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Generation and Accuracy Assessment of Digital Elevation Model Using Digital Photogrammetry and Differential Global Positioning System Techniques

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Abstract: Blasting can be considered as the most crucial process in an opencast mine. It is therefore important for mining engineers to understand the effect of geological discontinuities and blast design parameters on the results of blasting. Bench height and burden are very important parameters affecting stiffness of bench. Joints alter the results of blasting, by making explosive energy utilization ineffective. Modern tools like high speed videography reveal many aspects of fragmentation process, which otherwise are difficult to visualize and understand. An attempt is made through this paper to present some of the research results of model scale studies, coupled with field study results related to bench height and joints in order to improve blast results.

Keywords: *Rock Fragmentation, Geological Discontinuities, Bench Height, Burden, Joint Orientation, Gas Energy, Shock Energy, High Speed Videography*

1. Introduction:

Detonation of an explosive charge confined in a blasthole releases a vast amount of chemical energy, which is then transformed into gaseous energy. This gaseous energy exerts an enormous amount of pressure on the blasthole wall. This pressure results in the generation of shockwaves carrying shock energy. According to Sadwin and Junk (1965), the explosive can be categorised in this phase by two pressures:

- Detonation Pressure: Dynamic pressure associated with the detonation wave.
- Explosive Pressure: Pressure developed when the explosive reacts to produce gaseous products.

Thus, the detonation of an explosive under confinement creates two types of energies: strain energy (5-20% of total explosive energy) and gaseous energy (80-95% of total explosive energy).

For an efficient blast, it is necessary to utilize the explosive energy for productive work as much as possible. Design of an efficient blast requires sound knowledge about the role of blast design parameters, explosive characteristics and the structural discontinuities in fragmentation process. Joints, the most commonly occurring discontinuities in the rock mass play a significant role in influencing the blast results. This study makes an effort at relating various parameters of a blast, which can be used to design an efficient blast, with special emphasis on the usage of high speed videography.

2. Literature Review

2.1 Role of strain and gas energies

Before 1959, it was generally perceived that rock breakage is caused mainly by strain waves. However, Fogelson et al. (1959), conducted a series of tests to measure the explosive energy transmitted to rock mass by strain waves, and determined that the strain waves only play a minor role in fragmentation.

Burden rock movement studies by Noren (1956) showed that strain waves cannot be the dominant factor in rock fragmentation, as he showed that it had time to travel the burden distance at least six times before any surface rock movement started. Also, he observed that the burden accelerated continuously during its motion. Had strain waves acted upon it, then the motion would have been discontinuous. Saluja (1963) found out that in case of high explosives, the rock is fractured by a combination of gaseous and strain energy. However, he also showed that in low explosives (gunpowder), the breakage occurs solely due to gaseous energy (Clark and Saluja, 1964).

In 1971, Kutter and Fairhurst proposed a more generalized theory based on their experiments. They argued that:

- Both strain waves and gaseous energy play an important role in fragmentation.

- Strain wave functions to pre-condition the rock mass by initiating radial cracks.
- Gaseous energy then expands and extends the cracks.
- Presence of free surfaces favours extension of gas pressurized radial cracks.
- In-situ stresses significantly influence the direction of radial crack propagation.

According to the gas pressurization configuration given by Kutter and Fairhurst (1971), gases under high pressure penetrate into radial cracks forming a hydrostatically stressed cylinder of material where the hydrostatic stress is equal to the gas pressure (Figure 1). This large hydrostatically stressed cylinder applies pressure at the back of the burden, inducing bulk rock movement (Figure 2).

Fragmentation of rocks can be explained by another mechanism, known as “Flexural Rupture Mechanism”, which involves the transverse fracturing of segments formed by radial cracks (Figure 3).

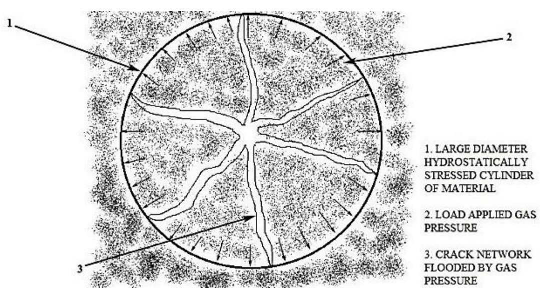


Figure 1. *Hydrostatic stress field created around a blasthole (Kutter and Fairhurst, 1971)*

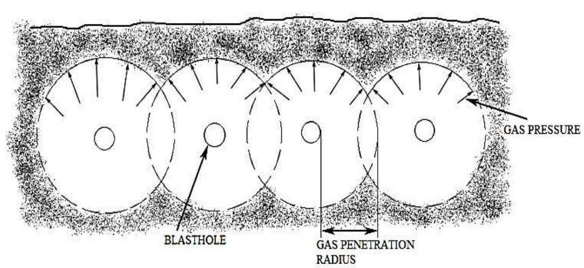


Figure 2. *Gas pressure applied load at the periphery of hydrostatically stressed cylinders causing rock movement (John, 1983)*

In this theory, 90% of the total energy required to break the rock was assumed to come from gaseous pressure alone. The sustained gaseous pressure drives radial cracks through the burden upto the free face and displaces the rock through bending, in the direction of the least resistance, generally following the naturally occurring weakness planes.

