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MARTINS CREEK QUARRY EXTENSION PROJECT BLASTING AND VIBRATION FOR INCLUSION IN EIS REPORT – NOVEMBER 2015

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Approved by:	A Smith
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EXECUTIVE SUMMARY

This report is aimed at providing information about blasting that is suitable for inclusion in an EIS for the proposed Martins Creek Extension Project and covers the following:

1. A standard drill and blast design for the quarry based on an 11m bench height which is considered appropriate for the rock types to be blasted based on past blast outcomes
2. Previous blast induced ground vibration results confirm no blast induced ground vibration licence limit exceedances using this design over the past 32 months since Daracon took ownership of the quarry with the highest recorded vibration being only 3.72mm/s PPV which is well under the likely lower limit environmental limit of 5mm/s PPV as per ANZEEC Guidelines 1990 and AS2187.2-2006. Martins Creek Quarry has a proven track record of meeting the ANZEEC Guidelines on blast vibration
3. The blast induced air overpressure results over the past 32 months confirm that the highest air overpressure result of 116.3dBL with this being the only result over 115dBL or a 1.47% lower environmental license limit exceedance rate which is well below the 5% allowed. A number of initiatives aimed at reducing the air overpressure have been implemented with no blast induced air overpressure over 115dBL since that blast. Martins Creek Quarry have a proven track record of meeting the blast induced air overpressure ANZEEC Guidelines 1990 limits
4. The likely blast induced ground vibration and air overpressure limits based on the ANZEEC Guideline 1990 is likely to be for:
    a. Ground vibration:
        i. Less than or equal to 5mm/s PPV achieved for at least 95% of the blasts in a year
        ii. Maximum of 5% of the blasts can have a PPV of greater than 5mm/s PVV but less than or equal to 10mm/s per annum and
        iii. Under no circumstances shall a blast generate a PPV of greater than 10mm/s
    b. Air overpressure:
        i. Less than or equal to 115dBL achieved for at least 95% of a blasts in a year
        ii. Maximum of 5% of the blasts can have an air overpressure greater than 115dBL but less than or equal to 120dBL and
        iii. Under no circumstances shall a blast exceed 120dBL.
5. The hours of blasting shall be between 9am and 5pm Monday to Saturday inclusive with no blasting on Sundays or public holidays. The blast frequency will be less than or equal to 50 times per annum but the aim is to minimise the blasts by firing a nominal 15000bcm blast size. Blasting will generally be typical opencut free face blasting, well sorted crushed angular aggregate of a nominal size of 9mm (7mm to 11mm) will be used as stemming to minimise air overpressure and a good quality licensed drill and blast provider with a proven track record in quarry and construction blasting with suitable systems and procedures vetted by the RMS for close in blasting will be used. Blasting will be undertaken to meet all legislative (NSW Explosives Act 2003), regulatory (NSW Explosives Regulation 2013), Australian Standards (specifically AS2187.2-2006) and Codes of Practice as a minimum including ANZEEC Guidelines 1990 and licence conditions with respect to blasting. There will be no storage of explosives or explosive precursors on site as they will be mobilised to each blast with any excess returned to the drill and blast contractors or the explosive supplier's premises.
6. The impacts on people will be minimal given the low level of vibration and air overpressure

likely to be generated at the closest residences. Animals are unlikely to be unduly affected due to the small number of blasts and the low levels of vibration and air overpressure. Buildings and natural features will not be affected due the low level of vibrations and air overpressure which are an order of magnitude below which the onset of damage occurs
7. Techniques to mitigate ground vibration, blast induced air overpressure, flyrock and fumes are being implemented in the Martins Creek Quarry and these are presented in the report
8. Wind can easily produce air overpressures in excess of blasting licence conditions and a low cost easy to implement technique is detailed in the report to minimise this risk which the Martins Creek Quarry has implemented.

1.0 INTRODUCTION

The Martins Creek Quarry is undertaking an extension project that is proposing to:

- Extract up to 1.5 million tonnes of hard rock material per annum;
- Expanding into new extraction areas and clearing approximately 36.8 hectares of vegetation;
- Increasing hours of operation for quarrying to 6am – 6pm Monday to Saturday, processing to 6am – 10pm (Monday to Saturday), mixing and binding to 4.30am – 10pm (Monday to Friday) and 4.30am – 6pm (Saturdays), stockpiling, loading and dispatch of road transport to 5.30am – 7pm (Monday to Saturday) and train loading to 24 hours per day, 7 days per week
- Consolidating existing operations and approvals; and
- Rehabilitating the site.

This report is aimed at providing information about blasting that is suitable for inclusion into an EIS and covers the following:

1. Generate an appropriate drill and blast design applicable for the site
2. Undertake ground vibration estimation based on an appropriate drill and blast design and current results over the last 2 years 8 months
3. Undertake blast induced air overpressure estimations based on an appropriate drill and blast design and current actual results over the last 2 years 8 months
4. Detail the likely blast induced ground vibration and air overpressure limits based on the Technical basis for guidelines to minimise annoyance due to blasting overpressure and ground vibration (ANZEEC, 1990)
5. Detail the proposed hours, the frequency and the methods of blasting
6. Provide an assessment of the likely blasting impacts on people, animals, buildings and significant natural features having regard to the relevant ANZEEC Guideline 1990
7. Detail easy to implement techniques to mitigate ground vibration and blast induced air overpressure including those associated with inversions
8. Detail the easy to implement risk mitigation required to avoid false blast induced air over-

pressure readings due to wind.

2.0 INFORMATION PROVIDED OR OBTAINED

The following was provided or obtained:

1. Quarry Plans for 5, 10, 15, 20 and 25 years
2. ANZECC – Technical Basis for guidelines to Minimise Annoyance Due to Blasting Over-pressure and Ground Vibration 1990
3. Assessing Vibration: A Technical Guideline – Department of Environment and Conservation NSW 2006
4. Google Maps of the general area of the proposed quarry as well as the closest residences
5. Maps of the area
6. The vibration and air overpressure results for the last 2 years and 8 months
7. Indicative blast designs
8. Current techniques in use to minimize air overpressure and flyrock
9. Several emails.

3.0 PROPOSED DRILL AND BLAST DESIGNS

The bench height in the current quarry at Martins Creek is a nominal 10m while the extension is based on 12m bench height. It is likely that the actual bench height will be in the region of 11m which the blast design below is based on. The design for the 89mm diameter holes is for the main body of the blast while the 76mm diameter holes are for the front or face of the blast. The smaller diameter holes are required for control of flyrock as well as air overpressure.

Main Body of the Shot:

1. Hole diameter = 89mm
2. Hole angle - vertical
3. Burden = 2.8m (31.4 hole diameters – average rock is 27 hole diameters)
4. Spacing = 3.2m (Burden to spacing ratio = 1.143 which is close to an equilateral triangle pattern which is 1.15 which is the best for explosive energy distribution = better blast outcomes as uniform explosive energy distribution in a horizontal sense)
5. Stem length = 2.5m – 2.7m depending on how close to residences or critical structures – 2.5m is 28.1 hole diameters while 2.7m is 30.3 hole diameters. 30 hole diameters are used for careful blasting to minimise flyrock or for very soft rock. Both stem lengths are sufficient to minimise explosive energy being vented from the top of the shot reducing flyrock and air overpressure
6. Subdrill = 0.5m = 5.6 hole diameters with 8 hole diameters being average rock. The smaller subdrill is to reduce vibration as this is the most confined area of the blast as well as charge weight which also minimises blast induced vibration
7. Hole length = Bench Height + subdrill = 11 + 0.5m = 11.5m
8. Charge length = Hole length – stem length = 11.5 – 2.6 (average stem length = 8.9m
9. Bulk Explosive = Emulsion or watergel with average density of 1.12g/cc
10. Charge weight per metre watergel/emulsion = 7.2kg/m
11. Hole charge weight watergel/emulsion = 7.2 X 8.9 = 64kg
12. Powder Factor = 64/ (2.8 X 3.2 X 11) = 0.64kg/bcm (Energy Factor = 1.98Mj/bcm)

13. Vertical Energy Distribution = percentage of bench loaded with explosives = (11 – 2.6)/11 = 76.4% which is very good and thus the blast has the best chance of minimising oversize at the top of the shot
14. Burden stiffness ratio (BSR) = Bench Height/Burden = 11/2.8 = 3.9 which is excellent – this means the design is set up to break the rocks easily
15. The high vertical energy distribution plus very good BSR means that the blast outcomes will be very good given the reasonable powder factor of 0.65kg/bcm and energy factor which is nearly 2Mj/kg.

11m Bench Height with 76mm Face Hole Diameter:

1. Hole diameter = 76mm
2. Hole Angle = vertical
3. Burden = 2.5m (32.9 hole diameters – average rock is 27 hole diameters – this is a large burden designed to reduce flyrock and air overpressure by fully containing the explosive gases and slowing down the piston like effect of face movement that creates air overpressure)
4. Spacing = 2.8m (really 1.4m as the 76mm diameter holes are double stitched – half spacing with every second hole only loaded with 3m of toe charge and the rest of the hole fully stemmed)
5. Stem length = 2.5m = 32.9 hole diameters which is larger than used in the main body 89mm diameter holes in terms of hole diameters and is designed to minimise air overpressure and flyrock from the top of the shot
6. Subdrill = 0.5m = 6.6 hole diameters with 8 hole diameters being average rock - designed to minimise blast vibration as well as remove toe via the double stitching
7. Hole length = 11 + 0.5m = 11.5m
8. Charge length = 11.5m – Stem length – Intermediate stem deck = 11.5m – 2.5m – 1.5m = 7.5m but this is split between two explosive loads of 3.75m each. The main holes have two charges separated by 1.5m of intermediate stem deck composed of crushed angular aggregate. This is to reduce the charge and hence vibration but also the amount of face moving at any one time to reduce the area of face and the speed of the movement thereby minimising the risk of blast induced air overpressure.
9. Bulk Explosive = emulsion or watergel of 1.12g/cc average density
10. Charge weight per metre emulsion/watergel = 5.3kg/m
11. Charge weight emulsion = 7.5m X 5.3 = 38kg or two charges of 19kg with 15kg in the double stitched toe loaded holes
12. Powder Factor Emulsion/watergel = 0.69kg/bcm (Energy Factor = 2.13Mj/bcm with emulsion/watergel being 3.1Mj/kg)
13. Vertical Energy Distribution = percentage of bench loaded with explosives = (11 – 2.5)/11 = 77.3% which is very good
14. Burden stiffness ratio (BSR) = Bench Height/Burden = 111/2.5 = 4.5 which is excellent
15. The high vertical energy distribution plus excellent BSR means that the blast outcomes will be very good even with the two decks and good fragmentation should result in a loose easier to dig muckpile.

