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Effective Fragmentation and Flyrock Control Strategies at Quarries

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Abstract: This paper presents the effective fragmentation and flyrock control strategies that could be applied at quarries to improve the productivity and safety. Fragmentation measurement and modelling as well as a comprehensive drill and blast audit are essential for improving the fragmentation. Face profiling and bore tracking are good tools to manage the "as-designed" and "as-drilled" conditions to get reasonable fragmentation from face burden zone and minimise the flyrock risk. In general, a large scatter in fragmentation data was observed at sites and the causes should be analysed by a detailed root-cause analysis technique. Two case studies were presented in this paper showing some of the effective fragmentation strategies. Finally, a flyrock model was shown to determine the safe blast exclusion zone for the mining equipment and personnel. Some key guidelines were suggested to minimise the occurrence of the flyrock.

Introduction

Drill and blast is understood to be the first stage of the comminution and known to affect the downstream processes (load and haul, crushing and grinding). Investigations by several researchers to date have shown that all the processes in the mine to mill value chain are inter-dependent and the results of the upstream mining processes (especially blast results such as fragmentation, muckpile shape and movement, rock damage) have a significant impact on the efficiency of downstream processes, especially crushing and grinding (Eloranta,1995; McKee et al., 1995; Kojovic et al., 1998; Kanchibotla et al., 1998; Simkus and Dance 1998; Scott et al., 1998; Kanchibotla et al., 1999; Valery et al., 1999, Valery et al., 2004; Dance et al., 2006, Esen et al., 2007; Valery et al., 2007; Kanchibotla and Valery, 2010).

The aggregates industry appears to be a perfect case for Mine-to-Mill optimization because its main focus is particle size reduction through blasting and crushing. Since, a typical crushed stone quarry can consume between 1.7 – 2.2 kWh/t, over 2.5 billion kWh of electrical energy is consumed per year by crushed stone production in the US. Clearly, if there is a segment of the mining industry, where energy saving research can have an impact, it is the aggregates industry (Adel et al., 2006).

As shown in Table 1, energy used in crushing is approximately 9 times more than drill and blast, which indicates that more efficient size reduction should occur at earlier step (blasting) to minimize the total energy consumption.

Table 1. Energy consumption at an aggregate quarry.

	Specific energy, kWh/t	Energy factor
Blasting	0.1-0.3	1
Load and haul	0.2-0.5	2
Crushing	1.0-2.0	9
Cement grinding	40-60	286

Fragmentation size distribution affects the crushing circuit in many ways. Coarse fragments and oversize present in the muckpile will reduce the primary crusher throughout and will lead to downtime to clear crusher bridging. The maximum feed size should be no greater than 80% of the crusher feed opening. Poor fragmentation will also increase the load to the secondary and tertiary crushing stages, because there will be less undersize to bypass these stages. This will affect productivity and energy consumption. The other effect of blasting is the production of the fractures that are produced within the rock fragments. There is substantial evidence that such cracking is produced (Nielsen and Kristiansen, 1996; Katsabanis et al. 2003). The effect of internal fractures is to soften the fragments, making them easier to break. Thus, the use of greater energy input in the blasting operation can often be less costly than expending energy downstream. This has benefits to productivity, energy expenditure and to the wear of crusher liners.

With the help of the improved blast outcomes at the aggregate quarries, the sites have reduced rock breaker hours due to less oversize, improved loader productivity, increased crusher throughout, less power draw and downtime at the crushers. A few examples are given below.

- A drill and optimization study at Linwood Quarry by Martin (2012) showed that a) 26% reduction in oversize material, b) 28% reduction in fines material, c) drill and blast savings of up to 22%, including oversize costs.
- Lawrance et al. (2009) carried out a mine-to-mill type project at a quarry in the US and they achieved impressive cost savings and increase in plant tonnage throughput: Crusher throughput resulting from all the validation blasts increased by at least 28%. In spite of a 28% increase in drilling and blasting cost, the standard cost model for the

project showed: a 10% to 27% increase in crusher plant capacity over a baseline of 373 tons per hour (TPH) to an average of 475 TPH. (A 102 TPH positive shift in capacity), a 17% to 31% reduction in net total cost per ton when scalping and even without scalping an 8.8% reduction in the net cost per ton.

