

# Rock Fragmentation Control In Open Cast Mining

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## **ABSTRACT**

The blasting operation plays a pivotal role in the overall economics of opencast mines. The blasting sub-system affects all the other associated sub-systems, i.e. loading, transport, crushing and milling operations. Fragmentation control through effective blast design and its effect on productivity are the challenging tasks for practicing blasting engineer due to inadequate knowledge of actual explosive energy released in the borehole, varying initiation practice in blast design and its effect on explosive energy release characteristic. This paper describes the result of a systematic study on the impact of blast design parameters on rock fragmentation at three mines in India. The mines use draglines and shovel-dumper combination for removal of overburden. Despite its pivotal role in controlling the overall economics of a mining operation, the expected blasting performance is often judged almost exclusively on the basis of poorly defined parameters such as powder factor and is often qualitative which results in very subjective assessment of blasting performance. Such an approach is very poor substitutes for accurate assessment of explosive and blasting performance. Ninety one blasts were conducted with varying blast designs and charging patterns, and their impacts on the rock fragmentation were documented. A high-speed camera was deployed to record the detonation sequences of the blasts. The efficiency of the loading machines was also correlated with the mean fragment size obtained from the fragmentation analyses.

## **1. INTRODUCTION**

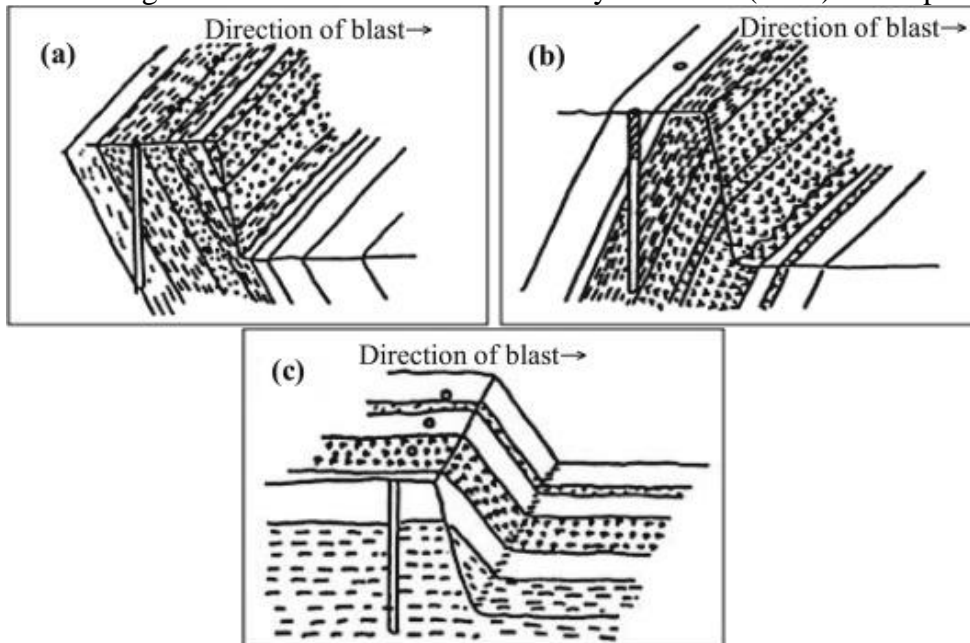
Rock fragmentation distribution influences a range of mining and milling processes including load and haul rates, crushing and grinding performance and ore recovery in beneficiation processes (Michaud et al., 1997). In opencast mining, where blasting is employed for excavation, the overall cost effectiveness of the production operations is compatible with optimization of drilling and blasting parameters. Rock fragmentation depends upon two groups of variables: rock mass properties which cannot be controlled and drill-and-blast design parameters that can be controlled and optimized. The costs of downstream operations can be reduced by optimizing the blast design parameters to provide target fragmentation. The parameters of target fragmentation are equipment specific and vary from category of mine to mine. The high level of mechanization and the integrated nature of the production systems adopted in the mining industry demand that all the units must function with the designed reliability and capacity to achieve planned production targets (Singh and Narendrula, 2009).

The objective of a blasting engineer in a mine is to generate a suitable muck pile having suitable size distribution of the rock that can be efficiently loaded, transported and milled

(Singh et al., 2005). The goal of efficient blasting can be achieved by investigating the relationship between blast design parameters and fragmentation. It is extremely important to make the connection between rock blasting results and their impact on the downstream operations. It is well accepted that fragmentation has a critical effect on the loading operations, but little quantitative information is available, upon which rational blasting strategies can be outlined. Spathis, 2002, Spathis, 2009 discussed some aspects of size reduction and its influence on mineral liberation, which mainly described the area of prediction and assessment together with the related assumptions: fines, mean size, oversize, cumulative size distributions, and measurement protocol.