It should be noted that the back and the sides of the shot have longer stem lengths of between 3.2m to 3.4m to reduce back break and side break near the crest that can be the site of flyrock and air overpressure in subsequent blasts. It also reduces the potential for flyrock and air overpressure in the current blast.

4.0 GROUND VIBRATION ESTIMATION The generalised form of any blast induced ground vibration is:

$$V = K\left(\frac{R}{Q^{0.5}}\right)^{B}$$

where:

V = ground vibration as peak particle velocity (mm/s)

K = constant related to the site and rock properties

R = distance between the charge and the point of concern (m)

Q = maximum instantaneous charge weight (kg)

B = constant related to the site and rock properties (usually -1.6)

The K factor is a site constant that can vary generally from 400 through to greater than 8000. AS 2187.2-2006 gives a K factor of 1140 for free face average rock blasting but this is a 50% confidence limit equation while the equivalent 95% confidence limit equation has a K factor of about 2000. The B factor is generally -1.6. It is proposed to undertake free face blasting (the front of the blast has no rock in front of it) for the vast majority of blasts so a K factor of 2000 would normally be appropriate for this style of blasting at the Martins Creek Quarry. However, blasting has been undertaken for a significant amount of time since Daracon assumed ownership of the quarry and detailed blast monitoring results are available in Appendix 1 of this report of all blasts undertaken since the start of 2013. This data can be used to determine if:

1. There is a current problem with blast induced ground vibration and
2. If there is going to be an issue with blast induced ground vibration in the future

Appendix 1 details 68 blast results over the 32-month period and the highest ground vibration recorded during this time is 3.72mm/s PPV at the Patterson Valley Estate monitor closely followed by a 3.63mm/s PPV for the Gully residence monitor at 336 Dungog Road and 1.78mm/s PPV at the back gate. It is the back gate monitor that provides the best indication of how the residences at Station Street are likely to be affected by blast induced ground vibration given that this is the closest monitor and is the direction of Station Street from the Quarry. None of these even approach the lower level Licence limit of blast induced ground vibration which is less than or equal to 5mm/s PPV for 95% of the time with 5% of the blasts allowed to have ground vibrations of greater than 5mm/s PPV but less than or equal to 10mm/s PPV. No blast induced vibration shall exceed a PPV of 10mm/s. The maximum ground vibration recorded is less than 75% of the 5mm/s PPV indicating that the systems, procedures and methodologies used to manage blast induced ground vibration are working extremely well and the Martin’s Creek Quarry has the capability of successfully managing this issue into the future by simply doing what has always been done by continuing to implement the systems, procedures and methods current used to manage blast induced vibration.

A K Factor has been developed for the Martins Creek Quarry which is 983. This report shall round this to 1000 and refer to this as the 50% confidence limit K Factor. The 99% confidence limit K Factor is 2000 as there has only been 1 K Factor above this. These K Factors have been generated based on the standard vibration equation presented at the start of this section and appearing in AS2187.2. so the site vibration law for Martins Creek quarry at 99% confidence limit using Nonel initiation is:

$PPV = 2000Q^{0.8}/R^{1.6}$

Blasting approaches within 270m of residences in Station Street which is the closest that blasting approaches any residences in any direction. The above equation using a hole charge weight of 65kg for an 89mm diameter hole gives an estimated PPV of 7.3mm/s which is too high. The alternative is to use a 76mm diameter pattern design as this provides a maximum hole charge of 48kg which gives an estimated ground vibration of 5.7mm/s which is also too high. It is therefore proposed to use two decks of explosives in 76mm diameter holes to reduce the blast induced vibration further so the maximum charge is 20kg. This also reduces the risk of flyrock and current practice is to use 76mm diameter blastholes near infrastructure or residences. A 20kg hole charge produces 2.9mm/s PPV which is acceptable. 76mm diameter holes using 2 decked charges will need to be used from distances from 270mm to 300m from residences. 89mm diameter patterns using two decks will need to be used from 301m to 340m then fully charged 76mm diameter holes from 341m to 400m and fully charged 89mm diameter holes at distances greater than 400m from residences. This will obviously be refined based on actual blast results but the aim is to always achieve blast induced vibration levels of less than 5mm/s PPV and this is why the 99% confidence limit equation is being used.

The surface initiation will always be designed to achieve 1 deck or 1 hole of charge blasting depending on the distance from residences.

Achieving the vibration limits for the residences automatically means that the vibration limits on the adjacent infrastructure like, roads (300mm/s PPV), bridges (10mm/s PPV), powerlines (100mm/s PPV) and the rail line (300mm/s PPV) are achieved.

In conclusion the blast induced vibrations are highly likely to be well under the 5mm/s lower limit environmental licence conditions likely to be placed on blasting at the Martins Creek Quarry Extension Project based on a proven track record of achieving this.

5.0 AIR OVERPRESSURE ESTIMATION

Air overpressure levels generated from blasting have been commonly estimated using the following cube root scaling
formula:

$$P = K_a \left(\frac{R}{Q^{1/3}} \right)^a$$

where
P = pressure, in kilopascals

Q = explosives charge mass, in kilograms
R = distance from charge, in metres
Ka = site constant
a = site exponent

For confined blasthole charges as used at Martins Creek Quarry the site exponent (a) is normally -1.45 and the site constant (Ka) is commonly in the range 10 to 100.

This report uses a Ka of 10 and an (a) of -1.45 and for a charge of 20kg at 270m the estimated blast induced air overpressure is 115dBL which is equal to the 115dBL likely lower environmental Licence limit as recommended by ANZEEC 1990. This is simply an estimate and it is the actual blasting results that should be considered in preference to a theoretical calculation.

It should be noted that the likely limits for blast induced air overpressure according to ANZEEC 1990 are:

- 95% of the blasts must be less than or equal to 115dBL with
- 5% of the blasts allowed to be greater than 115dBL but less than or equal to 120dBL with
- no blasts to exceed 120dBL.

Air overpressure is measured in dBL units and the dBL scale is logarithmic to the base 2 so each 6dBL increase represents a doubling of pressure or a 6dBL reduction is a halving of pressure.

Appendix 1 contains the ground vibration and air overpressure results of all blasts fired between January 2013 and the middle of August 2015 at the Martins Creek Quarry. There has been 1 blast that exceeded the lower environmental licence limit at 330 Dungog road with a 116.8dBL due to wind concentration of the blast induced air overpressure down a natural gully. Remedial actions have been taken to minimise the potential for this to occur again even though it is 1 lower environmental exceedance in 68 blasts which represents a 1.47% exceedance rate and 5% are allowed between 115dBL and 120 dBL. The remedial actions are:

1. not firing if the wind is greater than 2m/s towards 330 Dungog road
2. use of 76mm diameter face holes
3. decking of these holes
4. the use of a 67ms delays in the control row with 42ms delays back through the shot to slow the shot down and separate the movements of the face
5. not firing a shot if the wind speed and/or direction are unfavourable and sleeping of the shots overnight with a guard and firing the next day when the wind direction and/or speed are more favourable.

In conclusion the blast induced air overpressure is likely to be harder to control than the blast induced ground vibration due to other factors like wind, wind direction etc beyond the quarries control but Martins Creek Quarry has the track record proving it can manage blast induced air overpressure to meet the current and likely future air overpressure environmental license limits.

6.0 MANAGING FLYROCK

Appendix 3 contains detailed methodologies in controlling flyrock.

The Martins Creek Quarry has implemented a number of mechanisms to reduce the risk of flyrock to ALARP that have achieved the required outcome as there has never been a flyrock incident at the Quarry and especially in the last 3 years that Dracon have owned and managed the quarry.

The basis of safety to reduce flyrock is based on:

1. use of 76mm diameter face holes with smaller charge weights so it is less likely to generate flyrock
2. use of decked charges to reduce powder factor and hence flyrock
3. use of 2.5m of minimum burden for the 76mm diameter holes which is 32.9 hole diameters and is generally used for medium soft rock and the quarry is composed of hard Latite rock. This large burden therefore mitigates risk of flyrock and air overpressure
4. use of long stem lengths of 2.5m or 32.8 hole diameters which is for very soft rock while the Latite rock type at the quarry is very hard so the design is very conservative and is aimed at reducing flyrock
5. use of a RMS accredited drill and blast provider that has considerable experience in close in construction blasting for roads so has the appropriate quality systems and procedures to correctly manage blasting risk like flyrock
6. downloading the sides and the backs of all blasts via the use of longer stem lengths of 3.2m to 3.4m to minimise side break and back break near the crest area which could create damaged rock increasing the potential for flyrock in the next blasts
7. use of slower delays across the front of the shot so there is less reinforcement from adjacent charges and hence reduced risk of flyrock
8. attention to detail in all aspects of the blast charging and firing process via control of the drill and blast process through measurement of charge weights, stem lengths, hole length and the use of KPI's to minimise variation hence reducing the potential for unwanted adverse blast outcomes like flyrock.

In conclusion the Martins Creek Quarry has a proven track record in managing flyrock and as long as it continues to utilise the relevant risk management strategies required to reduce flyrock to ALARP then it will continue to achieve this.