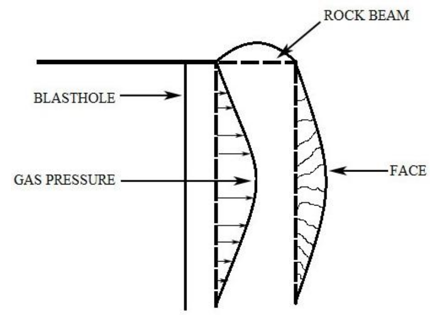


Figure 3. *Rock breaking by flexural bending (Ash, 1973)*

Ash and Smith introduced the stiffness theory in 1976, according to which, the degree of fragmentation depended upon the stiffness property of the burden rock. In terms of blasting, *Burden*, *Spacing*, and *Bench Height* are the three main factors affecting the burden rock stiffness. For achieving good fragmentation, the burden to bench height ratio needs to be properly analysed as the stiffness varies to the third power of this ratio. Reducing burden for a given bench height has been shown to have a positive effect. Thus, increase in bench height reduces the stiffness of burden rock mass.

2.2 Effect of joints

Joints are the most common discontinuities present in rock mass. They create impedance mismatch zones in the strain wave transmitting medium and thereby cause unusual reflection and/or refraction of strain energy. Joints interrupt the development of radial crack network, and thus control the shape and size of the crater to a large extent. The crater formed in the jointed rock mass closely conforms to the network of the weakness planes (Bauer et al., 1965).

When a blasthole is intersected by joints, explosive energy escapes through joints, opening them up by wedging action causing a sudden drop in blasthole pressure (Figure 4). In some cases, when weak or open joints extend up to the face, premature venting of gases takes place, giving rise to fly rock and air blast problems (Figure 5).

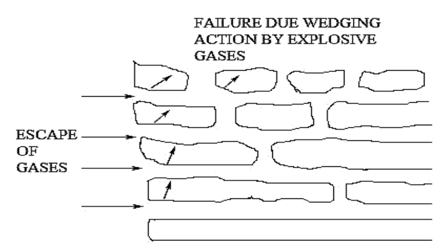


Figure 4. *Escape of gases into joints*

When there is any open joint opposite to the blasthole, the surfaces of joint cause reflection of the strain wave which in turn interacts with the incoming strain waves. If this intensity is sufficiently strong, fragmentation occurs due to the internal spalling in that zone. As a result, there may be more boulder formation on the other side of the joint (Singh and Sastry, 1986a; Sastry 1989). Rinehart (1970) analysed the effects of joints on the wave propagation and observed that localization of fragmentation occurs near joint planes.

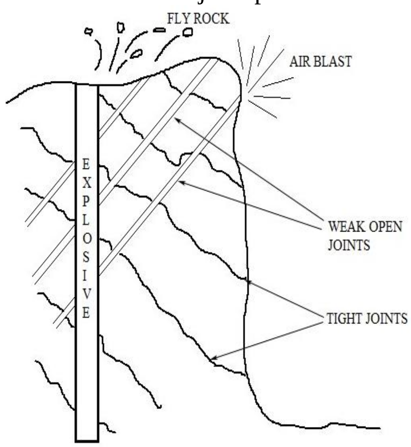


Figure 5. *Air blast and fly rock associated with jointed rock mass*

Similar results were obtained by Sastry (1989) in his laboratory studies on Chunar sandstone models. Pugleise and Atchison (1964) in their comparative studies of explosives in limestone with tight joints found that repeated blasting in the area opened the joints present in the rock mass and thereby affecting the subsequent blasts results.

Emergence of cheaper blasting agents has set a trend towards larger diameter blastholes with increased burdens and spacings. As a result, in blocky strata with large joint spacing, the effects of weakness planes become more pronounced as greater number of joints may be encountered between consecutive blastholes. These results in very poor fragmentation, creating problems to loading, hauling and crushing operations in addition to the unwanted toe formations, as a number of blocks are not penetrated by blastholes (Figure 6). In such cases, small diameter blastholes array with smaller burden and spacings makes the explosive energy distribution more even, giving better results.

Joint planes cause stress concentration zones and create new fractures along the pre-existing flaws. Tests conducted by Barker and Fourney (1979) on Homolite-100 models revealed this phenomenon. This was also supported by the studies of Lande (1983), who suggested that in jointed and highly fractured rock mass, short delays with smaller burdens give better fragmentation.

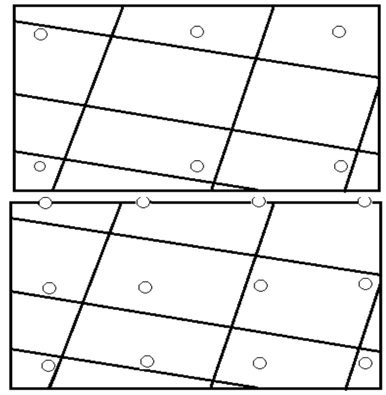


Figure 6. *Effect of joints on blasthole array*

2.3 Type of joints

Joints are of three types – tight, open and filled. The degree of impedance offered to strain waves depends upon the type of joints. Tight joints do not affect the transmission of strain waves as much as open or filled joints. Sometimes the joint plane itself acts as a pseudo-face, especially in the case of open joints, reflecting the strain wave (Sastry, 1989).