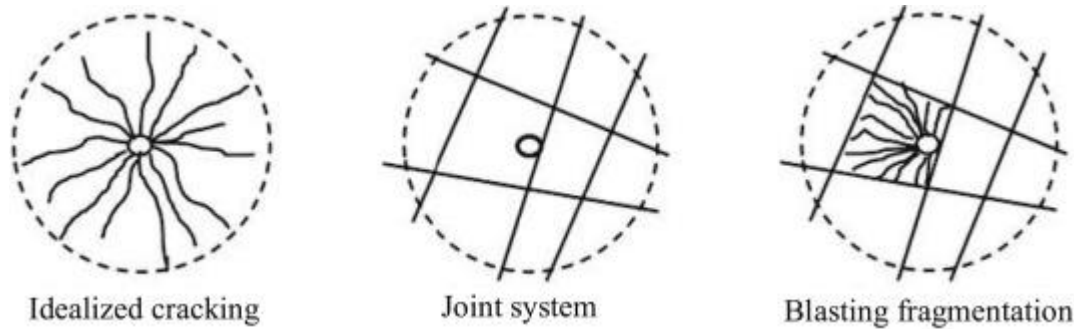
Total cost of aggregate production in a quarry has a minimum value at an optimum fragmentation size (Mackenzie, 1967, Morin and Ficarazzo, 2006). Prediction of the optimum fragmentation size will help the quarry owners in selecting blasting parameters to produce required material size at a known cost and also in selecting other crushers and conveyor systems. Optimum fragmentation size may not be the required size but knowing the size distribution for particular blast and rock mass conditions, the contractor can adapt the blasting if possible (Engin, 2009).

Hustrulid (1999) cited from Burkle (1979) that blasting results are affected by the orientation of the rock mass structures. Three cases which have to be considered are: (i) shooting with the dip, (ii) shooting against the dip, and (iii) shooting along the strike. While shooting with the dip, backbreak increases, toe problem decreases resulting in a smooth floor, and throw of the blast increases resulting in scattered and low muck pile (Fig. 1a). When shooting against the dip, there are less backbreak, more toe problems resulting in uneven floor, and decreasing throw of the blast resulting in higher muck pile profile (Fig. 1b). Finally, when shooting along the strike (Fig. 1c), the floor can be highly toothed due to the different rock types intersecting the floor. For the same reasons, the backbreak is irregular. The effect of jointing on rock fragmentation has been documented by Hustrulid (1999) and is presented in Fig. 2.



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Fig. 1. Diagrammatic representations of (a) shooting with dip, (b) shooting against the dip, and (c) shooting along strike (Burkle, 1979).



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Fig. 2. Effect of jointing on fragmentation (after Hustrulid, 1999).

The Kuz–Ram model is generally used for prediction of the fragmentation size after blasting. The Kuz–Ram model is an empirical fragmentation model based on the Kuznetsov (1973) and Rosin and Rammler (1933) equations modified

## 2. Experimental site details

Investigations were carried out at three mines in India, i.e. Nigahi project of Northern Coalfields Limited, Sonepur Bazari project of Eastern Coalfields Limited, and Kusmunda project of South Eastern Coalfields Limited.

The Nigahi project stands out as a hilly plateau with elevation of about 400–450 m above the mean sea level. The rocks are of lower Gondwana formation. There are three coal seams, i.e. Turra, Purewa bottom and Purewa top seams (Purewa bottom and top seams are combined at few locations). The thicknesses of the coal seams are 13–17 m, 11–12 m and 7–9 m, respectively. The direction of strike is towards E–W with broad swings. The dip of the coal seam is 1°–4° in northerly direction. The block has 491.8 Mt of coal reserves. In Turra and Purewa (bottom, top and combined) seams, the average stripping ratio is 1:3.76, i.e. 3.76 m<sup>3</sup> of overburden is to be removed for extraction of 1 t of coal. The mine is currently producing 14 Mt of coal per annum.

Sonepur Bazari project of Eastern Coalfields Limited is located in the eastern part of Raniganj Coalfields. Four coal seams, i.e. R-IV, R-V, R-VI and R-VII, are mainly exposed in the mine. Presently, seams R-V and R-VI are being extracted by opencast method of mining. The mine is producing about 4.5 Mt of coal and removal of overburden is about 12 Mm<sup>3</sup>. The stripping ratio of the mine is 1:4.72, i.e. 4.72 m<sup>3</sup> of overburden is to be removed for mining of 1 t of coal. The total coal reserve of the mine is 188.26 Mt.

Kusmunda project is located in the western bank of Hasdeo River in the central part of Korba Coalfields in the district of Korba, Chhattisgarh State. Kusmunda project is having a flat terrain with minor undulations. The general elevation ranges from about 290 m to 300 m above mean sea level. The seams generally have a dip ranging from 5° to 10° and the overall grade of coal is of Grade ‘F’. The mine produces about 18.75 Mt of coal per year. The total overburden handled is about 30.69 Mm<sup>3</sup>. The overview of the Kusmunda opencast project is shown in Fig. 3.