- Chavez et al. (2007) recorded about 30% increase in the primary crusher throughput and improvements in total cycle time in load and haul.

- JKMRC's mine-to-mill project at Pittsboro quarry (Adel et al., 2006) in the USA achieved a) throughput at the primary crushing stage increase. The ramp-up was 9.5% (to 1035 tonnes per hour) for material, Luck Stone was producing for itself from dacite tuffs and basalt and 14.6% (to 965 tph) for andesite material, the company was crushing for the quarry owner, 3M.

- Elliott et al. (1999) conducted a fragmentation study at Lafarge Exshaw operation, which resulted in 15.6% increase in the crusher throughput and 30% reduction in the power draw.

- Rock breakage is more cost effective using explosives than a hydraulic breaker and explosive consumption costs have been increased at the quarry to affect an overall reduction in the costs of quarrying. Although powder factor increased approximately 23%, overall cost (drill and blast and rock breaking) dropped by approximately 7% (Cox and Cotton, 1995). This study did not quantify the benefits in crushing.

Cement production is an energy intensive process. It consumes 2% of the global primary energy and 5% of the total global industrial energy. Grinding is a high-cost operation consuming approximately 60% of the total electrical energy expenditure in a typical cement plant and 40% of this energy is for raw material grinding (Fujimoto,1993; Benzer, 2005). Therefore, fragmentation of raw materials limestone etc. fed into the mills is crucial for reducing the energy consumed in raw material grinding stage. Table 1 shows that specific energy consumption is very low at blasting and crushing stages. As discussed earlier, cement grinding is a very energy intensive process and size reduction should occur as much as possible prior to the raw material grinding.

This paper presents the tools and methodologies followed by the author used in a drill and blast study to control the fragmentation and flyrock. Case studies are presented to demonstrate the application.

Fragmentatıon Measurement

There are numerous image processing softwares (Split-Desktop, WipFrag, FragScan, PortaMetrics etc) which are commercially available. For manual systems, usually 10-20 pictures should be sufficient to adequaltely describe the fragmentation size distribution from a blast. Pictures should be taken at different shift breaks, whilst blasted material is being excavated to accurately represent the fragmentation from inner and outer parts of the blast. Figure 1 shows an example of a muckpile image and the delineated picture used by the image analysis software to determine the particle size distribution.

Fig. 1 Original (top) and processed (bottom) image for fragmentation analysis.

Fragmentatıon Modelıng

Blasting community has been widely using the fragmentation model developed by Cunningham (1983) which was later revised a few times (Cunningham, 1987; 2005). Due to the Kuz-Ram model's poor ability to describe the fines, the two Component Model (Djordjevic, 1999), the Crush Zone Model (Kanchibotla et al., 1999) and Onederra and Esen's (2004) model were developed at the JKMRC in Australia. All combined two Rosin-Rammler distributions or components, one for the coarse part of the curve and one for the fines. Onederra and Esen (2004) showed that the Kuz-Ram model is not able to satisfactorily predict the complete size distribution of fragments, particularly in the fine and intermediate size fractions. The model was later updated (Esen, 2013) using Swebrec function (Ouchterlony, 2005). Figure 2 shows a calibrated model at a gold mine, where a partial sieving data is available for a muckpile.

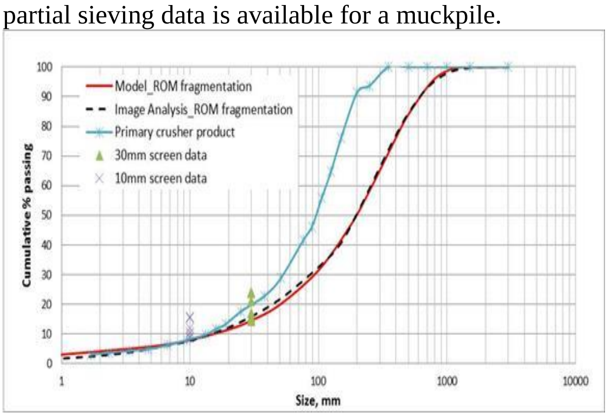


Fig. 2 Comparison of the sieve data at 10 and 30 mm with the fragmentation model.