Joint filling material, which may be the product of weathering or decomposition of the joint walls, is also a factor exerting considerable influence on the blast results. According to Yang and Rustan (1983), continuity of weakness plane is the major factor affecting fragmentation. Strength of joint, which depends on the filling material, is the next. They observed that open and air filled joints exert a strong control on the fragmentation.

Sastry (1989) observed from the tests on sandstone with four different filling materials (siliceous and calcareous materials, water and air), that the size and shape of bench crater were controlled more by the joint filling material (Figure 7). Larger fragments and larger sockets were observed in models with filled joints.

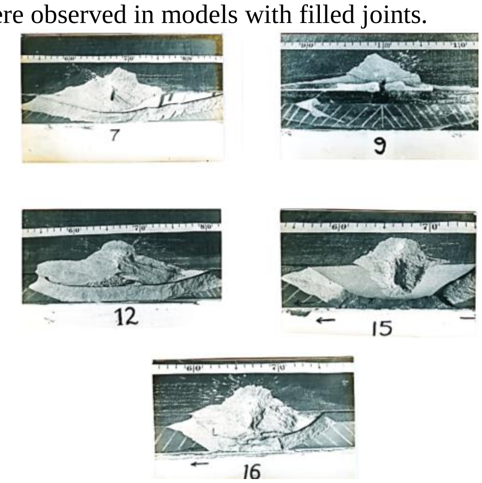


Figure 7. *Model studies performed using filling materials in joints*

2.4 Joint spacing

The spacing or frequency of joints plays a vital role in fragmentation of the rock mass. Resistance to the blasting increases as block size increases or joint frequency diminishes (Da Gama, 1970). For a successful blast design, data on spacing of joints should be obtained from a joint survey. Also, a detailed study of burden (B), joint spacing (S) and maximum allowable size of block (M) helps in overcoming the problems encountered in blocky formations. In general blasting practice, there are six possible cases of above mentioned variables (Table 1).

Table 1. *Relationships between Burden (B), Joint Spacing (S) and Maximum Acceptable Fragment Size (M) (Coates, 1981)*

Cases	Relative values of S and M			Dominant Influence	Boulder Formation (Size > M)
	Greatest	Intermediate	Smallest		
1	B	M	S	Jointing	Low
2	B	S	M	Jointing	High
3	M	B	S	Jointing	Low
4	M	S	B	Explosive	Low
5	S	B	M	Explosive	Medium
6	S	M	B	Explosive	Low

It is clear from Table 1 that Case-1 is the ideal, due to small amount of boulders and reduced explosive consumption, because joints dominate the fragmentation. Case-2 is not desirable due to the formation of large number of boulders as a result of joint spacing being greater than ‘M’. Cases-3, 4, and 6 have low probability of occurrence in properly designed blasts, because burden ‘B’ is smaller than the accepted block size ‘M’. Case-5 is also rare, for the burden being less than average joint spacing ‘S’ causing undesired boulder formation. Hence, blast design must be done in order to find situations, where B<M and M<S, for reducing secondary fragmentation as well as specific explosive consumption.

2.5 Joint orientation

Orientation of weakness planes (joints) has significant influence over size and shape of broken material and excavation (Ash, 1973; Gnirk and Pflieder, 1968). Formation and extension of cracks during blasting are controlled by the pattern of joints (Ash, 1973; Dally and Fourney, 1977). Bauer et al. (1965) and Ash (1973) from their studies concluded that craters formed closely conform to the geometry of weakness planes.

When the face is parallel to and on the dip side of the joints, excessive sliding occurs creating significant overbreak problems. When the joints dip away from the face, there may be problems of overhangs toe, etc., but the walls will be more stable (Larson and Pugleise, 1974). Results from small scale bench blasts showed that when a row of vertical blastholes was oblique to the joint direction, it resulted in poor fragmentation (Sastry, 1989). By orienting the free face parallel to the marked vertical joint planes, better results may be achieved (Belland, 1966).

It is reported that in horizontally bedded deposits, vertical lifter holes produce better fragmentation results (Wild, 1976). Thin and horizontally deposited brittle rocks require only horizontal holes, so that the overlying strata slides down by gravity. Rocks like Basalt, which are deposited in the form of thick vertical columns and the rocks with intersecting slips may require both vertical as well as horizontal holes.

According to Burkle (1980), blasting with dip causes more backbreak, less toe, smooth floor and lower muckpile profile, while blasting against dip creates less backbreak, more toe, rough floor and overhangs. Blasting against strike may result in unequal backbreak conditions, saw toothed floor and unfavourable orientation of face increasing the secondary blasting.

Singh and Sastry (1986a) from their tests on jointed models concluded that the formation of crater, and hence the fragmentation, was highly influenced by joint orientation (Figure 8).