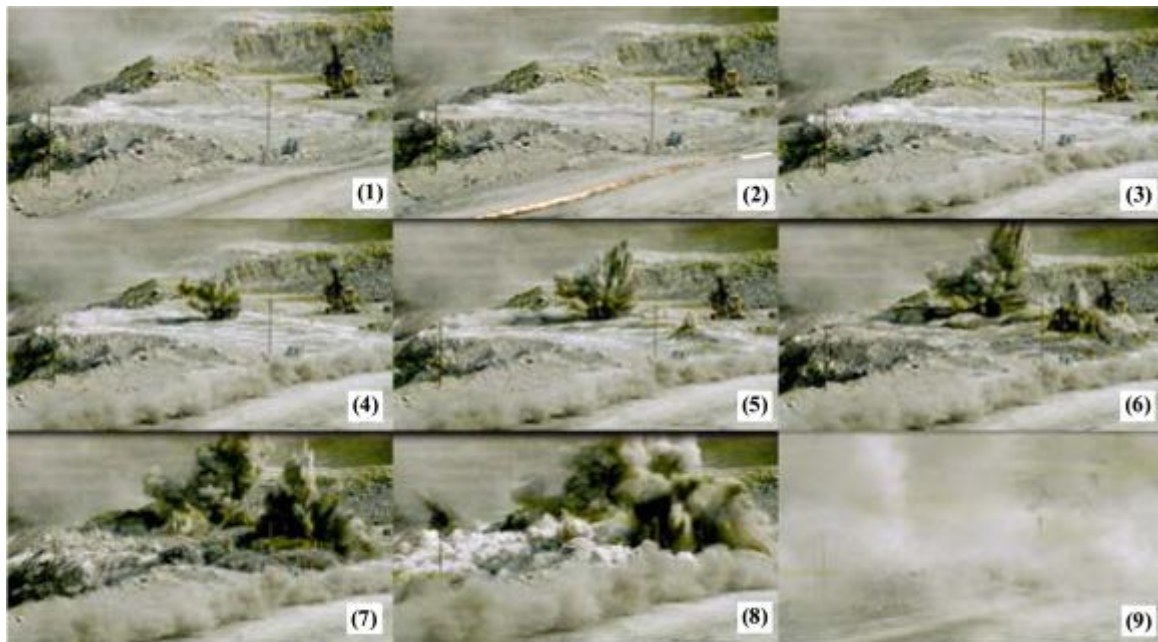
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Fig. 3. Overview of the Kusmunda project.

### 3. Methodology

Blast design parameters of bench blasting are the controlling parameters which regulate the desired fragmentation level of a particular blast. Rock mass properties and blasting parameters control the efficiency of a blasting operation. But, all the blast design parameters cannot be changed depending on the type of strata and bench height. Hole diameters of 159 mm, 259 mm, 269 mm and 311 mm were used depending on their bench height. The bench height is related to the working capability of loaders and varies from 5 m to 42 m. A few blasts were performed by the existing blast design practiced in the mine and after each blast, 18–25 scaled digital photographs throughout the complete mucking of the fragmented rock pile were taken as well as loading efficiency of the shovel was recorded. Fragmentation characteristics such as mean fragment size, uniformity index and characteristic size were calculated by using digital images in an image analysis system called Wipfrag<sup>®</sup> software. The physico-mechanical properties of rock sample collected at Nigahi, Sonapur Bazari and Kusmunda projects are presented in Table 1. Fig. 4 depicts the view of the detonation sequence of shovel bench blast at Kusmunda project. Fragmentation analyses were carried out for all the blasts in different segments. The view of the post blast results of different benches at Kusmunda project is depicted in Fig. 5.

Table 1. Physico-mechanical properties of rock collected from respective mines.

Name of the project	Rock type/Location	Uniaxial compressive strength (MPa)	Tensile strength (MPa)	Density (kg/m <sup>3</sup> )	Poisson's ratio	Young's modulus (GPa)
Sonapur Bazari	Sandstone/Dragline bench	37.29	3.46	2320	0.23	7.05
	Sandstone/Shovel bench	36.52	3.41	2300	0.23	7.02
Nigahi	Sandstone/Dragline bench	31.73	3.53	2054	0.21	3.41
	Sandstone/Shovel bench	29.56	3.23	2010	0.2	3.25
Kusmunda	Sandstone/Shovel bench	26.59	2.14	2017	0.25	5.57



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Fig. 4. View of the detonation sequence of shovel bench blast conducted at Kusmunda project.



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Fig. 5. Post blast view of different benches at Kusmunda project.

#### 4. Analysis of data

The blast design parameters data collected from 91 blasts from three experimental sites are analyzed to find out their impacts on rock fragmentation level. The main important parameters which decide the fragmentation level of particular blasts are burden to

hole diameter ratio, spacing to burden ratio, stemming column length, stiffness ratio, explosives amount and type, initiation mode and charge/powder factor. Table 2 summarizes the details of the trial blast conducted at three experimental sites. Fig. 6 represents the elements of blast design parameters. The near field blast vibration signals were also recorded to diagnose the impact of delay timing on rock fragmentation. The blast wave signals recorded at 100 m from one of the hard overburden dragline bench blast are depicted in Fig. 7. In this blast, the delay interval between the holes in a row was 17 ms and between the rows the delay intervals were 65 ms, 84 ms, 100 ms, 117 ms, 134 ms and 150 ms in subsequent rows. Fig. 8 represents the blast wave signals recorded at 100 m from the hard OB bench (dragline bench) blast at Sonepur Bazari project. In this blast, the delay intervals were 17 ms between the holes in a row and 65 ms and 84 ms between the rows. Other blast wave signals recorded at 100 m from the shovel bench blast with delay interval of 17 ms between the holes in a row and 59 ms, 84 ms and 101 ms between the rows are presented in Fig. 9