Having carried out the image analysis to determine the size distribution of the blasted muckpile, the fragmentation model was calibrated using the measured fragmentation data. Sieving was carried out on-site and the sieve sizes were 10 mm and 30 mm (Fig. 2). It is shown that results of the fragmentation model compared well with measured data (Esen, 2013).

Figure 3 shows another example of the validation, which shows a good agreement between sieved data and model prediction at Bararp Quarry in Sweden.

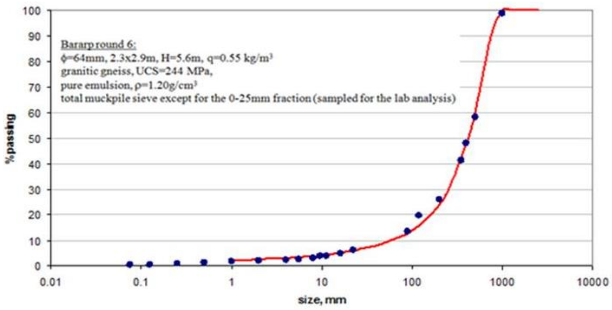


Fig. 3 Bararp Quarry fragmentation data – experimental vs model fit.

Qualıty Control at Bench

A good on-bench drill and blast audit can show how well the blast is implemented and show the detailed analysis of the hole depth (backfill/re-drills), hole collar deviations (deviations in burden and spacing), stemming (material type, size, length), priming quality, bulk explosive performance, initiation control (selection of delay times and burden relief), bench preparation and a general overview of the drill and blast process. The audit process can help understand the issues in the implementation. Figures 4-5 show two sites with poor and good drill control. Drill tolerance is 0.5 m for both sites. Example 1 has almost half of the blastholes out of tolerance limit whereas, example 2 is a much better site (approximately 20% out of tolerance). These analyses should be extended for hole length and stemming length. For quarries, face profiling and boretracking are key tools to manage the face burdens and hole deviations. Their use also minimizes the airblast and flyrock risks.

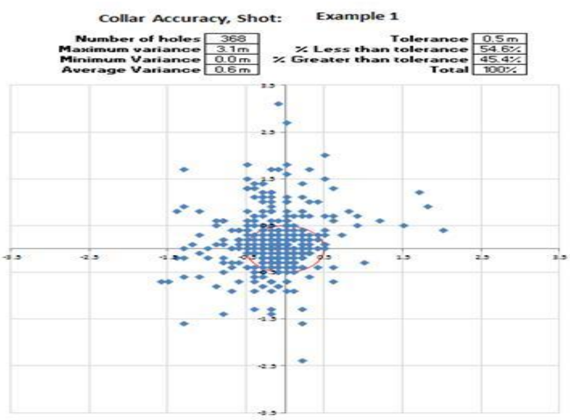


Fig. 4 Hole collar accuracy for Example 1.

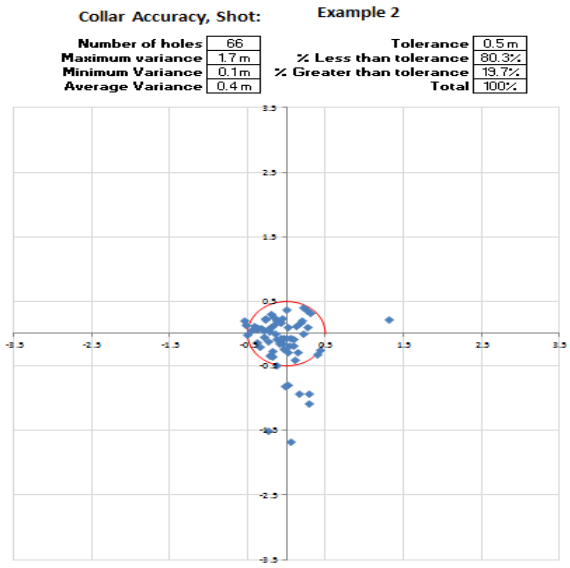


Fig. 5 Hole collar accuracy for Example 2.

Root Cause Analysıs for the Varıabılıty ın the Fragmentation Data

Figure 6 shows an example of fragmentation data obtained from a mine site. 80% passing size (F80) was chosen in this figure. It is shown that there is a significant variability in the data and coarse sizes are clearly seen (>300mm). So, what causes such a large scatter? The answer lies in the root cause analysis, which should be carried out by evaluating the QA/QC data, rock data (strength and structure) and blast design parameters using the fragmentation model. Two case studies were presented as examples.