7.0 BLAST INDUCED GROUND VIBRATION AND AIR OVERPRESSURE LICENCE LIMITS

It is highly likely that the blast induced ground vibration and licence limits will conform to the ANZEEC Guideline 1990 which indicates the following:

1. For blast induced ground vibration that 95% of the blasts in a year will be less than or equal to 5mm/s PPV with 5% of the blasts having vibrations greater than 5mm/s PPV but less than or equal to 10mm/s with no blast induced vibration greater than 10mm/s and
2. For blast induced air overpressure that at least 95% of the blasts in a year will be less than or equal to 115dBL with a maximum of 5% of the blasts having air over pressures greater than 115dBL but less than or equal to 120dBL with no blast exceeding 120dBL

The Martins Creek Quarry Extension Project should have no concerns with being able to achieve these limits especially the ground vibration but the air overpressure is affected by wind and other atmospheric conditions such as inversions.

The distance that airblast travels from a blast site is highly dependent on the prevailing wind and atmospheric conditions. Blasts that are similar can produce significantly different airblast readings at a particular monitor dependent on these conditions.

Air overpressure can be focussed by one of two mechanisms:

1. Wind – the effect of wind is directional with the airblast being focussed down wind and being at reduced levels up wind. The effect is similar to that experienced by talking to a person down-wind versus up wind with the speaker having to increase volume to make themselves heard by the person up wind while a low volume can be used to communicate with the person down wind. Figure 1 clearly shows wind causing the pressure waves to bend up and curl over down-wind thereby enhancing the airblast in this direction and reducing it up wind. This phenomenon can be used to reduce the effects of airblast by only firing blasts when the wind is in a favourable direction. Quarries and certain mines in environmentally sensitive areas keep in close contact with the local weather bureau to make decisions on when to blast. Other mines will fire small calibrated unconfined test charges with the airblast being recorded at the sensitive area. Depending on the result the quarry will either blast or hold blasting to another time.

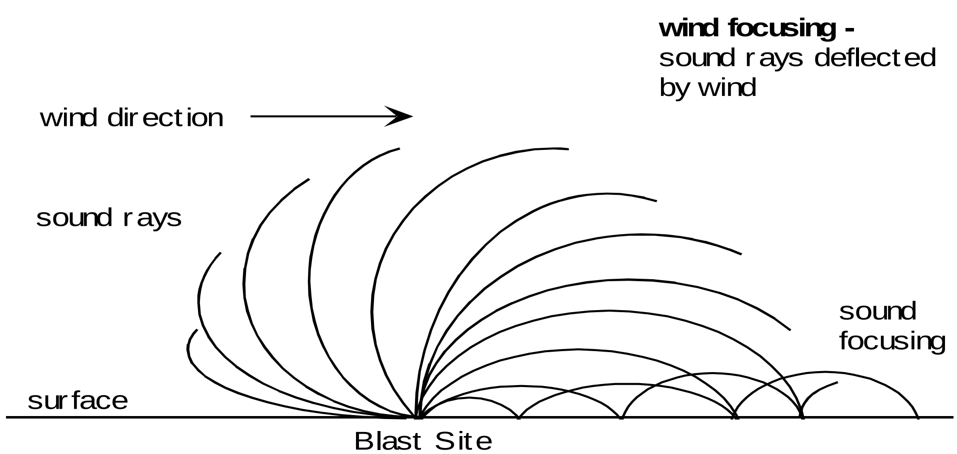
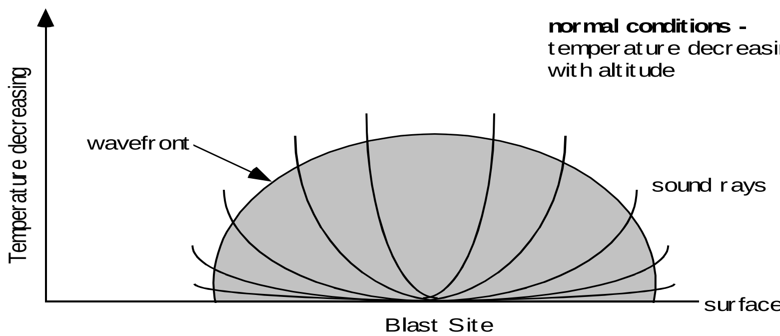


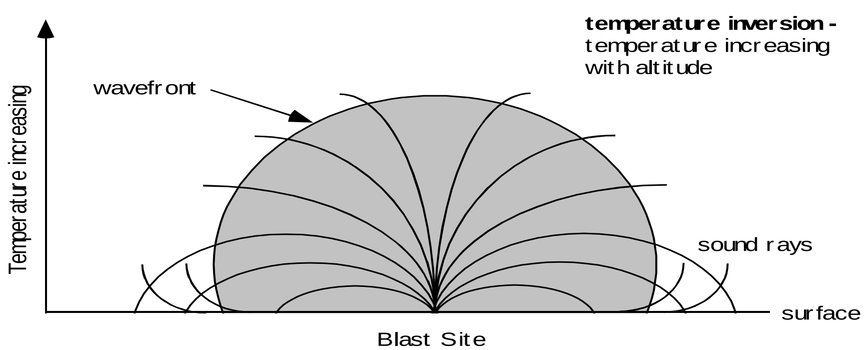
Figure 1 - Wind Direction Focussing Airblast Down Wind after J Floyd 2005

Wind can also create a turbulence layer when it passes over hills. This is a little understood mechanism but this turbulence layer has been known to create a layer above blast sites such that the airblast is reflected and causes disturbance at sensitive sites.

2. Temperature inversions – in normal atmospheric conditions the temperature of the air gradually reduces as altitude increases in the atmosphere. Under these conditions any sound produced on the ground generates a series of air overpressure waves that curve up into the atmosphere as per figure 2. This is favourable for blasting as any airblast produced will curve up harmlessly into the atmosphere. The airblast will only affect a limited area of extent and will not be focussed at any sensitive areas. However, when a temperature inversion occurs the normal temperature relationship with height in the atmosphere is

reversed and the temperature increases with altitude until a cut off layer is reached. With temperature inversion the airblast overpressure rays are bent back towards the earth (figure 3) and produce high airblast at the points where they impact on the earth's surface and if these happen to be a population centre then airblast complaints are likely to occur. Temperature inversions are generally indicated by fog, haze or smoke plumes that only travel a short distance vertically before they travel horizontally. For sensitive sites it is important to gather information from the local weather bureau to determine if an inversion is occurring and if it is at what time is it expected to disperse. Depending on the information provided blast loading may need to be cancelled if the inversion is unlikely to clear prior to blast time which would be unusual as inversions normally occur in the early mornings and have usually disappeared by midday. Continue to liaise with the weather bureau to confirm the dispersal of the inversion prior to blasting. Do not blast if an inversion is still present especially in environmentally sensitive sites.





Figures 2 and 3 Normal Conditions where Airblast dissipates into the atmosphere versus Inversion Conditions Causing Airblast Waves to Bend Down to Earth after J Floyd Efficient Blasting Techniques Course 2005

As previously discussed wind direction affects airblast but so does the actual wind speed. This becomes more apparent by viewing the data presented in table 1 below, where the dynamic wind gust pressures and velocity conversions for a range of wind gust speeds are listed.

Velocity			Pressure (Pa)	dBL Equivalent
m/s	km/hr	Knots		
0.5	1.8	1.0	0.125	75.9
1	3.6	1.9	0.5	87.9
2	7.2	3.9	2.0	100.0
3	10.8	5.8	4.5	107.0
4	14.4	7.8	8.0	112.0
5	**18.0**	**9.7**	**12.5**	**115.9**
6	21.6	11.7	18.0	119.0
7	**25.2**	**13.6**	**24.5**	**121.9**
8	28.8	15.6	32.0	124.0
9	32.4	17.5	40.5	126.1
10	36.0	19.4	50.0	127.9
11	39.6	21.4	60.5	129.8
12	43.3	23.3	72.0	131.1

Table 1 - Wind Gust Pressure/Velocity Conversions

The above data indicates that air overpressure measurements should be affected by the wind conditions if the velocity of the wind is greater than or equal to 5m/s as this can generate air overpressures of greater than or equal to 115.9dBL although the wind shields are supposed to reduce this wind affect. However, actual airblast records indicate that the wind shields are not very effective as the prevailing wind with velocities of greater than or equal to 5m/s at the time of a blast frequently show up in recorded wave traces when examined by experienced personnel. The effect of the wind can be difficult to distinguish from blast induced air vibration unless the interpreter has appropriate experience.

The data in table 1 also indicates that air pressure changes due to wind can exceed the typical environmental licence condition limits of not exceeding 115dBL (Lin Peak) for more than 5 percent of the blasts over a period of 12 months or not exceeding 120dB (Lin Peak) at any time. This is not well known and even less understood in the quarrying and mining industry although AS2659.1-1998 Guide to the Use of Sound Measuring Equipment Part 1 – Portable Sound Level Meters specifically addresses this issue in clause 3.3.3 part (a):

“Gusty wind conditions. In general, measurements are not practicable in wind speeds above 20km/hr (approximately 5 m/s)”

The key word in the above is ‘Practicable’ which The Macquarie Dictionary (published by Macquarie Library Pty Ltd 1981) defines as – 1. “capable of being put into practice, done, feasible or 2. capable of being used. So AS2659.1-1998 indicates that in wind speeds above 5m/s portable sound level metres, i.e. airblast monitors, are not capable of being used or put into practice i.e. the readings are suspect if not invalid.

It is therefore, prudent to establish a procedure for either measuring the wind speed or obtaining an estimate of the wind speed at the time of the blast. The measurement alternative involves purchasing a wind speed monitoring device while the second technique involves utilising the data generated by the local weather station. The direct measuring option is preferred as this monitors the actual wind speed at or near the time of the blast while the weather bureau data is usually 15-minute average readings taken at some distance from the blasting site, which may not accurately reflect the prevailing wind conditions at the time of the blast. AS2187.2 2006 indicates that weather conditions, especially wind speed and direction and cloud cover and other conditions such as rain should be noted on the blast records. The Martins Creek Quarry use the wind speed data from the Tocal weather station as well as a portable wind speed monitor but the actual wind speed and direction is affected by the topography surrounding and created by the actual quarry and has been found not to reflect the wind at the monitor sites so the preference is the use the data from the Tocal weather station.