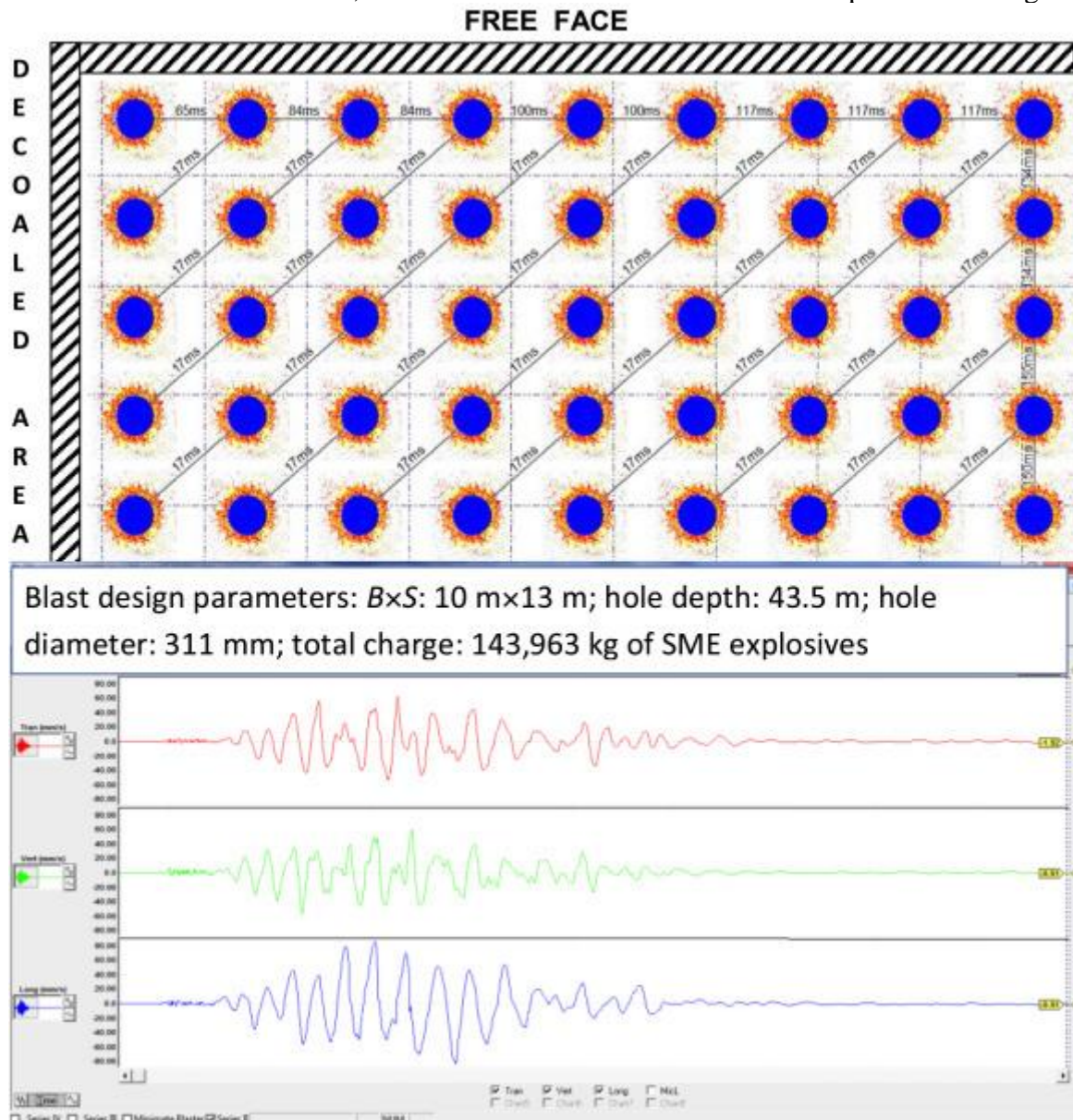
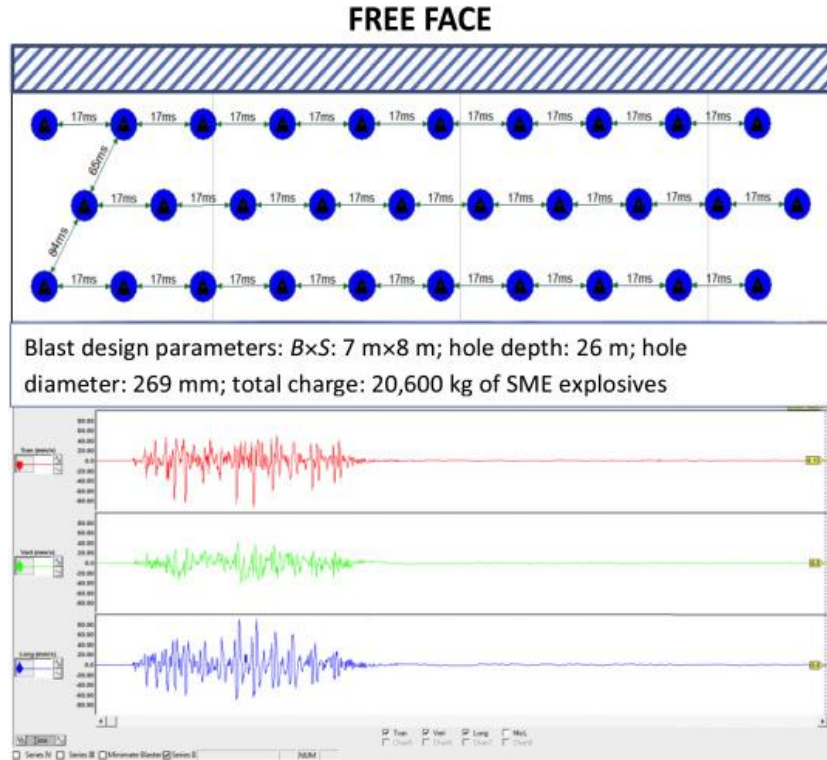