Case Study 1

Figure 7 a shows the results of the hole depth compliance from a site. Figure 7 b shows the energy distribution, which is an output given by JKSimBlast software. It shows the hot and cold spots (high and low explosive energy respectively, as represented by MJ/m^3). This figure is a critical one as, it identifies a few key issues on-site.

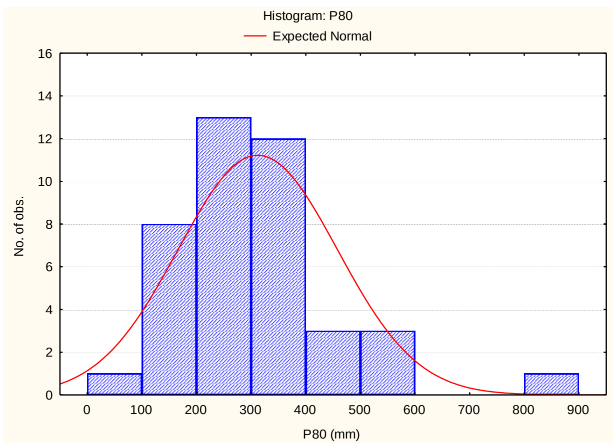
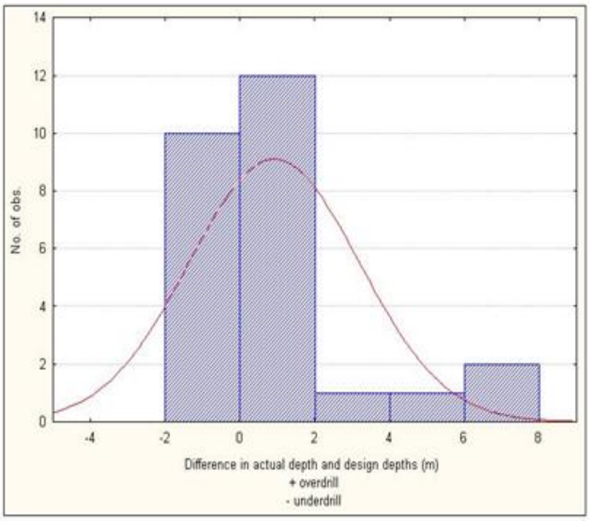
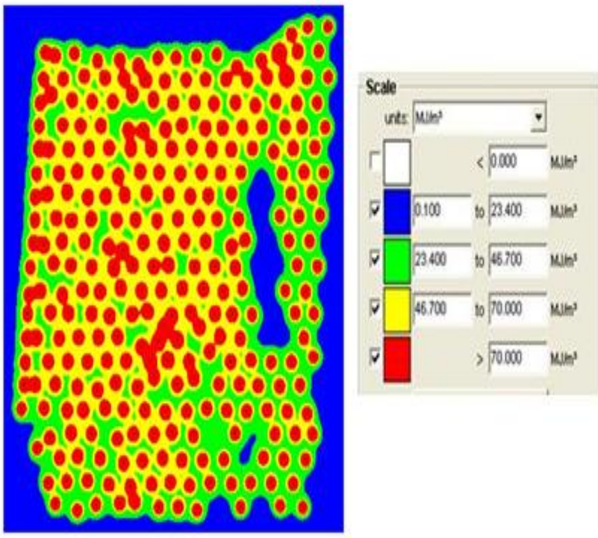


Fig. 6 Histogram of the fragmentation data (F 80) from an audited mine site.



(a)



(b)

Fig. 7a) Histogram of hole depth (actual – design) b) Explosive energy distribution showing the variability in the energy levels.

- Overdrilling and backfilling issues,
- Inadequate bench preparation and re-drill issues (some large areas represented by blue color indicating no blasthole),
- Large variations in burden and spacing causing non-uniform energy distribution,
- Poor blast shape.

The site implemented the recommendations to address above issues. In addtion, they improved the size distribution of the stemming material (from 15-40 mm to 5-20 mm for 165 mm blastholes). Table 2 indicates the major changes and the fragmentation results. The site experienced reduced crusher downtime and increased crusher throughput after this study.

