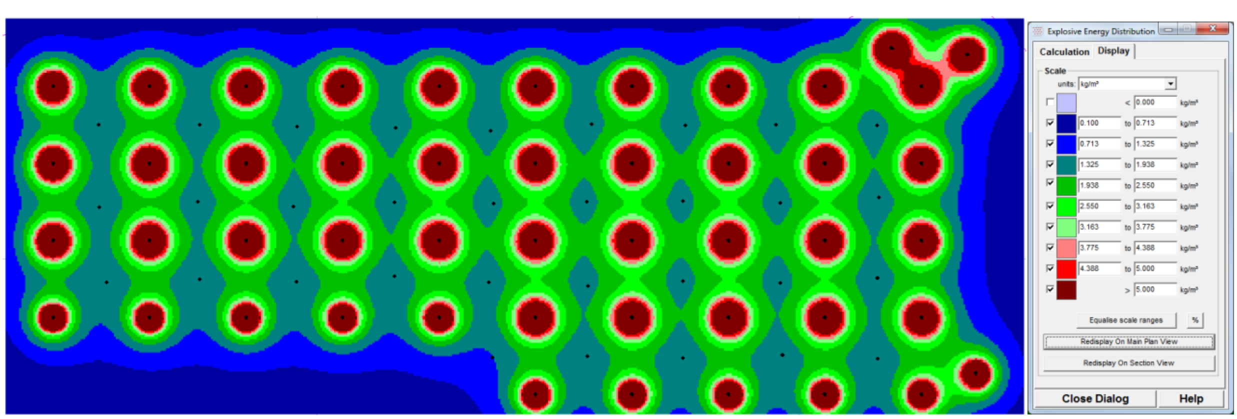


**Figure 30.** Static energy distribution of explosive charge for the experimental blast conducted with the pilot holes and pocket charge at the 1st Bench, Matin-Di area.

that more uniform energy distribution of the explosive was achieved with pilot holes and pocket charges combined. The amount of explosive charge used in the pilot holes varied from 3.0 kg to 4.0 kg (around 1/2 to 2/3 portion of 6.25 kg cartridge explosive). The amount of explosive charge used for pocket charging varied between 2.0 kg and 3.0 kg. With the increased quantity of explosive charge in pilot holes and pocket charges, further better fragmentation could be achieved. The overall charge factor was maintained between 0.50 and 0.58 kg/m$^3$ to minimize the explosive cost.

## 6. Observations

(1) The rebound hardness values indicated that the medium grained, white coloured sandstone, above the coal seam in Quarry No. 2, was massive in nature but not very hard and therefore it required a uniform distribution of explosive charge to obtain good fragmentation.

(2) Although good fragmentation was noticed in the lower portion of the blastholes, boulders were mainly generated from the top uncharged portion of the holes i.e., in the stemming column areas. Due to concentration of more explosive charge at the bottom of the holes, backbreak and overbreak were observed which resulted into boulder formations in subsequent blasts.

(3) The lineal charge concentration for a 160 mm blasthole diameter was measured to be 23 kg/m for a 1.15 g/cc density of explosive. Therefore, increasing the explosive column without an increase in the explosive charge was difficult.

(4) The use of pilot holes and pocket charges was seen to produce good blast fragmentation. It was further expected that the enhanced explosive charge in pilot holes and pocket charges would result in better blast fragmentation.

(5) It was apparent that for a drilling geometry of 6 m blasthole depth, 4 m burden, and 5 m spacing with pilot holes of depths 1.75 to 2.0 m, a charge factor of more than 0.58 kg/m$^3$ was required to obtain proper fragmentation. It also reduced the powder factor from 2.0 to 1.73 m$^3$/kg.

(6) The explosive charge for the last row of holes was distributed using a decked charge, and the bottom charge was reduced to minimize overbreak/backbreak. The overbreak from the last row of holes was significantly minimized using such charging pattern.

Therefore, proper distribution of the explosive charge was found to be essential to improve blast fragmentation due to the massiveness of the sandstone strata. To increase the charge column, it was proposed either to apply 115 mm blasthole diameter instead of 160 mm with air-decking.