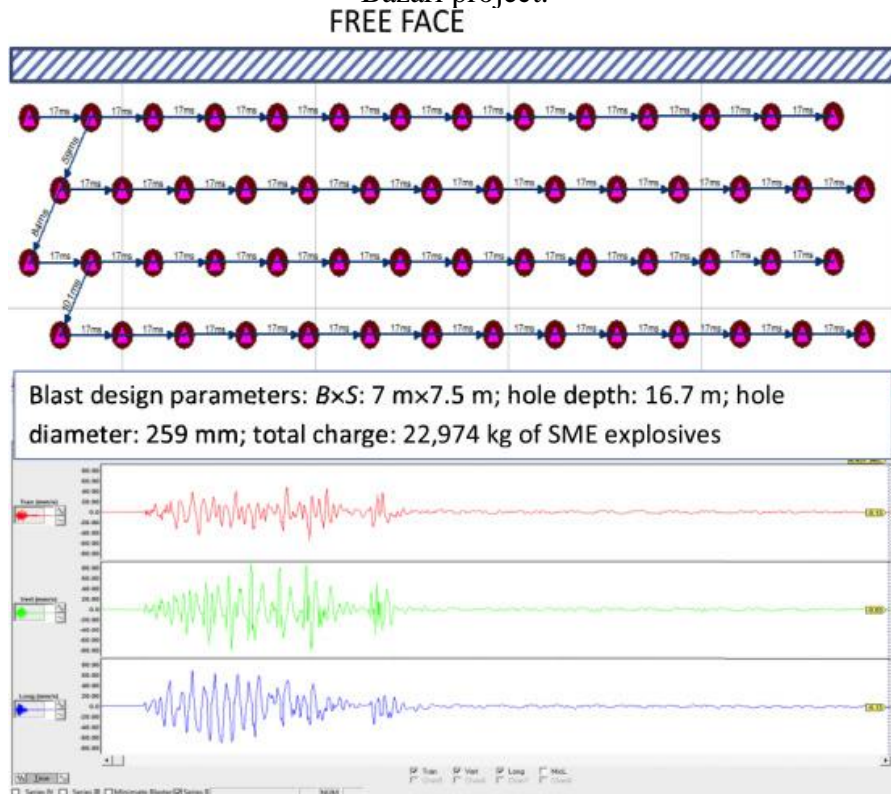