Table 2. Changes in the drill and blast.

Better blast shapes		
Re-drills and backfills were carried out		
Improved stemming size		
Better hole collar location accuracy		
	Basecase	**Modified case**
Diameter, mm	165	165
Drill pattern	3.8x4.4	3.5x4.1
Powder factor, kg/m3	1.2	1.4
Stemming, m	3	3
Measured F80, mm	380	255

Case Study 2

A fragmentation study was carried out at an Australian Quarry due to the coarse fragmentation complaints at site. Figure 8 shows an image from an oversize pile. Rock type was basalt with in-situ block size of 0.5m. Table 3 summarizes the blast design parameters for the base case. Emulsion explosive with 30% ANFO at a density of 1.20 g/cm3 is used. Face profiling is carried out to manage the blasthole's locations and their angles to achieve face burden of 3.3-3.8 m. Boretracker is used to manage the issues caused by drill deviations.

Fig. 8 Oversize piled at a separate stockpile. Scale is 1m.

Table 3. Blast design parameters.

Dia,mm	89
Hole length, m	10.3
Bench height, m	9.3
Hole angle	10
BxS, m	2.7*3 (rectangular pattern)
Face burden, m	3.6
Face burden range, m	3.3-3.8
Stemming length, m	2.2
Number of rows	4
Control row, ms	42
Echelon row, ms	25
Face row powder factor, kg/m3	0.61
Inner row powder factor, kg/m3	0.81

Numerous pictures were acquired from an oversize pile and from the blasted muckpile. Measured F80 (80% passing size) and top sizes were 293mm and 580mm, respectively for the production blasts. Top size was 960 mm for the oversize pile (Fig. 9). The site was happy with the fragmentation obtained from the inner rows. However, the causes of oversize which were present in the stockpile needed to be identified and minimized.

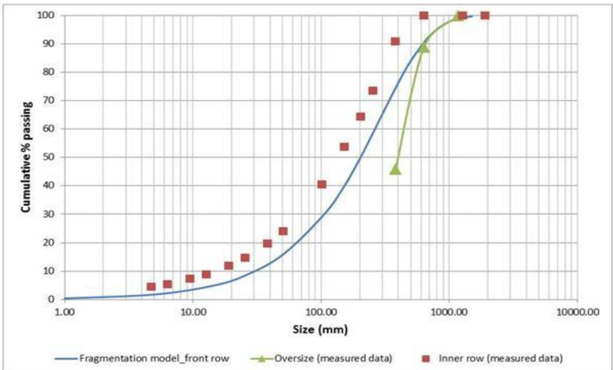


Fig. 9 Measured fragmentation data and fragmentation model.

Fragmentation model was calibrated with the coarse size matching to the measured oversize data. Two cases were run with the model as shown in Table 4. Alternative cases had different front row burden, spacing and stemming length values. As shown in Table 5, Case 2 had a top size around 700 mm and had a similar fragmentation, when compared to the inner rows. The site adopted Case 2 with improved fragmentation outcomes. Staggered drill pattern was chosen for better energy distribution.

Table 4. Base case and two alternative cases.

	Front burden, m	Spacing, m	Stemming, m
Base Case	3.6	3	2.2
Case 1	3	3	2
Case 2	2.7	2.8	1.8

Table 5. Fragmentation results.

	F80,mm	Top size, mm
Base Case	443	1177
Case 1	359	851
Case 2	303	682

Flyrock Control and Determınıng the Blast Exclusıon Zone

Flyrock

Flyrock can be defined as the rock fragments, which were projected beyond the clearance zone. The clerance zone is the zone around a blast beyond which there should be no risk to personnel from flying rock fragments and beyond, which the blaster must evacuate all personnel prior to firing the blast (Stiehr, 2011).

Flyrock is one of the most blast-related incidents seen at mine sites. Some Australian examples are listed below.

During a quarry blast, flyrock was projected more than 500 metres onto the Pacific Highway. A rock of approximately 100 mm diameter was also projected onto a nearby property, where it caused damage to a shed and parked vehicle.