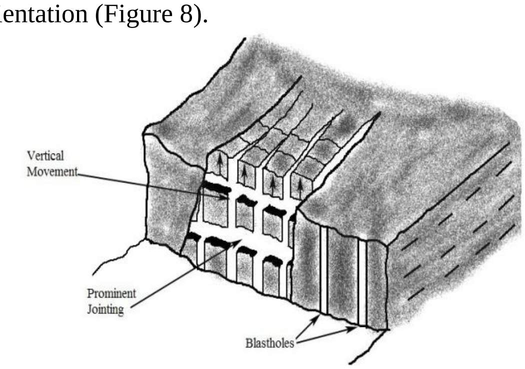


Figure 8. *Vertical movement of rock mass*

Singh et al. (1986) concluded from their study that both the mass and average fragment size of broken fragments were affected by joint orientation. Singh and Sastry (1986a) have done extensive studies on the effect of joints on Chunar sandstone models incorporating joints running parallel, perpendicular and angular to the face (Figure 9). They found that:

- Minimum yield results, when joints are running perpendicular to the face.
- Severe overbreak with uneven face formation results, when orientation of joints is perpendicular to face.
- More overbreaks occur in the condition with joints dipping into the face.
- Mass of fragments, average fragment size, mass surface area, fine and coarse fragmentation indices were significantly affected by orientation and direction of joints (at 5% level).

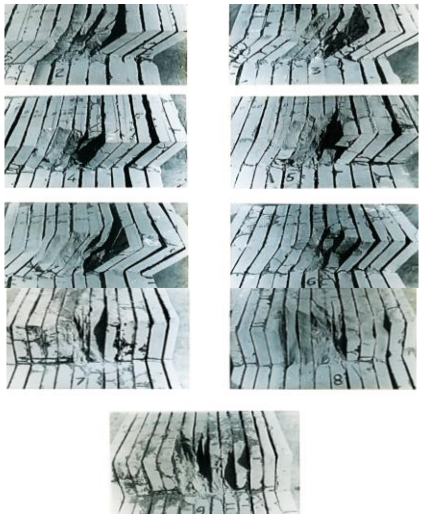


Figure 9. *Models with joints running perpendicular to free face*

2.6 Role of burden

Burden is one of the most critical geometric parameters of blasting. Burden is considered to have the greatest influence on blast results (Allsman, 1960; Singh and Sastry, 1987; Singh et al., 1986). For any given set of conditions, there exists a burden, which may be termed as "Critical Burden", where the strata gets fractured without displacement. According to Rustan et al. (1983) critical burden is an important factor when describing blastability nature of any rock. They recommended 40 to 90 % of critical burden as the maximum acceptable burden for satisfactory results. When the burden falls below its optimum value, then the effectiveness of strain energy increases and gaseous energy decreases. For very small burdens, strain wave fracturing occurs so rapidly in front of the blasthole that, much of the gaseous energy is lost to the atmosphere resulting in excessive throw of rock (fly rock).

In multi-row blasts, it is essential to keep the front row burden low in order to achieve proper burden relief and displacement, so that subsequent rows are blasted over smoothly, without any problems (Hagan, 1983). Otherwise, there is a possibility of encountering more fly rock and ground vibrations, in addition to undesired toe formation (Figure 10).

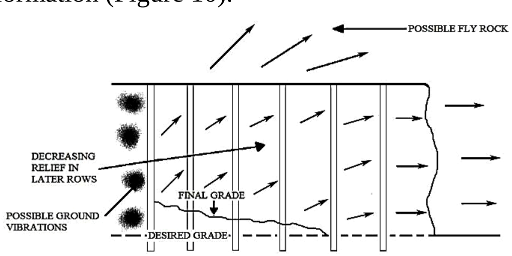


Figure 10. *Effect of insufficient burden relief*

2.7 Role of bench height

Bench height plays a vital role in influencing the blasting results. For each burden, there exists a maximum bench height to produce a full crater (Mason, 1973). The explosion generated strain in the rock alongside a charge increases as length to diameter ratio of charge increases in the approximate range of 0 to 20, and remains constant for >20. If it decreases to below 20, the optimum burden distances decreases. Therefore, when a charge becomes very short (the case with shallow benches), the burden needs to be reduced considerably. The breakage angle for a given burden increases with increase in bench height up to a certain point, beyond which no further significant change occurs (Atchison, 1968). As bench height increases, burden rock stiffness decreases.

2.8 Rock stiffness

Bending mechanism in rock blasting is not new and was recognised long back, even dating to 1898 (Daw and Daw, 1898). This mechanism was made popular subsequently by Ash (1973), Ash and Konya (1979) and Smith (1976). Stiffness principle and its use in blasting provide a guideline for the selection of an appropriate combination of burden, spacing and bench height.

Ash (1973) constructed an analogy between burden rock and structural beam to analyse the effect of bending on rock fragmentation. Burden on a blasthole was considered as thickness of beam, bench height as its length and average width of crater produced as its width. Cross section of the burden rock beam was defined by burden and spacing (Figure 11).

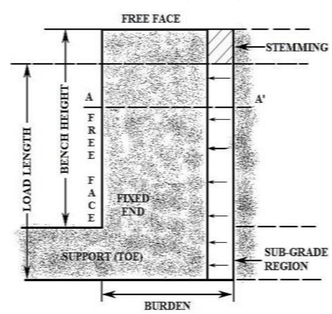


Figure 11. *Analogy between burden rock and a structural beam (Ash, 1973)*

Smith (1976) correlated the B, S and BH with stiffness of burden rock as:

$$K = \text{CE } Bx^{a}\, Sx^{b} / \text{BH}$$

where,

K = Stiffness of the burden rock, kg/cm
B = Burden dimension, cm
S = Spacing dimension, cm
BH = Bench height, cm
C = Constant depending on the shape and location of the area

a, b = Exponential constants depending on the shape of the area

E = Young's Modulus, kg/cm^{2}.