The use of a 115 mm diameter for a 6 m blasthole depth would definitely result in better blast fragmentation compared to a 160 mm diameter, since the explosive column (or loading density of explosive) can be increased with the same charge factor or the same quantity of explosive charge. This will also improve the powder factor. The explosive charge column and the length of the top stemming column for 115 mm and 160 mm blasthole diameter with different quantities of explosive charge for a blasthole depth of 6 m are shown in Figures 31 and 32. It is apparent from the adjacent bar charts that more explosive column length could be achieved with a 115 mm diameter compared to a 160 mm diameter for the same explosive quantity. Hence, better blast fragmentation could be achieved with an improved powder factor.

Another way of improving the powder factor as well as fragmentation with a 160 mm blasthole diameter is by air-decking technique. With the introduction of air-decking, the charge length can be raised while maintaining the same quantity of explosive. Specially made gas-bags or pre-fabricated wooden spacers can be used for air-decking. However, the air-decking technique cannot generally be used in bulk explosives. Therefore, its application requires extensive trial blasts for evolving a fullproof system.

## 7. Conclusions and recommendations

The massive deposit of sandstone at Chotia OCM required proper distribution of explosive charge in the blastholes (primary and pilot holes) in order to obtain proper blast fragmentation. The rebound hardness values of medium-grained, white sandstone present immediately above the coal seam at Quarry No. 2 varied between 24 and 29 indicating that the sandstone rock was not very hard in nature. The in-hole VOD using HandiTrap-II of SME Shakti Bulk 101 of M/s Special Blast Limited was determined as 4360.8 m/s, whereas the unconfined VOD obtained from the D'Autriche method was 4276 m/s, indicating the consistency of the explosive product.

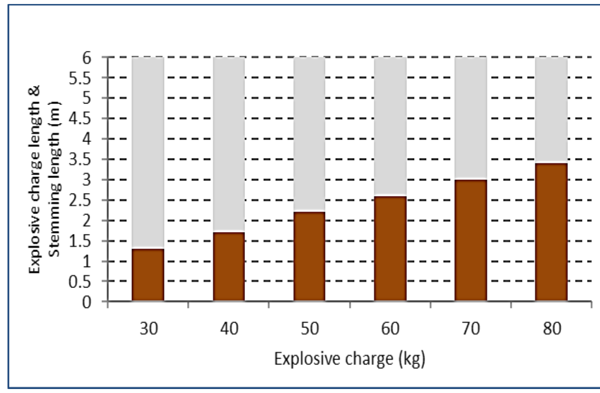


**Figure 31.** Explosive charge length and top stemming column length for a 160 mm blasthole diameter (explosive density – 1.15 g/cc).

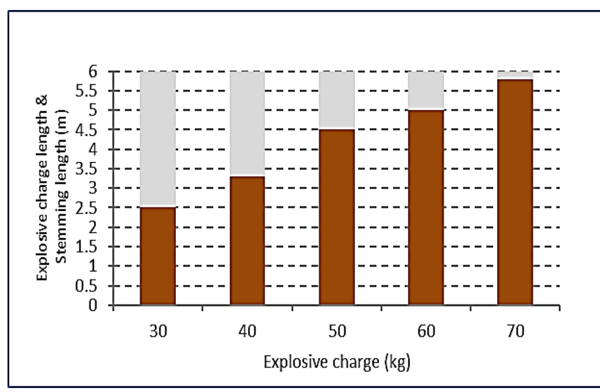


**Figure 32.** Explosive charge length and top stemming column length for a 115 mm blasthole diameter (explosive density – 1.15 g/cc).

The results of trial blasts conducted with a 160 mm blasthole diameter indicated that the main fragmentation problem and boulder formations arose from the top uncharged portion i.e., the stemming column, whereas the good breakages were obtained from the charged column in all the blasts. It also created severe overbreak and backbreak. However, the use of pilot holes and pocket charges was found to produce better blast fragmentation. The amount of explosive charge used in each pilot hole varied between 3.00 and 4.00 kg, and the pocket charge varied from 2.00 to 3.00 kg.

With a 160 mm blasthole diameter, it was difficult to enhance the explosive charge column due to the higher lineal charge concentration. The overall study showed that a 0.58 kg/m$^3$ charge factor (i.e., a powder factor: 4.2 ton/kg) was required to achieve good fragmentation with a 160 mm blasthole diameter. Alternately, the usage of a 115 mm blasthole diameter or 160 mm with air-decking were found to be useful.

## Acknowledgements

The financial assistance provided by M/s Prakash Industries Limited for the field trials is greatly acknowledged. The research team also deeply appreciates the cooperation and help extended by M/s Rungta Private Limited and M/s Special Blast during the field trials.

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