- A rock was thrown 1300 metres from a blast consisting of 89 mm diameter blastholes.
- Flyrocks resulting from a trim blast at a gold mine caused significant damage to the four drills and one excavator, which parked less than 150 m from the blast.
- At a gold mine, one of the blastholes caused flyrock hitting and braking the window on a drill rig, which was located 181m from the blast.
- A quarry blast had thrown material a maximum of 170 meters and striking the main office, which was 150 meters from the blast and caused damage to buildings.
- A shotfirer was struck on the right side of his face by flyrock after a toe was blasted at a quarry and was

videoing the shot 75 metres from the blast area, whilst sheltering behind a steel hopper with another person.

Root Causes of Flyrock

Flyrock can be generated from a bench blast (either free-faced or buffered) b) oversize blasting. This paper deals with the flyrock generated by the bench blasts only.

The causes of flyrock in a bench blast are; Design faults: inappropriate face burden and stemming length, inappropriate stemming material selection (e.g. drill cuttings), poor choices of powder factor and initiation sequence.

Deviations in implementation: "as-drilled" face burden and final stemming length being less than design, blasthole deviation (not measured and/or incorrect loading for holes with significant deviations), explosive run-away into cavity (failing to detect this issue). Unforeseen geological conditions: cavities, weak seams, fault zone and broken zone in the stem zone or in the face burden area, etc.

Determining the Blast Exclusion Zone – A Case Study

There are two main flyrock models (Richards and Moore, 2004, McKenzie, 2009), which have been widely used in the industry. In this paper, a case study was presented using Richards and Moore's model.

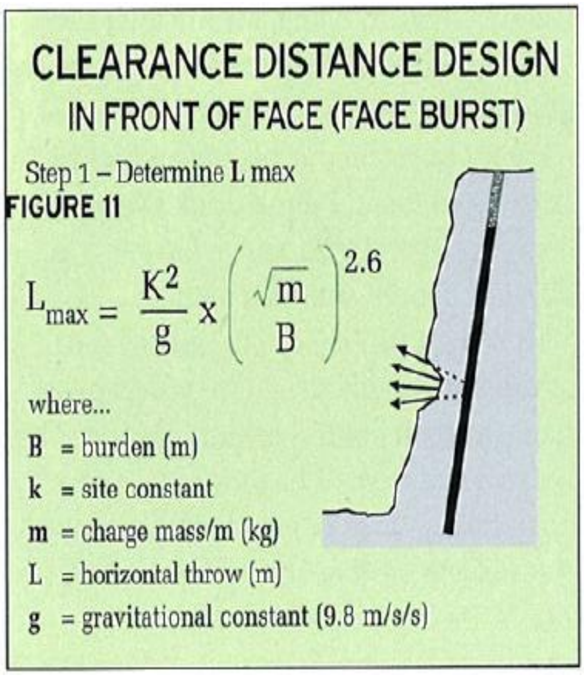
According to Richards and Moore (2004), there are three common sources of flyrock (Fig.10).

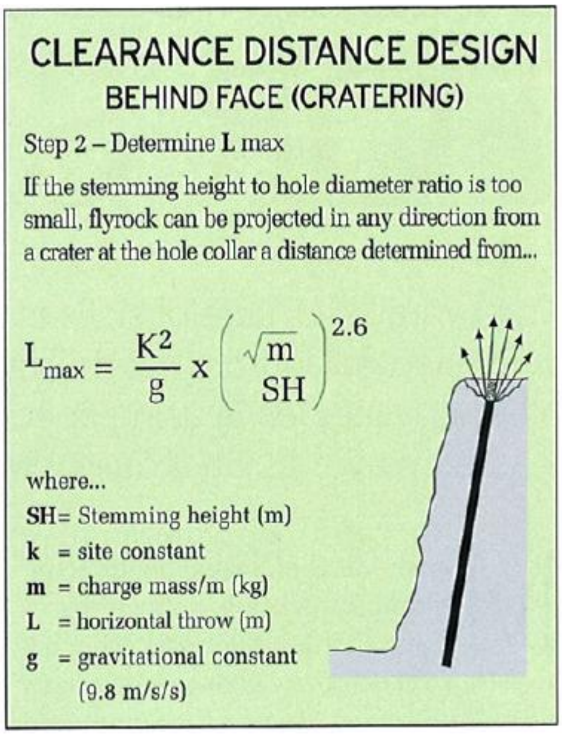
- The face of the blast, in which flyrock is generated through a 'face burst',
- The bench top, through a phenomenon known as 'cratering' and the stemming zone, where flyrock is generated through stemming ejection or 'rifling'.
- The stemming zinc, where flyroc is generated through stemming ejection or refiling.