Breakage in shorter benches will be less than that in taller benches for same burdens, as stiffness decreases in the latter case (Figure 12). The cause of fly rock and collar overbreak is that the burden rock has become too stiff due to hole depth being too small or relief of burden being inadequate (Lundborg et al., 1975).

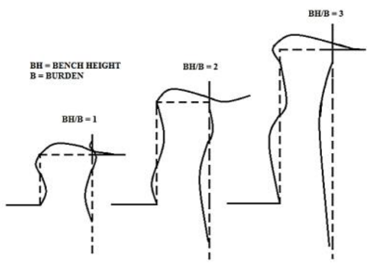


Figure 12. *Bending conditions in blasting (Smith, 1976)*

3. Investigations

3.1 Role of high speed videography in assessing blast Performance

An average blast is completed within seconds and is not possible to analyse the blasting process with naked eye. Ever since the advent of High Speed Video Cameras (HSC), it has been possible to view an entire blast in a sequence of frames, making analysis of the blasting process much effective. HSC with a capacity of 1000 FPS is capable of recording blasts, with the ability to capture one frame every millisecond in order to track down the delay performance as well. This enables the user to analyse every tiny movement happening in the blast. The HSC can be used to assess blast performance in terms of:

- Tracking of blasted rock mass.
- Tracking of burden rock movement.
- Checking the credibility of delays.
- Assessing the effectiveness of stemming by analysing stemming / gas ejection during blast.
- Determining the displacement of blasted rock mass.

3.2 Methodology

Studies were carried out with a series of bench blasts, in one coal and three limestone mines. All blasts were recorded with a S-Motion type High Speed Video Camera, AOS Technologies AG, Switzerland. High speed videos were analysed using ProAnalyst software to determine the displacement of burden rock and happenings in bench. An attempt was made to assess the influence of Bench Height to Burden ratio (BH/B) to identify important traits of the blasts and the role of jointing in blasting process. Details of blasts studied are given below (Tables 2 to 5).

Table 2. *Details of the blasts in Mine-1 (Coal Mine)*

Sl. No.	Parameter	Blast. 1	Blast. 2	Blast. 3	Blast. 4
1	Diameter of Blasthole (mm)	250	250	250	250
2	Burden (m)	6	6	7	7
3	Spacing (m)	8	8	9	9
4	Drilling Pattern	Staggered	Staggered	Staggered	Staggered
5	Depth of Blasthole (m)	11.1	11.1	15	14
6	Stemming (m)	5	3.5	7.5	05
7	No. of Rows	6	5	3	4
8	No. of Blastholes	25	30	19	70
9	Expl.Charge / Hole (kg)	276	290	350	410
10	Max.Charge / Delay (kg)	350	290	390	410
11	Total Charge / Blast (kg)	6918	8717	7415	22845
12	Initiation System	Shock tube	Shock tube	Shock tube	Shock tube
13	Rock Mass Movement (m/s)	53.0	111.1	108.3	67.1
14	BH/B Ratio	1.85	1.85	2.143	2

Table 3. *Details of the blasts in Mine-2 (Limestone Mine)*

Sl. No.	Parameter	Blast. 1	Blast. 2	Blast. 3	Blast. 4
1	Diameter of Blasthole (mm)	115	115	115	115
2	Burden (m)	2.7	2.7	2.5	2.7
3	Spacing (m)	3.7	3.7	3.5	3.7
4	Drilling Pattern	Square	Square	Square	Square
5	Depth of Blasthole (m)	10.0	10.5	8.0	10.0
6	Stemming (m)	2.5	2.75	3.0	3.25
7	No. of Rows	2	2	2	5
8	No. of Blastholes	17	14	16	14
9	Expl. Charge / Hole (kg)	66.17	76.78	48.43	66
10	Max. Charge / Delay (kg)	463.19	307.12	242.15	264
11	Total Charge / Blast (kg)	1125	1075	775	925
12	Initiation System	Shock tube	Shock tube	Shock tube	Shock tube
13	Rock Mass Movement (m/s)	135.7	132.7, 136.9	70.5	108.4
14	BH/B Ratio	3.704	3.889	3.2	3.704