With the measurement alternative a suitable wind meter must be purchased and a procedure incorporating an appropriate recording sheet such as that detailed below in figure 4 established. One suitable wind meter is a Kestral 1000 Pocket Wind Meter which has an accuracy of the greater of +/- 3% of the reading or +/- the least significant digit. The range of the device is 0.3m/s to 40m/s wind speeds and it operates in a temperature range of -15^{O}C to 60^{O}C. The accuracy, range and temperature tolerance, portability and ease of operation and low cost make the Kestral 1000 Pocket Wind Meter an ideal wind speed measuring device for blast monitoring situations.

Form No: Issue : 1 Date : May 2005	**Wind Velocity Record**	

Project: **Readymix Bohle**

Blast No: 300805 #0535
Date: 30/08/05
Time of Measurement: 11.30 am
Wind Velocity Measurement Procedure Slide cover off the Thermo-Anemometer. Press the **on** button. Select the operating mode by pressing and releasing the **mode** button until maximum 3 second gust appears on the monitor (**MAX**). Then select Measurement Scale by holding the **on** button down while pressing the **mode** button to select **M/S** on the screen. To take the measurements simply point the unit into the air flow you wish to measure for a minimum of 3 seconds but preferably for 10 to 12 seconds. Take 3 such measurements and record the highest reading in the space provided on this form. The wind direction should be estimated and the appropriate direction circled on this form. These measurements shall be taken a maximum of 5 minutes prior to the blast being fired. The form must be signed and dated by the responsible wind measurement personnel.

Maximum Wind Velocity (m/s): 4.8………………………

Wind Direction: N NE ENE SE S WSW W NW

Signed: *Graeme Cox*
Date: 30/08/05

Figure 4 Typical Wind Speed Recording Form

File the wind speed forms in the appropriate folder after transferring the velocity and direction data to the blast vibration and airblast data base.

When sending data to the relevant Regulatory Authority like the EPA always supply the wind speed data for all blasts with those blasts with wind speeds of greater than or equal to 5m/s highlighted with a note indicating that these are suspect/invalid readings as per AS2659.1-1998. Even though this standard is not current no other standard has replaced it and the science behind it has not changed so it is still valid and relevant government departments like the RMS still refer to this standard.

8.0 PROPOSED HOURS, FREQUENCY AND METHODS OF BLASTING

8.1 Proposed Hours

Blasting will be undertaken between the hours of 9am to 5pm Monday to Saturday with no blasting on Sundays or public holidays. Blasting will not take place more than once per day except if a misfire occurs as this is a safety issue which is likely to require a second blast to refire the misfired portion of the blast on the same day. It is therefore proposed that blasting generally be undertaken in the afternoon prior to 3pm to enable misfire remediation in the unlikely event of a misfire occurring.

8.2 Frequency of Blasting

It is expected that blasting will be undertaken about 50 times per annum as this will facilitate:

1. Less disruption to near neighbours although this is expected to be minor based on the estimated blast induced vibration and air overpressure and
2. Will enable cost effective blasting.

Obviously the rate of blasting will not increase to 50 per annum from the current 25 -30 per annum as soon as the extension is approved but will gradually increase over time as production ramps up.

8.3 Methods of Blasting

1. Free face blasting will be used where possible to reduce blast induced vibrations
2. Good quality stemming – a well sorted crushed angular aggregate of a nominal size of 9mm (7 to 11mm) will be used to minimise air overpressure
3. A good quality drill and blast provider with a proven track record in quarry and construction blasting should be used and an excellent shotfirer that has attention to detail – that is excellent QA/QC procedures and is used to close in blasting
4. Bulk emulsion/watergel will be used in both dry and wet ground to minimise fume generation
5. See sections 10 and 11 that specifically target blast induced vibration and air overpressure reduction and Appendices 2 and 3 for fume reduction and flyrock reduction respectively
6. The drill and blast contractor must have suitable systems and procedures that have been accepted by the RMS for close in blasting via working on RMS projects and have a current OHSE plan and relevant licences
7. All blasting will be undertaken to meet all legislative (NSW Explosives Act 2003), reg-

ulatory (NSW Explosives Regulation 2013), Australian Standards (specifically AS2187.2-2006) and Codes of Practice as a minimum including ANZEEC Guidelines 1990 and licence conditions with respect to blasting

8. There will be no need to store explosives or explosive precursors on site as they will be mobilised to each blast with any excess returned to the drill and blast contractors or the explosive supplier's premises.

9.0 INITIAL ASSESSMENT OF THE LIKELY IMPACTS OF BLASTING

The initial assessment of the likely impacts of blasting is:

1. People – minimal due to the low vibration and air overpressures to be generated and the lack of after blast NOx fumes
2. Animals – minimal as once they become used to blasting and determine it poses no threat then they are not affected by the proposed low levels of vibration or air overpressure. This is based on personal experience where two horses relatively close to blasting – they were inside the personnel clearance zone – were observed who had no experience with blasting. They heard the blast clearance sirens for the first blast and stopped, the ears went up and they turned towards the siren and when the blast went off they galloped away. The second blast the ears went up and the horses turned towards the sound and when the blast went off they propped but did not run away. The third blast the ears did not go up and when the blast went off they did not respond as they had experience which indicated that there was no danger from this new activity.
3. Buildings are extremely unlikely to be damaged as the environmental licence limits are for human quality of life and these are well below those that cause damage to buildings
4. Infrastructure is highly unlikely to be damaged for the reasons in point 3 above
5. Significant natural features are even less likely to be damaged for the reasons in point 3 and wind will generate far more air overpressure than blasting.

10.0 GROUND VIBRATION MITIGATION

There are a number of methods and techniques to reduce and control blast induced ground vibrations:

1. Develop accurate site laws (blast induced ground vibration equations) for each of the monitors at a site based on actual blasting results. These site laws will reflect the actual conditions at the site and provide a realistic blast design tool to determine the Maximum Instantaneous Charge (MIC) for each new blast based on its location i.e. distance from the monitors. It is important that separate site laws are developed for each monitor location as different locations can have significantly different blast induced vibration equations based on the effectiveness or lack of geological filtering or enhancement – this has been done at the Martins Creek Quarry.
2. Reduce charge weight per delay i.e. MIC to under that predicted by the site vibration laws. Determine the monitor location that has the greatest influence for a new blast and design the MIC for the blast based on this. There are a number of methods of reducing MIC such as:

    - Using smaller diameter blastholes – increases drill and blast unit costs

- Using reduced bench heights – significantly increases drill and blast and mining costs. It also reduces yield per blast, creates access problems and decreases mining efficiency via more equipment downtime in shifting to cope with double the number of benches and blasts. Overall fragmentation is also coarser due to two stemming zones with no explosive being loaded into these
- Use hole by hole firing – is relatively easy to implement and there is no increase in costs
- Use decking – this is where a blasthole has 2 or more explosive decks each separated by a minimum of a 1m length of crushed angular aggregate. These decks are fired top down to reduce confinement and are separated by 25ms timing between each deck. This is more expensive and time consuming to implement than point 3 but is less expensive than reduced bench heights and using smaller diameter blastholes. Hole by hole firing is generally the best technique but this is restricted to quite large MIC's. As blasting gets closer to housing etc blasthole MIC's are often too large and this is where decking or reduced bench height should be considered. It is in these situations where the requirement for accurate site laws is essential. Martins Creek Quarry do this

3. Reduce explosive confinement. There are a number of methods to achieve this:
    - Maximise the number of free faces but always have a minimum of 1 free face to promote forward movement – minimum at the quarry
    - Remove any loose buffer material from the face as this will reduce forward movement and increase vibration. Always clean up the face rill and minimise this – to be implemented at the quarry
    - Reduce burden and spacing as this will increase the energy level which aids in muckpile movement. The face burden must still be sufficient to minimise the potential for airblast and flyrock
    - Reduce stemming but the stem length must still be of sufficient length to minimise airblast
    - Reduce sub-drill to the minimum to prevent toe. The sub-drill region is where a significant amount of vibration is generated as this material has no effective free face and must wait for the material above it to move before a free face is generated and the fractured sub-drill rock is then free to move
    - Ensure that the length of the blast is at least 3 times the depth as this promotes movement and reduces vibration – to be implemented at the quarry
    - Minimise the depth of the shot to a maximum of 4 rows as this is where the blast commences to choke up as there is not sufficient time for the row of blastholes in front to detach and the next row to start moving into the gap created. The back row wants to move prior to the row in front detaching. If more rock is required, make the blasts wider rather than deeper – to be implemented at the quarry. Martins Creek Quarry undertake many of these initiatives
4. Use the correct delays to ensure hole by hole and deck by deck firing and hence minimise the MIC to that designed. If required change the delays being used or in vibration sensitive areas, consider using electronic initiation. Martins Creek Quarry use the correct delays to ensure hole by hole or deck by deck firing
5. Initiate the shot so that the initiation sequence progresses away from the area of concern as any reinforcement will then be away from the area of concern

6. Allow for variations in RL to ensure that all the holes are drilled to a common floor level which is the actual floor level. This minimises any excess sub-drill and hence blast induced ground vibration. Martins Creek Quarry implement this
7. Allow for toe in the design by checking the actual floor level and not assuming the level. If required clear any toe out prior to the main blast or time the toe holes to fire before any main blast holes detonating. This is to ensure that the main blast is free to move. Martins Creek Quarry implement this as standard practice
8. Use longer delays as this provides more time for the blasted material to detach and provide room for the next row of material to move into the space created. Beware that longer delays affect the blast induced vibration frequency forcing them to lower frequencies. Martins Creek Quarry implement this as standard practice
9. Minimise blasthole deviation as this affects effective burden and spacing. Larger than design burdens and spacing's increase vibration as the material in these areas does not heave effectively causing the material behind the affected holes to choke increasing ground vibrations – Martins Creek Quarry implement this as standard practice
10. Implement appropriate systems and procedures to minimise variation in the drill and blast process by improving quality control. Ensure accurate drilling – sub depth, hole angle, burden, spacing, hole diameter. Plumb holes for depth and backfill with crushed angular aggregate as required. Ensure the correct down the hole delays are used and the primer is pulled up into good explosive product. Ensure explosive is not contaminated by water or rock inclusions that could result in deflagration or misfires. Ensure that correct stem length and stemming type are used. Ensure correct surface timing design is used and no surface and downline connections are missed out and all are attached correctly to minimise misfires that generally increase blast induced vibrations. Ensure that the correct explosive density and column rise are obtained so that the MIC restrictions are adhered to – to be implemented at the quarry. Martins Creek Quarry implement tis as standard practice
11. Reduce the number of blasts by using larger shots as this will reduce the frequency of exposure to blasting. Generally, the greater the number of blasts the greater the number of complaints – implemented at the Martins Creek Quarry
12. Where possible keep the total time of the blast to under 1 second as blasts longer than this tend to result in proportionately more complaints
13. Time blast firing to coincide with highest background noise and vibration or when the near neighbours are out.