Face burst occurs, when front row burdens are insufficient to contain the explosive energy. This mechanism can produce flyrock in front of the blast area. Stemming ejection (or rifling) occurs when stemming material is of poor quality or where the hole is not fully stemmed (e.g. hang ups). This mechanism can produce flyrock behind the blast area, depending on the angle of the blast hole.

Note that cratering calculations are removed from original model as cratering calculations are not representative of the blasts at this quarry. Cratering is not valid as stemming length is well above 20 times hole diameter and should not cause cratering effect. The equations shown in Figure 10 provide a tool, which can be used to predict the maximum flyrock distance likely to result from a blast, given the specific parameters of that blast. Based on this prediction, a safety factor is applied to give a minimum blast clearance distance. The safety factor applied for buildings and equipment is 2.0. For humans the safety factor is 4.0.

The site constant, K, as shown in Figure 10 accounts for the blasting response of the rock mass at a specific site. K takes a value between 13 and 27 depending on the observed blast outcomes at that site. The model can be "tuned" to an individual site's blasting conditions based on a history of measured blast outcomes and maximum rock movement.





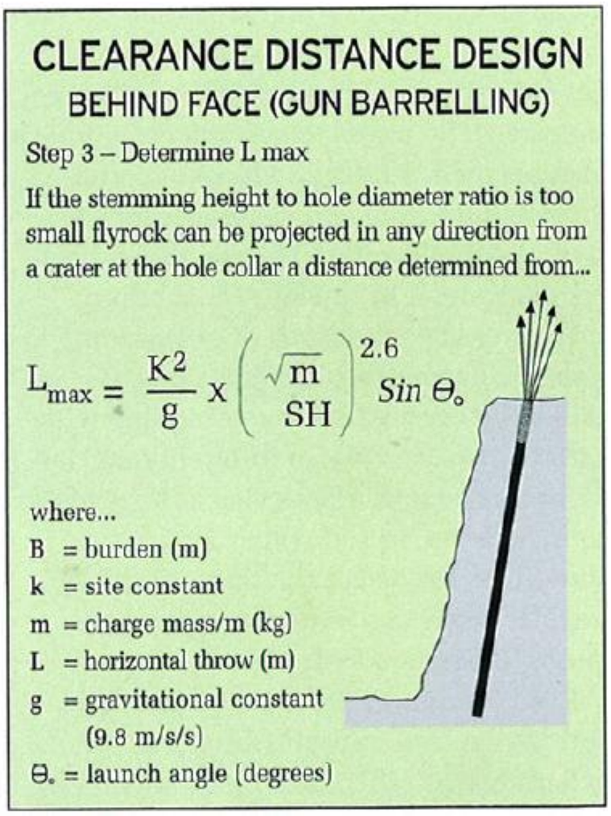


Fig. 10 Flyrock model (Richards and Moore, 2004).

A few survey points were marked on the ground around the blast to determine the horizontal distance of the flyrock. Videos of the blast and other blasts were analysed. Maximum horizontal flyrock distance is determined, as approximately 45 m and the maximum vertical distance is approximately 25 m.

Base case blast had 89 mm hole diameter, 10.6 m hole length, 2.2 m stemming length and 4 m face burden. The model constant is calibrated (K=21) to match the observed collar projection and maximum flyrock range of 45 m. Base case stemming lengths of 2.2 m at 89 mm pattern indicate a Scaled Depth of Burial of 1.41 using pumped emulsion with density of 1.20g/cm³ which should not cause significant flyrock distances, provided that stemming collar is not in a broken ground and stemming material is appropriate (free flowing ensuring bridging does not occur). Face holes are designed with minimum of 3.2 m face burden.

The calibrated model determines the blast exclusion zone for personnel as 193 m. It assumes that a) minimum of 2.2 m stemming is applied, b) bridging does not occur and appropriate crushed aggregate fills the stemming column c) minimum face burden is 4 m.