Table 4. *Details of the blasts in Mine-3 (Limestone Mine)*

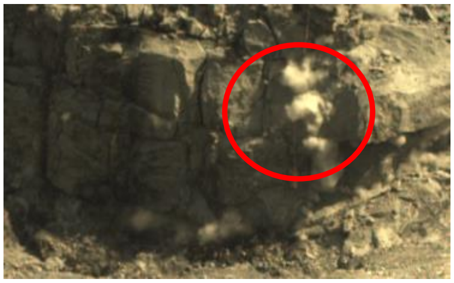
Sl. No.	Parameter	Blast. 1	Blast. 2	Blast. 3
1	Diameter of Blasthole (mm)	115	115	115
2	Burden (m)	2.7	2.7	2.7
3	Spacing (m)	3.0	3.0	3.2
4	Drilling Pattern	Rectangular	Rectangular	Staggered
5	Depth of Blasthole (m)	7.5	7.0	9.0
6	Stemming (m)	2	2	2
7	No. of Rows	3	2	4
8	No. of Blastholes	13	17	43
9	Expl. Charge / Hole (kg)	54	37.76	55.23
10	Max. Charge / Delay (kg)	215	294.45	994.14
11	Total Charge / Blast (kg)	700	642	2315
12	Initiation System	Shock tube	Shock tube	Shock tube
13	Roc Mass Movement (m/s)	66.2	110.9	77.2, 113.5
14	BH/B Ratio	2.778	2.223	3.148
Sl. No.	**Parameter**	**Blast. 4**	**Blast. 5**	**Blast. 6**
1	Diameter of Blasthole (mm)	115	115	115
2	Burden (m)	2.7	2.7	2.7
3	Spacing (m)	3.0	3.2	3.2
4	Drilling Pattern	Rectangular	Rectangular	Staggered
5	Depth of Blasthole (m)	9.5	9.5	8.0
6	Stemming (m)	2	2	2
7	No. of Rows	3	2	3
8	No. of Blastholes	10	20	18
9	Expl. Charge / Hole (kg)	62.24	72	44
10	Max. Charge / Delay (kg)	311.2	506	267
11	Total Charge / Blast (kg)	622.4	1145	3133
12	Initiation System	Shock tube	Shock tube	Shock tube
13	Rock Mass Movement (m/s)	122.8, 119.4	125.8	97.0
14	BH/B Ratio	3.334	3.519	2.778

Table 5. *Details of the blasts in Mine-4 (Limestone Mine)*

Sl. No.	Parameter	Blast. 1	Blast. 2	Blast. 3
1	Diameter of Blasthole (mm)	115	115	115
2	Burden (m)	2.5	2.5	2.5
3	Spacing (m)	3.0	3.0	3.0
4	Drilling Pattern	Square	Square	Square
5	Depth of Blasthole (m)	5.0	5.0	5.0
6	Stemming (m)	2.25	2.5	2.0
7	No. of Rows	3	2	4
8	No. of Blastholes	14	16	29
9	Expl. Charge / Hole (kg)	28.57	37.5	31.2
10	Max. Charge / Delay (kg)	114.28	150	156.03
11	Total Charge / Blast (kg)	400	600	905
12	Initiation System	Shock tube	Shock tube	Shock tube
13	Rock Mass Movement (m/s)	49.8, 114.0, 71.9	67.0	104.4
14	BH/B Ratio	2	2	2

4. Results and Analysis

Escape of gas energy was observed through major joints in the bench. Also the escape of gas energy through stemming zone was observed by high speed videos. Figure 13 shows the escape of gaseous energy through weakplanes and stemming zone in the bench, causing depletion of blasthole pressure. Both these reasons caused poor fragmentation.



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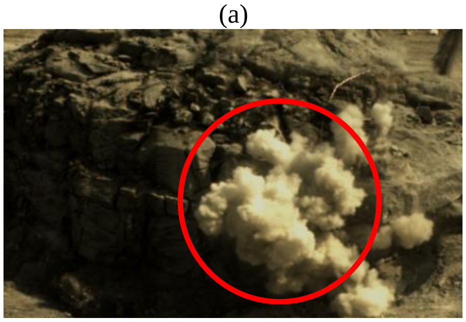

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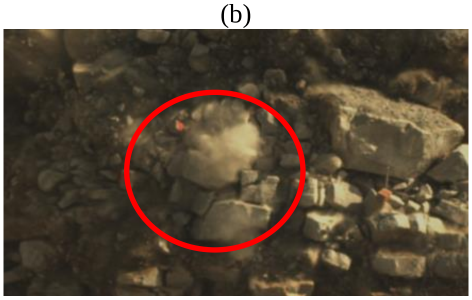

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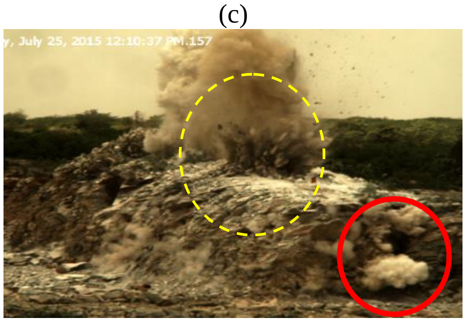

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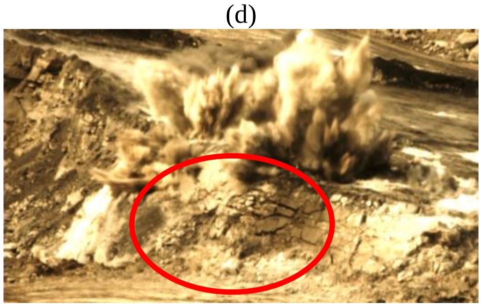
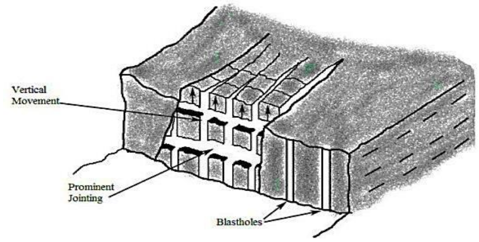


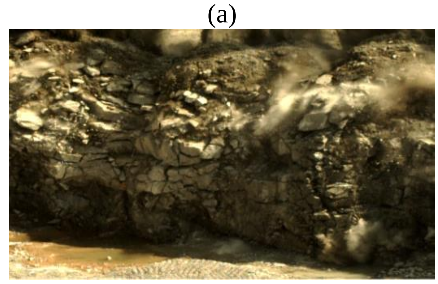
(e)

Figure 13. *Escape of gas energy through weakplanes & stemming zone*

Gas energy is found to escape through horizontal jointing and pushing the beds upwards (Figure 14).