Fig. 7. Blast wave signals recorded at 100 m from the dragline bench blast at Nigahi project.



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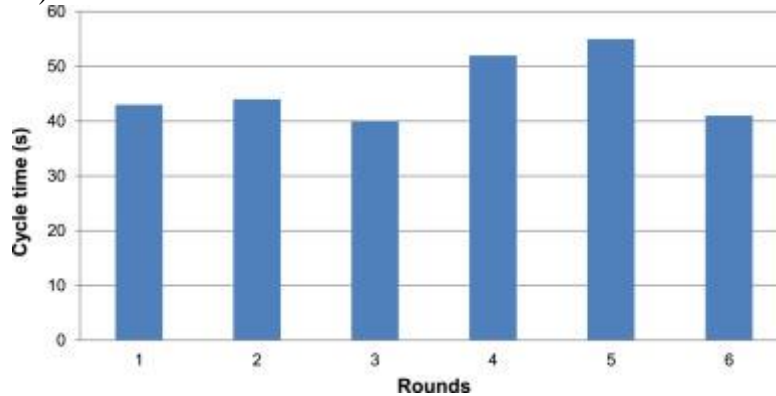
Fig. 8. Blast wave signals recorded at 100 m from the dragline bench blast at Sonapur Bazari project.



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Fig. 9. Blast wave signals recorded at 100 m from the shovel bench blast at Kusmunda project.

The fragmentations achieved from these blasts reflected different results. Most of the blasts results were excellent in terms of fragmented rock mass and its uniformity. A few blasts results also have shown scattered results in terms of large size boulders and fine and dust particles as represented in terms of uniformity index ( $n$ ). Most of the blasts resulted in good muck piles. The fragment size analyses were carried out using Wipfrag software. The output of the analyses are in the form of number of exposed fragmented blocks, maximum, minimum and mean sizes of the fragmented blocks, sieve analysis as per the requirement, i.e. at different percentile sizes of  $D_{10}$ ,  $D_{25}$ ,  $D_{50}$ ,  $D_{75}$  and  $D_{90}$  (percentile sizes: for example,  $D_{10}$  is the ten-percentile, the value for which 10% by weight of the sample is finer and 90% coarser. In terms of sieving,  $D_{10}$  is the size of sieve opening through which 10% by weight of the sample would pass).

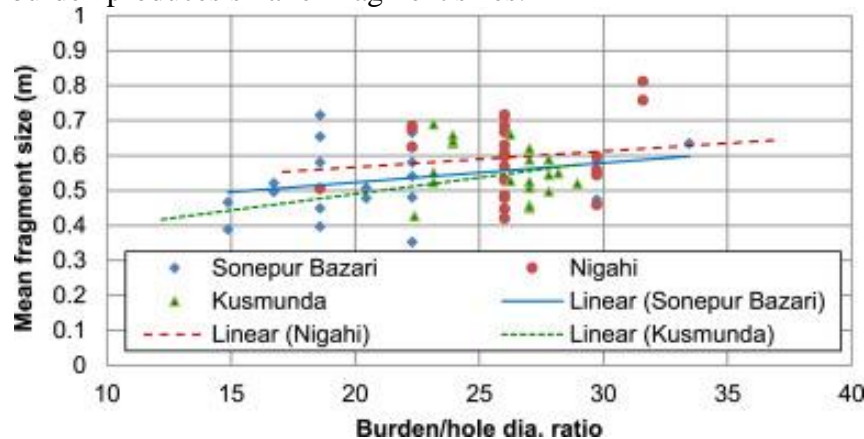


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Fig. 17. Plot of cycle time of shovel at medium hard overburden shovel bench of Kusmunda project.

#### 4.1. Burden to hole diameter ratio

Hole diameter and burden are two important blast design parameters. In these trial blasts, hole diameters were of three different types, i.e. of 259 mm, 269 mm and 311 mm but out of 91 blasts, 85 blasts were conducted with 259 mm and 269 mm diameters. Therefore, it can be said that the variation in burden to hole diameter ratio was in fact the variation in burden alone. Fig. 18 depicts the plot between burden to hole diameter ratio vs. mean fragment size. It is observed from Fig. 18 that the mean fragment size decreases with decrease in burden to hole diameter ratio. A few data do not show the expected trend probably because of the impact of geology on blast fragmentation. In general, the small diameter holes with smaller burden produces smaller fragment sizes.

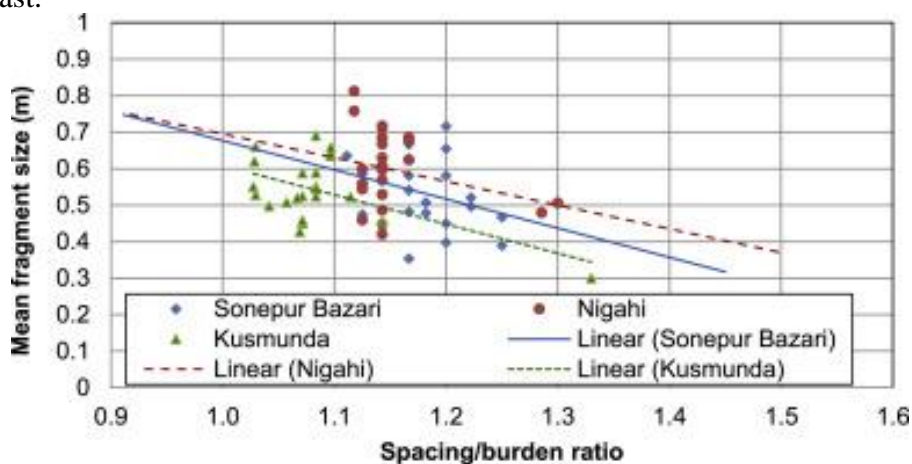


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Fig. 18. Burden to hole diameter ratio vs. mean particle size.