A sensitivity analysis (Appendix 1) was carried out by varying the K constant, stemming length and face burden to account for the natural variation that may exist in geology, face burden and loading. It is shown that medium (10%) variation in geology (K constant), face burden and loading (stemming length) increases the personnel exclusion zone distance to 277 m. Therefore, it is recommended to adopt 300 m as the exclusion zone distance for personnel. This would ensure that natural variation in geology, face burden and loading is accounted for the selection of the distance.

Based on the safety factor of 2 for the equipment, the nominal exclusion zone distance is calculated as 138 m considering the variation (medium: 10%) in the loading, face burden and stemming. Therefore, it is suggested to use a minimum of 150 m of clearance distance for the equipment and infrastructure around the quarry.

Suggestions for Managing Flyrock Issues at Quarries

All employees should be removed to a safe location away from the blast area during blasting. Blast exclusion zone calculations should be carried out by an external consultant.

All entrances to the blast area should be securely guarded to prevent inadvertent entry of employees or visitors. Pit plans showing the blast exclusions zones for equipment and personnel as well as guard positions should be prepared for each blast. Good communication is a key to a safe blasting operation.

Proper blast design and an effective blasting plan will reduce the chances for flyrock. Most flyrock incidents occur because a) the burden has not been checked b) appropriate stemming material and/or stemming length were not used. Laser profiling, with or without boretracking is a useful tool for checking burden.

Crushed aggregate with size of 1/10th of the hole diameter should be used. Stemming length calculations should be based on the scaled depth of burial given by Chiappetta et al. (1983) or Stiehr (2011).

In this paper, the author recommends a flyrock modelling and blast exlusion zone calculations to be carried out by an external blast consultant. In the absence of such reports, I recommend using below guidelines at quarries until a report is prepared and made available for use.

- For nonblast personnel 800 m in front of the shot and 400 m to the side and rear of the shot.
- Blast personnel should be positioned greater than 400 metres from the shot and not positioned in the direct line of fire and within retreat distance of a protective structure (i.e. fixed plant or blasting bell).
- No mobile plant is to be within 300 metres of the initiation point without signed site manager's approval.
- Where a blast is to occur within 100 metres of fixed plant an appropriate blasting specialist should be engaged to design and control the loading and firing process.

Conclusıon

Effective control strategies for fragmentation and flyrock are presented in this paper. Some of the key conclusions are as follows.

- Mine to mill type optimization studies can help improve the productivity of the quarries and reduce the total cost per ton significantly.
    - Some researchers showed that total drill and blast and oversize cost dropped by 7-22% with the drill and blast optimization study,
    - 10 to 30% increase in the primary crusher throughput.
    - Up to 30% decrease in power draw.
    - 17% to 31% reduction in net total cost per ton.
- Fragmentation measurements should be carried out on-site to understand the variability of the data and minimise the variability to provide a consistent feed to the crusher without oversize.
- Fragmentation model calibrated to the measured data offers significant benefits to the quarries as it presents alternative designs for better fragmentation.
- Blast auditing is crucial at any site as it identifies the issues with QA/QC, pit planning, drill and blast process as well as safety. It should be conducted at every site regularly.
- Safe blasting requires that rock throw be controlled to prevent danger from flyrock. Flyrock incidents have occurred in the past and investigations of these incidents commonly conclude that the flyrock was due to over charging and/or under confinement of the explosive charge.
- A practical flyrock model was presented in order to determine the safe blast exclusion zone for the mining equipment and personnel. Some key guidelines were suggested to minimise the occurrence of the flyrock.
- An external specialist should be engaged to carry out the blast exclusion zone calculations.

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Appendix 1 Sensitivity Analysis – Exclusion Zone Distance

	K	Exclusion zone distance (Personnel), m	Exclusion zone distance (Equipment), m
Minor - 5%	22	237	119
Medium - 10%	23	259	130
Major - 20%	25	306	153
	Stemming Length, m	Exclusion zone distance, m	Exclusion zone distance, m
Minor - 5%	2.1	244	122
Medium - 10%	2.0	277	138
Major - 20%	1.8	364	182
	Face burden, m	Exclusion zone distance, m	Exclusion zone distance, m
Minor - 5%	3.8	216	108
Medium - 10%	3.6	216	108
Major - 20%	3.2	216	108