11.0 BLAST INDUCED AIR OVERPRESSURE RISK MITIGATION

There are a number of methods and techniques to reduce air overpressure in environmentally sensitive quarrying and construction locations:

1. Use correct stemming – angular aggregate of 10% of the blasthole diameter to minimise stemming ejection – implemented at Martins Creek Quarry
2. Use adequate stemming length – use minimum of 25 hole diameters to minimise cratering and stemming ejection – Martins Creek Quarry use greater than 30 hole diameters
3. Remove toe and buffer material from the face as this promotes forward movement and reduces the potential for face burst, cratering and stemming ejection – implemented at Martins Creek Quarry
4. Laser profile the face shape and locate face holes to minimise face burst

5. Boretrack at least the face holes and combine this with laser profiling to determine the loading of each front row hole
6. Deck through under burdened portions of face holes using crushed angular aggregate or use air decks or packaged explosives depending on the amount of face burden
7. Use sufficiently long inter-row delays so that the row in front is in the process of detaching when the row behind detonates. This promotes forward movement and minimises cratering and stemming ejection – implemented at Martins Creek Quarry
8. Blasts should preferably be at least 3 times wider than they are deep to promote forward movement and minimise cratering and stemming ejection – implemented at Martins Creek Quarry
9. Blasts should normally be a maximum of 4 to 5 rows deep to minimise choking and hence cratering and stemming ejection – implemented at Martins Creek Quarry
10. Deck through weak rock, fractured zones or weak seams to minimise face burst and explosive loss leading to reduced forward movement and hence cratering and stemming ejection
11. Routinely video all shots to determine if face burst, stemming ejection or cratering are occurring – implemented at Martins Creek Quarry
12. If cratering is occurring, then analyse the cause(s). Monitor stemming placement, stemming material and stem lengths and if this is correct then increase stem lengths or reduce burden or do both
13. If stemming ejection is occurring, then analyse the potential causes. Examine the stemming material, monitor stemming placement and stem lengths and if this is correct then consider longer stem lengths or reduce the burden and/or reduce the hole diameter
14. If face burst is occurring then analyse the cause and implement remedial action such as reducing the column height, stem through weak zones, profile and laser profile the face and front row holes and use the data to individually load each front row hole
15. Implement appropriate systems and procedures and audit these covering the entire drill and blast process on a regular basis designed to minimise variation but especially hole deviation – implemented at Martins Creek Quarry
16. Orient the faces so they do not directly face residences, buildings or structures of concern. This can often be difficult to implement – implemented where possible at Martins Creek Quarry
17. The timing design should be such that the blast fires away from the affected residence or structure – implemented where possible at Martins Creek Quarry
18. Fire blasts at the noisiest part of the day
19. Fire shots when the nearby residences are empty
20. Develop a good relationship with the near neighbours
21. Keep face heights to a practical minimum – implemented at Martins Creek Quarry
22. Eliminate secondary blasting by using rock breakers etc – implemented at Martins Creek Quarry
23. Consider delaying the shot if the weather conditions are unfavourable i.e. wind speed and direction, inversion present of interest – implemented at Martins Creek Quarry
24. Only initiate shots when the weather conditions are favourable eg the wind is blowing away from the residence or structure – implemented at Martins Creek Quarry
25. If face movement is generating the airblast consider slowing the timing of the face holes down and/or increase the face burden or do both – implemented at Martins Creek Quarry.

12.0 BLAST MONITORING

All blast monitoring will be undertaken as per the requirements in AS2187.2-2006 Appendix J which are more stringent than the ANZEEC Guideline 1990. Monitoring will be undertaken at the closest affected residences as per the likely requirement of the blasting licence conditions. Figure 5 provides a plan of the extent the Martins Creek Quarry Extension Project as well as the

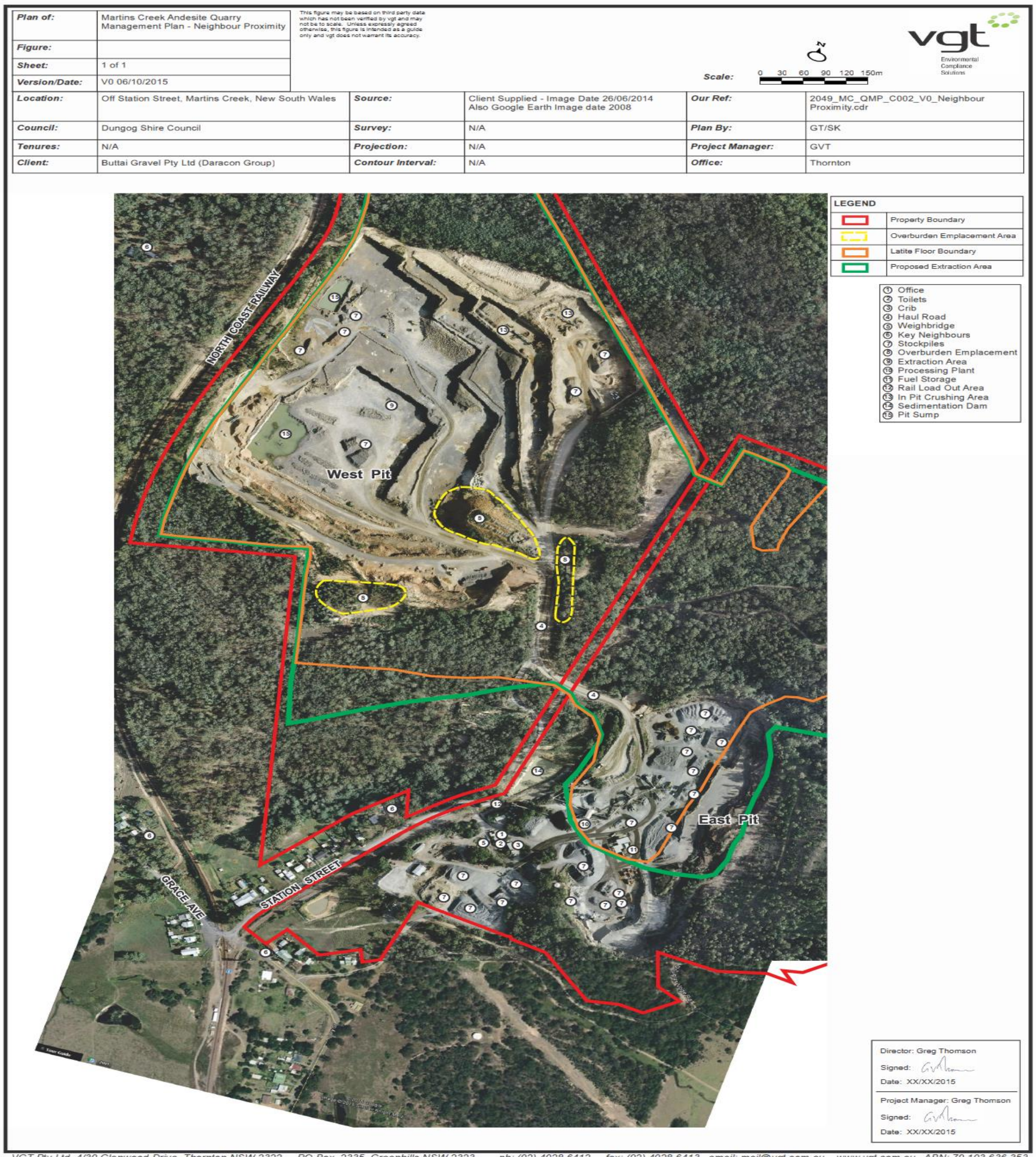


Figure 5 – Near Neighbours Plan Showing the Extent of the Extension Project

residences nearby which are limited to Station Street and Grace Avenue with the other being the Gully residence at 336 Dungog Road. Monitoring will still be undertaken at the Patterson View Estate and the front and back gates initially as at present but the back gate will be removed to Station Street and the front Gate to Grace Avenue as blasting moves within 600m of these residences.

12.1 Measurement of Blast Induced Ground Vibration

The equipment required to measure blast induced ground vibrations is specified in AS 2187.2-2006 - Appendix J. The measurement of blast induced ground vibrations is generally undertaken by instruments called seismographs that use geophones to measure particle velocity in millimetres per second (mm/s). The geophone is capable of measuring on three mutually perpendicular (at 90^{O}) axes with one being vertical and the other two being in the horizontal direction. The measurement equipment is capable of:

- Recording and playing back the blast vibrations generated by the full duration of the blast
- Indicating the maximum velocity for each direction as well as the Peak Particle Velocity which is the maximum vector value of the three directions
- Achieving instrument noise less than 10%of the PPV
- A frequency ranges of at least 2Hz to 250Hz with a tolerance of 10% over this range.
- The seismographs have a frequency response of 5Hz to 250Hz should be allowed in the vast majority of situations where this frequency range is adequate.

The purpose of blast induced ground vibration measurement is to accurately measure the magnitude of the ground vibration that is transmitted to the structure of interest at ground level. The measurement location should be sufficient distance from the structure so that any structural reflections do not cause spurious readings. Measurements can be taken on the foundation of the structure at ground level but under no circumstances should measurements be taken on the structure above ground level as they are affected by structural response.