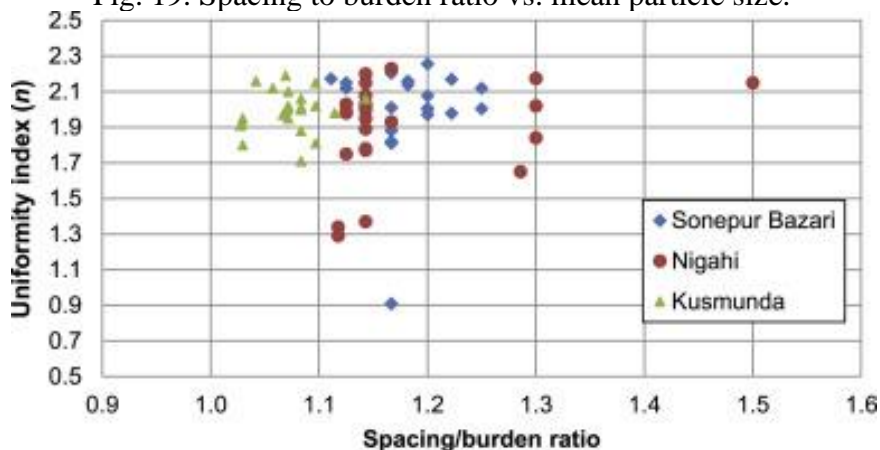
### 4.2. Spacing to burden ratio

Spacing and burden are important parameters and have immediate impacts on rock fragmentation in blast design. Excessive burden creates resistance to penetrate the explosion gases into the fracture and displace rock, and will also produce excessive vibration level. Small burden allows the gases to escape and push the blasted rock uncontrollably with high speed. Small spacing causes excessive crushing between the holes and superficial crater breakage. Excessive spacing results in inadequate fracturing between the blast holes which creates irregular faces with toe problems. The spacing to burden ratio can also be adjusted by changing the blast detonation sequence through different cut designs, i.e. diagonal firing, V-cut and elongated V-cut firing. If burden is not compatible with spacing, the blast holes will not connect, resulting in inadequate use of explosive energy. In general, the spacing to burden ratio varies between 1 and 2 but the optimal spacing to burden ratio was 1.15 for staggered pattern and 1.25 for rectangular pattern (Hagan, 1983). Mean fragment size and uniformity index ( $n$ ) versus spacing to burden ratio are plotted in Fig. 19, Fig. 20, respectively. As most of the data have slight variation in spacing to burden ratio, the outcome of the graphs is not so significant. It also appears from Fig. 20 that uniformity index in few blasts is not in accordance with trend line which resulted in either poor fragmentation in terms of fine fragment or large size boulders. However, in Fig. 19, the spacing to burden ratio between 1.1 and 1.3 shows excellent blast results except for a few blasts which have low uniformity index ( $n$ ) due to presence of joints and backbreaks of previous blast.



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Fig. 19. Spacing to burden ratio vs. mean particle size.

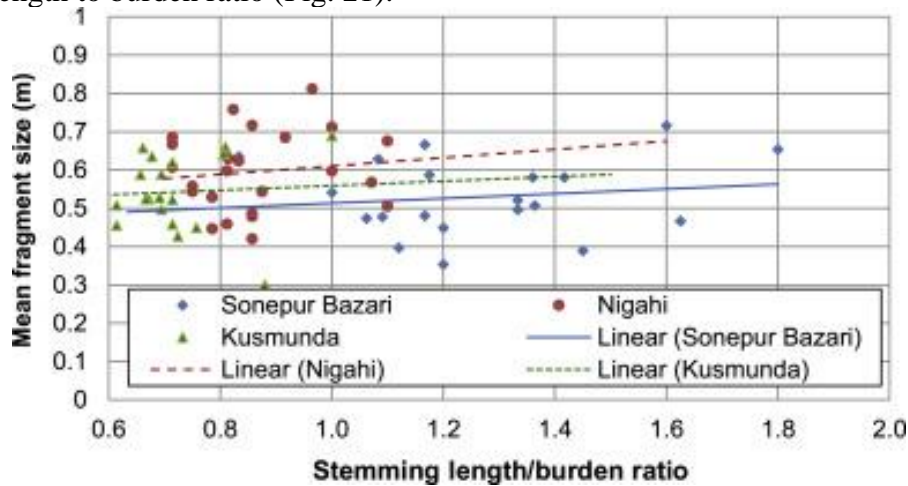


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Fig. 20. Spacing to burden ratio vs. uniformity index ( $n$ ).

### 4.3. Stemming length to burden ratio

Stemming length is another blast design parameter that affects rock fragmentation. This becomes even more significant when the blast faces encounter hard rock near the blast hole collar zone. If the rock has natural cracks in burden portion, then long stemming may be recommended, but on the other hand, for massive rock, stemming column is required to be kept short. For the blasts in sandstone benches of coal mine, stemming length to burden ratio was plotted against mean fragment size. The data points are relatively scattered but the general trend shows that the mean size of fragmented rock decreases with the decrease in stemming length to burden ratio (Fig. 21).