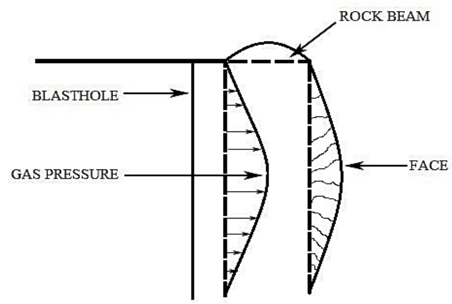
(a)



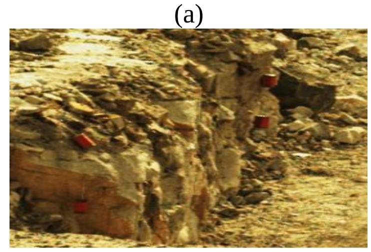
(b)

Figure 14. *Rock fragmentation affected by jointing (recorded by HSC)*

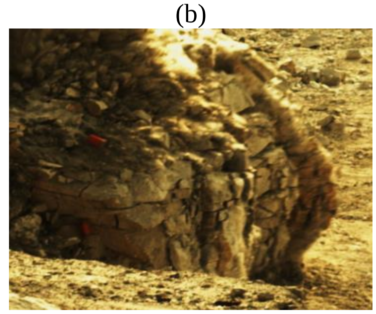
The HSC could clearly establish the beam bending mechanism as shown in Figure 15. Also the effect of joints on rock fragmentation could be observed as recorded in one of the blasts in a limestone mine.



(a)



(b)



(c)

Figure 15. *Flexural bending mechanism*

It could also be seen from Figure 15 that to a large extent, size of one side of the fragments resulting from the blast is controlled by joint spacing.

4.1 Burden rock velocity

Analysis of high speed videography study results revealed that the velocities of blasted rock are higher in case of limestone (7-13m/s) than in argillaceous sandstone overburden formation (6-10m/s) (Table 6). This is due to limestone being more compact and stronger than overburden sandstone, and transmission of strain waves is better in limestone formation.

Table 6. *Relationship between BH/B and average velocity of burden rock mass*

BH/B	BH/B Average	Burden Velocity (m/s)	Avg. Burden Velocity (m/s)
2		53.0	
2		111.1	
2		67.1	
2	**2**	49.8	79.8
2		114.0	
2		71.9	
2		67.0	
2		104.4	
2.14		108.3	
2.22	**2.5**	110.9	95.6
2.77		66.2	
2.77		97.0	
3.14		77.2	
3.14		113.5	
3.20	**3.2**	70.5	100.7
3.33		122.8	
3.33		119.4	
3.51		125.8	
3.70		135.7	
3.70	**3.7**	108.4	127.9
3.88		132.7	
3.88		136.9	

4.2 Effect of BH/B on delay timing

Initiation sequence in a blast is very important, and is a vital factor to be considered in blast design, since several initiation sequences radically alter effective burden and spacing during the blasting process. It also affects rock movement with respect to face and thereby influences the amount of rock shearing and design boundaries of blast pattern. A systematic release of explosive energy from one hole/row to the other is crucial in maintaining a continuous momentum required for inter-hole/row delay displacements.

It has been suggested by earlier researchers that the burden from first row of blastholes should be displaced by at least one third of the burden distance (1/3 B) before next row of blastholes is fired, for an efficient blast (Figure 16).

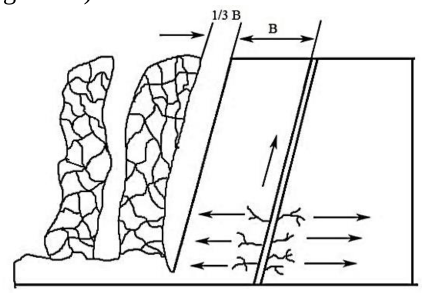
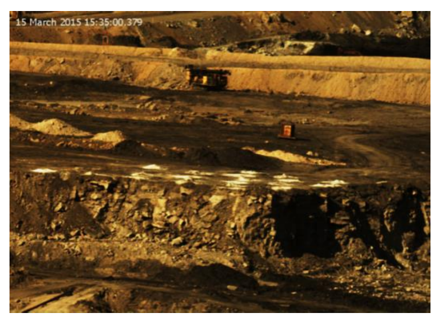


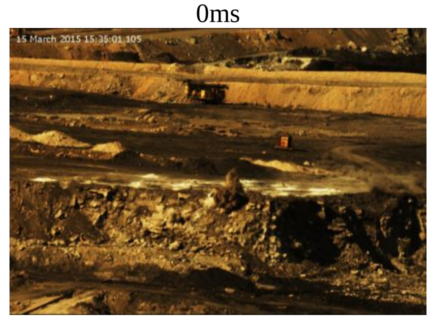
Figure 16. *Required burden movement before blasting of next row*

Burden rock velocity was calculated for different conditions by tracking down the movement of burden rock mass. Figure 17 shows some sample screen shots of high speed videographs of some of the blasts recorded in different mines.