The basis for coupling the geophone is to ensure that it faithfully records the actual motion of the ground and the preferred coupling method depends on the site conditions. Where there is a rigid surface i.e. concrete or rock suitable adhesive cement like plastibond can be used. Where the surface is soil vibrations can only be accurately measured using a buried mount such as a concrete block that is a 200mm concrete cube or a squat cylinder having a length equal to the diameter of 200mm. The concrete mount should be placed in a hole that is excavated about twice the size as the mount i.e. 400mm by 400mm by about 300mm deep so the top of the geophone when mounted on the block is just below ground surface. The block can have mounting bolts and a baseplate on its top to enable the geophone to be securely attached to the block if the measuring location is a permanent one. The block is placed in the excavated hole that has a level floor and some of the excavated soil placed around the side of the block about 40mm deep. This loose soil is then carefully compacted to ensure that the block is firmly in place with this process being repeated a number of times until the tamped soil is level with the top of the block thereby firmly anchoring the block to the remainder of the soil. The geophone must either be cemented or the mounting bolts and plate must be oriented so the geophone is facing towards the blast locations.

Blast vibration monitors are supplied with spikes that can be attached to the geophones and these used to spike the geophone in. Coupling with soil spikes in soft conditions can lead to erroneous blast vibration results that are 2 to 3 times larger than the actual vibration so the use of spikes is not recommended. Any mount in soil must be securely locked in place by the technique mentioned above.

12.2 Measurement of Blast Induced Air Overpressure
The measurement of air overpressure uses a microphone and the reading obtained is usually in dBL or decibels linear which does not modify the frequency content and is therefore able to be used for assessing air overpressure damage. The frequency range of the measuring equipment is at least 2 – 250Hertz with a tolerance of +/- 1dBL over this range and the monitoring equipment indicates the absolute maximum of the air overpressure (dBL). Where the air overpressure measurement is triggered by the ground vibration, the recording duration must be sufficient to account for the monitoring distance. Allow 3 seconds per kilometre of distance from the blast plus the duration of the blast.

The microphone is oriented towards the blast and has a suitable windshield. Many of the manufacturer’s windshields are quite ineffective and these may require replacing especially in windy areas. Consult the manufacturer for suitable alternative windshields. The microphone should be mounted on a suitable stand a minimum of 1 metre from the ground level unless specific site tests have been undertaken showing that accurate air overpressure measurements can be taken at a lower height. The microphone should be located away from structures at a minimum distance of 3.5m that may produce reflections and hence incorrect readings

13.0 CONCLUSION
There appear to be no significant issues to blasting being undertaken at the proposed Martins Creek Extension Project based on a proven track record in achieving Licence limits and meeting all regulatory requirements and the analysis and information presented in this report.

APPENDIX 1 BLAST MONITORING DATA FROM MARTINS CREEK FROM JANUARY 2013 TO AUGUST 2015

Blast Monitor Results									
				Site 1: 336 Dungog Rd		Site 2: Paterson Valley Estate		Site 3: Back Gate	
Date	Blast No	Bench	Blast Time	PPV (mm/s)	Overpressure (dBL)	PPV (mm/s)	Overpressure (dBL)	PPV (mm/s)	Overpressure (dBL)
1/03/2013	MC_P_008	3	13:49	1.89	109.3	3.72	106.4	No trigger	No trigger
20/03/2013	MC_P_009	2	12:19	1	103.4	0.76	98	No trigger	No trigger
28/03/2013	MC_P_010	4	10:39	3.15	108.5	0.88	99.3	0.62	109.7
11/04/2013	MC_P_011	4, N Face	12:15	2.06	114.6	1.34	89.4	0.58	109.2
26/04/2013	MC_P_012	2	12:18	0.79	110.7	0.53	83.4	No trigger	No trigger
17/06/2013	MC_P_013	4	14:00	1.99	83.4	0.7	101.7	0.58	113.6
24/06/2013	MC_P_014	3	12:33	3.11	87.8	1	86.9	No trigger	No trigger
4/07/2013	MC_P_015	4	12:47	1.58	106.5	1.68	92.9	1.11	106.5
2/08/2013	MC_P_016	4, S/W	11:10	1.97	112	1.12	97.3	0.34	112.4
9/08/2013	MC_P_017	Gully cnr	13:48	2.03	108.2	1.37	97.4	0.51	106.2
23/08/2013	MC_P_018	4, S/W cnr	12:19	2.36	112.6	0.89	96.6	No trigger	No trigger
17/09/2013	MC_P_019	3	12:16	1.88	111.5	0.88	91.4	No trigger	No trigger
26/09/2013	MC_P_020	4	13:54	No trigger	No trigger	1.23	114.8	No trigger	No trigger
24/10/2013	MC_P_021	4	13:12	No trigger	No trigger	0.25	113.1	No trigger	No trigger
1/11/2013	MC_P_022	3	13:04	1.23	93.8	0.85	86.9	No trigger	No trigger
7/11/2013	MC_P_023	4	11:57	2.3	112.9	1.33	101.5	No trigger	No trigger
22/11/2013	MC_P_024	3	13:28	No trigger	No trigger	1.28	104.9	No trigger	No trigger
3/12/2013	MC_P_025	2	13:09	1.16	111.6	1.67	87.7	No trigger	No trigger
23/01/2014	MC_P_026	3	12:49	1.82	110.3	1.11	87.7	No trigger	No trigger
31/01/2014	MC_P_027	0, Top Red	12:13	0.98	105.8	0.74	94.4	No trigger	No trigger
3/02/2014	MC_P_028	4	14:26	1.21	111.9	0.5	87.7	No trigger	No trigger
7/02/2014	MC_P_029	4	14:35	0.98	108.6	0.66	84.2	No trigger	No trigger
13/02/2014	MC_P_030	4	13:42	1.63	108.3	0.83	99.3	No trigger	No trigger
21/02/2014	MC_P_031	0, Mbl Crsh	14:26	0.81	107.9	No trigger	No trigger	0.47	96.6
25/02/2014	MC_P_032	3	13:56	1.2	109.3	0.89	102.4	No trigger	No trigger
27/02/2014	MC_P_033	3, 1 of 2	13:43	1.45	110.2	0.54	105.6	No trigger	No trigger
10/03/2014	MC_P_034	1	13.04	No trigger	No trigger	No trigger	No trigger	No trigger	No trigger
12/03/2014	MC_P_035	4	14:52	1.60, 1.20	106.5, 109.8	0.88	97.9	No trigger	No trigger
14/03/2014	MC_P_036	0	13:44	1.34	109	0.85	87.7	No trigger	No trigger
19/03/2014	MC_P_037	4	14:41	0.71	109.1	0.64	99.9	No trigger	No trigger

1/04/2014	MC _P_038	3	13:52	1.27	111.3	1.44	90.2	No trigger	No trigger
4/04/2014	MC _P_039	3	14:03	2.79	112.7	1.2	87.7	No trigger	No trigger
09/1/04/14	MC _P_040	1, New Orien	14:59	1.15	106.5	No trigger	No trigger	No trigger	No trigger
15/04/2014	MC _P_041	4, W Face	13:21	1.59	107.5	0.74	84.2	0.86	106.4
23/04/2014	MC _P_042	3, S/W Red Cnr	13.17	1.13	111.3	No trigger	No trigger	No trigger	No trigger
6/05/2014	MC _P_043	1	13:04	0.64	112.1	No trigger	No trigger	No trigger	No trigger
23/05/2014	MC _P_044	4	13:46	2.57	115	1.42	97.9	No trigger	No trigger
2/06/2014	MC _P_045	3	13:42	2.4	113.5	1.09	90.2	No trigger	No trigger
10/06/2014	MC _P_046	3	13:31	No trigger	No trigger	1.73	105	1.33	107
27/06/2014	MC _P_047	4	13:38	1.36	105.6	No trigger	No trigger	0.56	100.9
3/07/2014	MC _P_048	4	13:20	2.03	105	1.62	93.7	0.32	84.2
14/07/2014	MC _P_049	4	11:17	1.92	111	1.77	97.9	No trigger	No trigger
18/07/2014	MC _P_050	0	11:51	1.17	85	1.05	92.1	No trigger	No trigger
25/07/2014	MC _P_051	1	12:29	No trigger	No trigger	0.96	84.2	No trigger	No trigger
6/08/2014	MC _P_052	4	13:49	1.99	112.2	1.35	95.1	0.3	101.9
18/08/2014	MC _P_053	0	12:21	0.44	107.5	0.74	97.2	No trigger	No trigger
22/08/2014	MC _P_054	1	14:53	0.57	113.6	0.95	82.4	0.98	99.7
5/09/2014	MC _P_055	4	13:23	2.05	111.3	2.19	85.9	1.01	84.2
15/09/2014	MC _P_056	1	12:49	0.75	109.8	0.53	91.9	0.38	90.2
8/10/2014	MC _P_057	2	13:46	1.17	116.3	1.16	82.4	0.56	103.5
31/10/2014	MC _P_058	4	13:46	2.2	105.8	1.72	90.3	0.53	104.1
25/11/2014	MC _P_059	1	13:06	0.73	98.8	0.43	99.8	0.38	103
16/12/2014	MC _P_060	4	11:07	1.4	109	1.16	96	0.57	102.9
20/01/2015	MC_P_061	1	9:35	0.61	109.4	0.60	98.5	0.33	104.2
3/02/2015	MC_P_062	1	14:12	0.70	111.2	0.85	103.3	0.59	102.6
13/02/2015	MC_P_063	4	11:41	3.63	114.3	0.89	103.3	0.59	109
3/03/2015	MC_P_064	4	13:11	1.44	113.9	1.21	98.2	0.82	103.8
19/03/2015	MC_P_065	4	13:08	3.23	107.4	1.16	96.6	No trigger	No trigger
24/04/2015	MC_P_066	2	15:00	0.66	113.7	No trigger	No trigger	0.83	82.9
29/04/2015	MC_P_067	2	13:32	1.06	103	1.94	102.9	1.78	103.4
4/05/2015	MC_P_068	1	13:04	0.89	102.1	0.89	90.3	0.89	103.3
14/05/2015	MC_P_069	3	13:39	0.78	109.6	0.92	95	0.35	109.6
4/06/2015	MC_P_070	2	12:09	0.56	109.8	0.70	97.7	0.75	97.1
26/06/2015	MC_P_071	4	13:41	2.41	108.2	1.55	99.1	1.02	106.1
28/07/2015	MC_P_072	2	13:19	1.17	113	0.57	106.9	0.72	97.7
4/08/2015	MC_P_073	4	12:34	1.58	105.9	0.70	93.4	0.21	109.8
11/08/2015	MC_P_074	4	12:34	0.77	109.4	0.55	100.6	0.52	98.5
17/08/2015	MC_P_075	4	12:34	1.58	109.8	0.88	98.2	0.53	98.5

APPENDIX 2 REDUCTION IN BLAST FUMES

Fume occurs when post detonation yellow to red-brown to purple coloured NOx gases are produced. These are observable where as those blasts producing colour less Carbon Monoxide are not normally considered to fume as nothing can be seen. A number of factors can potentially cause fume:

1. Improper priming
2. Lack of confinement
3. Insufficient water resistance
4. Excessive sleep time
5. Incomplete product reaction
6. Reaction of explosive with rock
7. Incorrect hose handling
8. Incorrect product formulation

So when a fume problem occurs it is often fairly difficult to determine an exact cause or causes due to the significant number of factors that could be causing the fume either singularly or in combination.