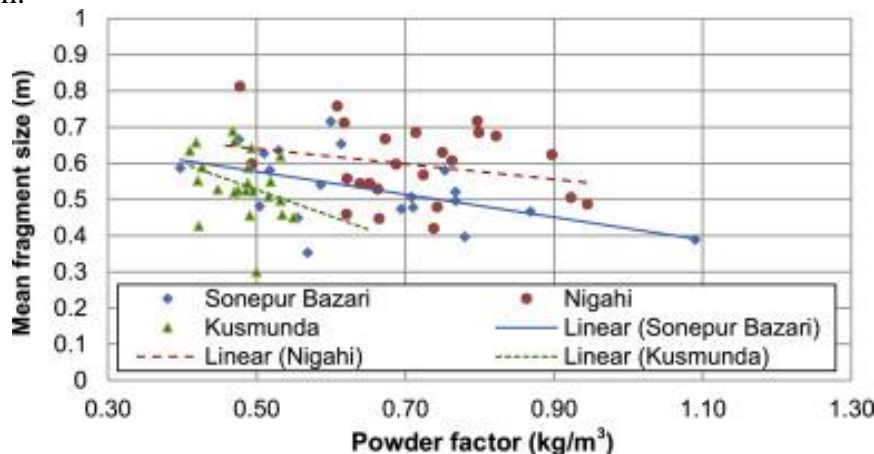


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Fig. 21. Stemming length to burden ratio vs. mean fragment size.

### 4.4. Powder factor

Powder factor is the ratio between the amount of rock broken and total weight of explosive consumed. It is an important parameter in blast design and has a vital influence on the resultant fragmentation. Higher powder factor causes oversize and lower powder factor results in crushed rock. Mean fragment size was plotted against the powder factor for 91 blasts of coal overlying overburden benches, as presented in Fig. 22. The general trend shows that, with increase in powder factor, the mean fragment size decreases. A few scattered data in this graph are expected due to the geological discontinuities of the rock mass of the blasting patch.

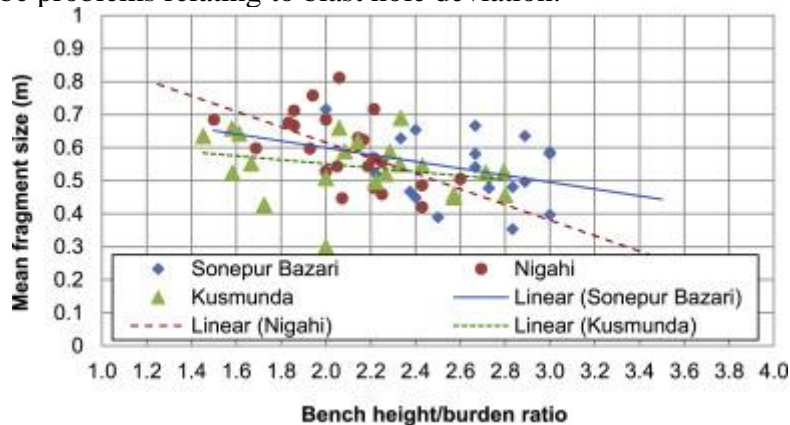


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Fig. 22. Powder factor vs. mean fragment size.

#### 4.5. Stiffness (bench height to burden ratio)

Stiffness is the bench height to burden ratio and also influences the resultant fragmentation. Although, the bench height is usually decided on the basis of the working specification of the loading equipment, the bench height should also be adequate to achieve optimal burden, spacing and powder factor (Singh and Abdul, 2012). The mean fragment size was plotted against stiffness as shown in Fig. 23. It is observed that the stiffness value of less than 2 gives coarser fragmentation and the best optimum value comes around 3. Change in the burden or spacing has significant effect on rock fragmentation. In case of high stiffness value, it is easy to displace and deform rock especially at the center of the bench (Ash, 1985) but on the other hand, there can be problems relating to blast hole deviation.



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Fig. 23. Bench height to burden ratio vs. mean particle size.

#### 4.6. Joint plane orientation and spacing

Joint and bedding planes act as natural pre-splits during blasting and if possible, should be used to improve blast performance. For example, horizontal bedding allows pull to be maximized and the blasted rock will tend to split horizontally. Spacing of joints within a rock mass will have significant impact on the size distribution of the blasted muck. In general, the joint spacing will also improve the fragmentation level. It is suggested that in a rock mass with small joint plane spacing, explosives having lesser shock energy and high gas energy should be used; while in case of rock mass having larger joint spacing, higher shock energy and lesser gas energy should be used for better shattering effect.

### 5. Conclusions

The main conclusions of the study are drawn as follows:

1. Mean fragment particle size increases with the increase in the burden to hole diameter ratio. This increase was mainly due to the increase in burden as the hole diameter was kept constant.
2. Mean fragment size of the blasted muck decreases with the increase in the spacing to burden ratio. The optimum value of spacing to burden ratio ranged from 1.1 to 1.3 and resulted in excellent rock fragmentation. It was found that the uncontrollable parameters such as joints and fractures have significant influence on uniformity index ( $n$ ).
3. Stemming length to burden ratio was plotted against mean fragment size and the general trend shows that mean fragment size of fragmented rock decreases with the decrease of stemming length to burden ratio.
4. As anticipated, the increase in the charge/powder factor will increase the rock fragmentation level, i.e. decrease in the mean fragment size of the rock.

5. The stiffness (bench height to burden ratio) vs. mean fragment size plot indicates decrease in mean fragment size with increasing stiffness.

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