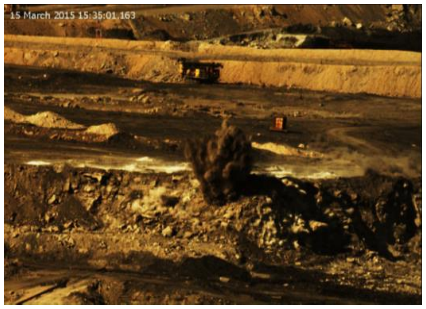
ProAnalyst software was used for tracking down the burden rock movement, for determining the velocity of rock mass. Some sample snap shots of the same are shown in Figure 18.



0ms



100ms

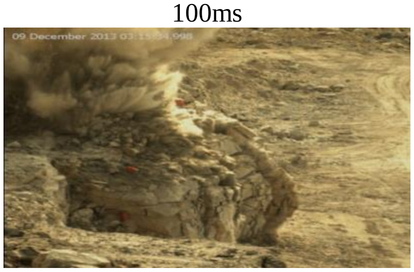


200ms

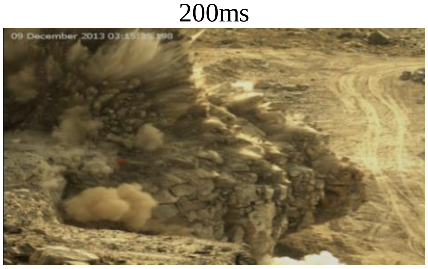
***a.** Screenshots of Blast-4 recorded in Mine-1*

0ms

100ms



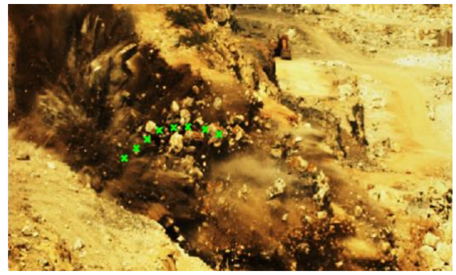
200ms



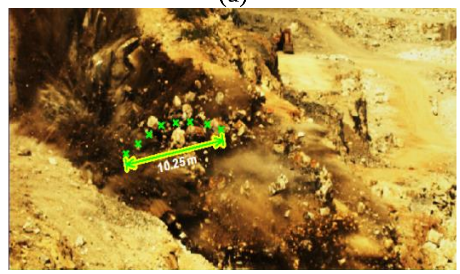
300ms

***b.** Screenshots of Blast-5 recorded in Mine-3*

***Figure 17.** Videographs of some blasts recorded by HSC in different mines*



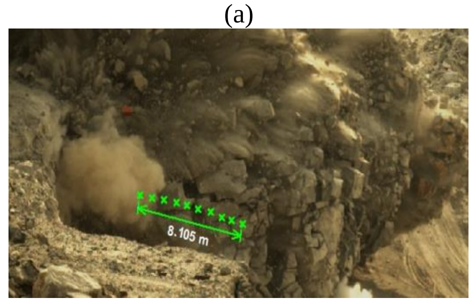
(a)



(b)

***a.** Tracking of rock mass movement (Blast-2 in Mine-3)*

(a)



(b)

***b.** Tracking of rock mass movement (Blast-3 in Mine-4)*

***Figure 18.** ProAnalyst analysis of some blasts recorded by HSC in different mines*

Based on the burden movement velocity, the minimum delay timing required between rows was analysed (Table 7). Study has shown that as BH/B ratio increasing, the required delay time per metre distance throw of burden rock mass is decreasing. For a BH/B ratio condition of two, the delay time required was determined as 12.5ms per metre, whereas with BH/B value of 3.75 the required delay reduced to 8ms per metre distance. This is due to the fact that as BH/B is increasing, the bench is becoming less stiff and more flexible resulting in faster movement of burden rock mass.

Table 7. *Delay time required between rows based on high speed videography data*

BH/B Ratio	Avg. Burden Velocity (m/s)	Delay Time Required (ms/m)
2.0	79.8	12.5
2.5	95.6	10.5
3.2	100.7	10.0
3.75	127.9	8.0

4. Conclusions

Following are the major conclusions drawn from the studies carried out using High Speed Video Camera in four different mines:

- High speed video camera is an excellent tool for analysing the blast results and designing efficient blasts.
- High speed video of the blasts provides clear information about weak zones in the bench being blasted from where escape of gas energy is taking place. Based on this the necessary zones of stemming decks could be finalised.
- For small benches, the velocity of blasted rock mass is slower. This is due to the increased stiffness of short benches.
- Conversely, for the taller benches, the velocities of blasted rock pieces have been recorded to be higher, as benches are becoming more flexible.
- In taller benches, the delay between holes/rows could be 8ms per metre distance. In case of shorter benches the delay time required is about 12ms per metre distance. According to the burden/spacing provided, the necessary delay timing may be adopted.

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