1. Improper Priming

The causes of improper priming are:

- Primer located in contaminated bulk explosive near the base of the hole. As pumped bulk explosive is loaded it tends to churn up the material in the base of the hole contaminating a portion of the product. Explosive companies put dispersers on the bottom of the hose to reduce the force of the bulk product being loaded but some contamination still occurs
- Primer is located in water damaged or water included product
- Primer is too small
- Primer is the incorrect type

The techniques to minimise fume from improper priming are:

- Pull the primer up into good bulk product lifting it off the base of the hole. The primer should be pulled up at least 30cm
- Consider double priming in wet holes even when the blasthole depth is less than 10m. Always multiple prime in wet holes above 14m and use the rule of thumb for an additional primer for every 15m of blasthole in wet conditions as multiple priming minimises the risk of both fume and misfires
- Audit hose handling practices and ensure hose handlers have undergone competency based training by requesting copies of training records to minimise risk of included water
- Follow explosive supplier recommendations for the size and type of primer
- Consult with explosive supplier personnel or other technical personnel on the number and type of primers to use.

2 Lack of Confinement

The causes of lack of confinement are:

- Geology - highly fractured rock types with open joints tend to fume more than other rock types. When an explosive detonates a number of different explosive gases are initially produced including steam, nitrogen gas, carbon dioxide, carbon monoxide and NOx as the detonation occurs very rapidly. If these gases are confined the intense heat means that the carbon monoxide can take the oxygen from the NOx to produce carbon dioxide leaving nitrogen gas and no NOx results. If, however, the initial gases are free to flow into open cracks they rapidly chill as they flow away from the blast hole and some of the initially generated NOx gas survives. This is even more pronounced if the surrounding rock is damp and fume occurs
- Geology – soft or plastic rock types for similar reasons for highly fractured rock types
- Incorrect stemming that results in rifling when the stemming ejection is pushed out violently and quickly resulting in a loss of temperature and detonation gases.

The techniques to minimise fume from lack of confinement are:

- Use an explosive that is either ANFO or pure watergel eg RIOFLEX or emulsion as ANFO/RIOFLEX or ANFO/emulsion mixtures fume more
- Use an over fuelled product as this has an excess of carbon which is more reactive than nitrogen so it will scavenge any oxygen from the nitrogen
- Explosive formulation – specifically formulated explosives are available to reduce fume

None of these techniques will totally solve the fume problem in the highly fractured or soft plastic geologies due to the mechanism of fume formation but they will significantly reduce the amount of fume. Shane – removed this last sentence from being a bullet point.

3 Explosive has Insufficient Water Resistance

If the explosive of choice has insufficient water resistance the water attacks the explosive robbing it of both fuel and oxidizer which produces fuel rich and fuel poor areas. It is the fuel poor areas that produce the NOx gases. A similar mechanism is responsible for the NOx gases produced when a bulk explosive has been slept too long in a water affected area. In any application where the rock contains water it is always recommended that minimal sleep times be used and preferably load and shoot in the shortest time frame be incorporated into the firing plans to minimize fume and explosive deterioration leading to poorer than expected blasting outcomes.

The causes of insufficient water resistance are:

- Water resistance of the explosive being used
- Type of water in the ground - static versus dynamic. Static water does not move in the ground so has less effect on bulk explosives than dynamic water that is moving in the ground.

The techniques to minimise fume from lack of confinement are:

- Match the water resistance of the bulk explosive to the sites water conditions
- Minimise sleep time by going to load and shoot
- Ask explosive supplier for recommendations.

4 Incomplete Explosive Product Reactions

The causes of incomplete explosive product reaction are:

- Improper priming
- Minimum diameter issues caused by explosive product being damaged by water
- Incorrect hose handling where bulk explosive has water inclusions
- Bulk explosive product contamination – stemming bulk explosive product interface and bottom of hole

The techniques to minimise fume from incomplete explosive product reaction are:

- Implement correct priming
- Select bulk explosive or packaged explosive suitable to the water conditions
- Ensure correct hose handling by obtaining training records and auditing
- Insist on hose flow dispersers and ensure that the stemming process commences with a small amount of drill chips/dust being placed down the hole to stop stemming spearing into the explosive product.

5 Reaction of Bulk Product with Rock

The causes of reaction of bulk explosive with the rock are:

- If the rock is excessively basic or acidic in nature this can interfere with the gassing chemicals in the emulsion based explosive leading to a reduction in the amount of nitrogen gas produced that sensitizes the product. This increases the density making the explosive less sensitive and in the worst cases can result in a density close to the critical density in the skin area of the explosive column i.e. that portion closest to the rock. The outside skin of the bulk explosive may not properly detonate leading to NOx gas generation.

The techniques to minimise fume from reaction of the bulk product with rock are:

- Adjust gassing chemicals to ensure that the effect of the rock on the bulk emulsion is reduced. This is difficult to achieve as the vast majority of the bulk emulsion is not affected and any adjustment to the gassing chemicals affects all the bulk emulsion.
- Change to a bulk product to one that is less sensitive to the rock conditions

- Line the hole to provide a barrier between the bulk explosive and the rock
- Change the sensitisation technology from gassing to using glass micro-balloons but this is more expensive
- Change product to one that uses mechanical gassing.

6 Incorrect Hose Handling

Incorrect hose handling can lead to water being entrapped in the bulk explosive column. The hose must be placed at the bottom of the hole over the top of the primer. It must then be withdrawn at a steady rate always ensuring that the hose is at least a third to a half metre inside the top of the explosive column. This ensures that no water is trapped inside the explosive column that will lead to the formation of fume, poor blasting outcomes and in the worst case misfires

The causes of incorrect hose handling are:

- Lack of training
- Lack of attention to detail.

The techniques to minimise fume from incorrect hose handling are:

- Insist that only competency based trained hose persons operate on site by obtaining training records
- Audit hose handling.

7 Incorrect Explosive Product Formulations

The causes of incorrect explosive product formulation are:

- Product has less fuel than required by the formulation so there is an excess of oxygen available to produce NOx
- Product is beyond its shelf life due to contaminants introduced during manufacture, transport or storage
- Trace chemicals for gassing left out at manufacturing stage
- Incorrect trace chemicals or incorrect concentration of trace gassing chemicals put into the explosive delivery vehicle
- Trouble with mechanical aeration devices.

The techniques to minimise fume from incorrect product formulation are:

- Obtain report on quality control from manufacture through to explosive delivery vehicle operations
- Ask explosive supplier if other customers are experiencing similar problems
- Ring around other customers
- Review density cup measurements for the affected blast(s).

Incorrect formulation can also affect the water resistance/sleep time of the explosive leading to fume production.

In conclusion, fume problems can be difficult to resolve due to its multi causal nature. Most sites do not have the resources or the technical expertise to resolve fume problems in a timely manner. The most efficient way of resolving a fume problem is often to involve the explosive supplier and/or an appropriate consultant to resolve the issue.

APPENDIX 3 FLYROCK REDUCTION

Flyrock occurs when the high pressure explosive gases are vented violently from a blast propelling rock outwards from the blast area. Flyrock only becomes a concern if it impacts on machinery left in the blast exclusion zone or it reaches the boundary or flies beyond the boundary of the blast exclusion zone. Flyrock is the leading cause of blasting related fatalities and equipment damage in the US and is a major concern of most blasting operations in Australia. This is especially so in the construction industry where blasting can take place at close proximity to housing, businesses, hospitals etc or next to roads. In these locations flyrock must be limited to a very small distance from the blast measured in terms of metres or tens of metres. The quarry industry normally has larger zones free of equipment, infrastructure and houses but in some cases this is limited to less than 100m. It is therefore important to understand the causes of flyrock so that appropriate strategies and techniques can be put in place to minimise the risk of this adverse blasting outcome occurring.

The main causes of flyrock and the remedial actions to minimise the risk posed by each of these are:

1. Blasthole overloading – results in cratering as the path of least resistance is via the top of the shot, figure 1. It occurs via a number of factors such as:
   - short holes
   - hole smaller diameter than design
   - hole blockages
   - failure to monitor column rise

   The remedial actions required to minimise the risk of blasthole overloading are:
   - plumb all holes and either redrill short ones or adjust the designed hole charge
   - develop hole diameter/drill bit diameter standard and audit this
   - monitor column rise as part of standard practices and develop a column rise standard. Audit this.

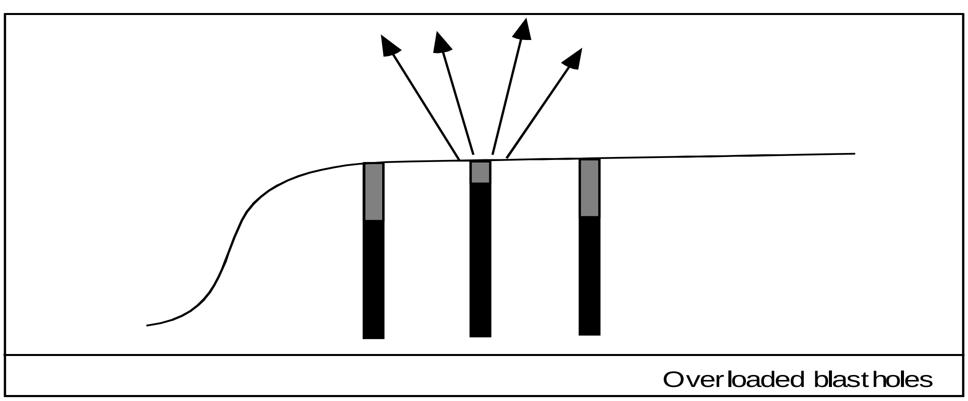


Figure 1 Overloaded Blastholes after J Floyd Blast Dynamics Efficient Blasting Techniques Course 2005

2. Less than design face burden – results in the path of least resistance being the face and the lack of confinement means that the high pressure explosive gases violently vent through the face ejecting rock fragments with the distance being a function of confinement, figure 2. Smaller than design face burden results from the following causes:
   - hole deviation
   - face shape – embayment
   - improper blast design
   - failure to calculate burden at the top of the explosive column
   - incorrect pattern layout
   - collar position accuracy during drilling
   - over digging

   The remedial actions required to minimise the risk of smaller than design face burdens are:
   - develop hole deviation standard and bore track front row holes
   - laser profile the face and combined with face row bore tracking provides vital information with which to design the charging configuration of each front row hole using the face hole loading standard based on burden ranges and style of loading i.e. crushed angular aggregate deck, air deck, packaged explosive etc
   - use laser profiling to design front row hole positions
   - develop a face burden design standard that has minimum and maximum face burdens
   - check blast design and pattern layout in the field and adjust front row hole collar burdens so they match the field conditions and laser profiling results
   - develop a collar position accuracy standard and audit to this
   - utilise digging tapes to ensure that face is not over dug

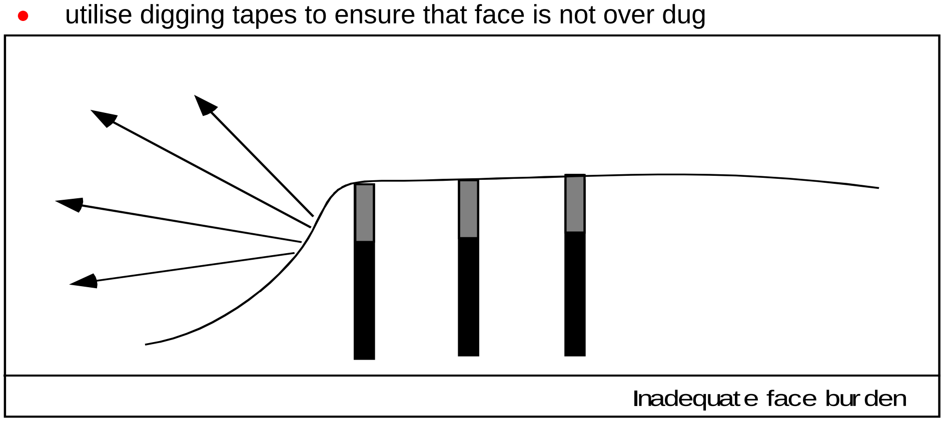


Figure 2 Inadequate Face Burden after J Floyd Efficient Blasting Techniques 2005

3. Excessive face burden – causes the remainder of the shot to choke up behind the face as it fails to move placing pressure on the stemming region and potentially resulting in cratering, figure 3. The causes of excessive face burden are:
   - incorrect design
   - incorrect pattern layout
   - buffer of broken material ie blast not free faced
   - incorrect face hole collar positions

The remedial actions to overcome excessive face burden are:

- insist on free face prior to drilling front row
- utilise laser profiling data to design the collar positions
- develop a face burden design standard that has minimum and maximum face burdens
- check pattern layout in the field and adjust as required
- develop collar position accuracy standard and audit this in the field.

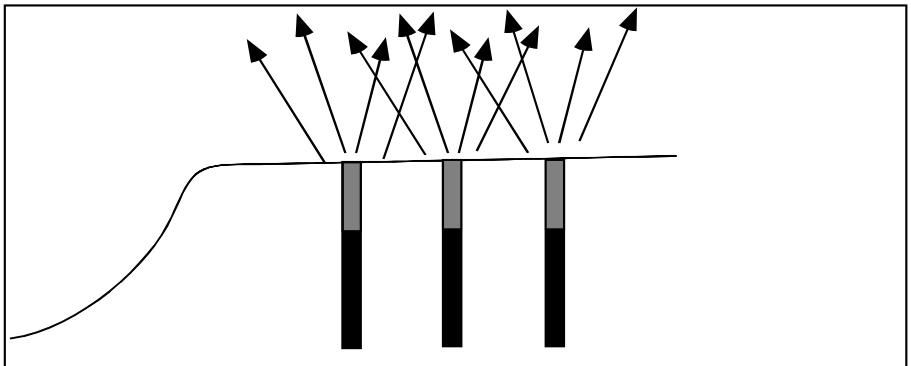


Figure 3 Excessive Face Burden after J Floyd Efficient Blasting Techniques 2005

4. Incorrect stem material – results in the stemming being ejected from the high pressure high velocity post detonation explosive gases that can rip loose rock fragments from around the blasthole collar region creating flyrock. The main causes of incorrect stem material are:
   - incorrect size
   - incorrect type – drill chips versus crushed angular aggregate
   - lack of understanding of the importance of using appropriate stemming material
   - water logged ground providing lubrication for the stemming to be ejected

The remedial actions for incorrect stemming are:

- use the correct size at 10% of the blasthole diameter
- use crushed angular aggregate.
- attending suitable training courses
- ensure a free face for water logged ground. Increase the stem length to a minimum of 30 blasthole diameters and increase the stemming size to a minimum of 14mm and a maximum of 20mm for 76 to 102mm diameter holes as this allows sufficient space for water to be pushed through the stemming and not to fluidise it which is the case for smaller 7-10mm diameter material.

5. Weak seams – results in the high pressure explosive gases preferentially venting into the soft weak seam as this is the path of least resistance, figure 4. The weak material can be explosively vented from the face and any newly created faces as the blast progresses creating unwanted flyrock. The main causes of flyrock from weak seams are:

- no drill logs
- no face inspection
- no discussion with driller
- not loading explosive to geological conditions

The remedial actions for minimising the potential for flyrock from weak seams are:

- instigate drill logs that the driller can allocate a hardness too on a metre by metre basis (colour coded)
- undertake face inspections
- discuss the pattern with the driller when undertaking audits and/or conduct a post drilling debrief
- utilise the results of the drill logs to plan the hole loading to deck through weak zones using crushed angular aggregate
- instruct the shot firer to load to the hole loading plan emphasizing the importance to minimising the risk associated with flyrock
- audit hole loading

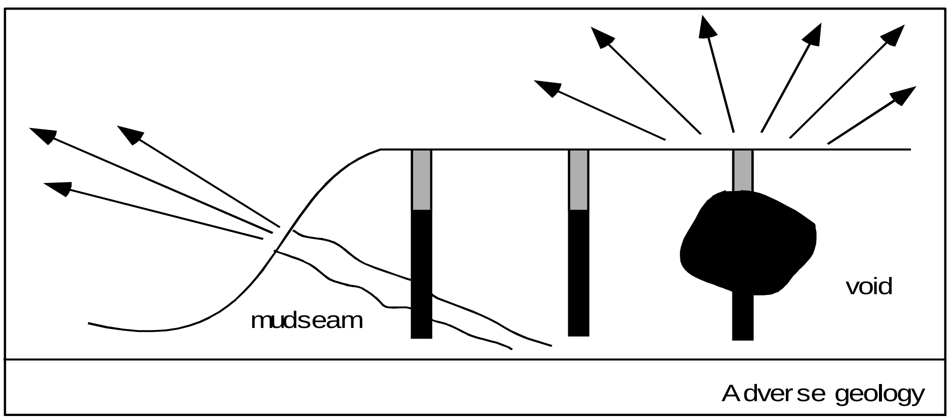


Figure 4 Role of Weak Seams and Voids in Creating Flyrock after J Floyd Efficient Blasting Techniques Course 2005

6. Water logged conditions - results in the high pressure explosive gases preferentially venting by ejecting the stemming creating flyrock by plucking rock fragments from the collar region. The causes of flyrock generation from water logged ground are:
   - not paying attention to details
   - use of incorrect stemming instead of crushed angular aggregate
   - failure to use a free face

The remedial actions for minimising the potential for flyrock from water logged ground are:

- conducting a pre blast design visit to the area of the blast as part of the design risk assessment. The water logged nature of the area should be apparent
- utilising the knowledge of the water conditions to ensure that at least one free face is available but preferably two.
- order larger diameter stemming in the range of 14 to 20mm
- brief the shot firer re using the larger diameter stemming

- use longer stem lengths i.e. a minimum of 30 hole diameters

7. Burden too large - results in the path of least resistance for the high pressure explosive gases being the top of the shot via cratering, figure 4. The causes of burden being too large are:
    - incorrect design
    - incorrect pattern layout
    - hole collar positioning inaccuracy
    - hole deviation
    - face holes too close to face and each hole only removing a narrow V of rock instead of extending halfway to the hole so face rock is left in place and the second row holes see a large burden

The remedial actions for minimising the potential for flyrock from the burden being too large are:

- check design against previous designs in the area
- check design against rules of thumb
- check pattern layout against the design
- develop hole collar positioning standard and audit against this
- develop hole deviation standard and conduct bore tracking on both face and a selection of main body holes to audit against the standard
- use laser profiling results to position face holes and check these at pattern layout to ensure correct face burdens as per the standard

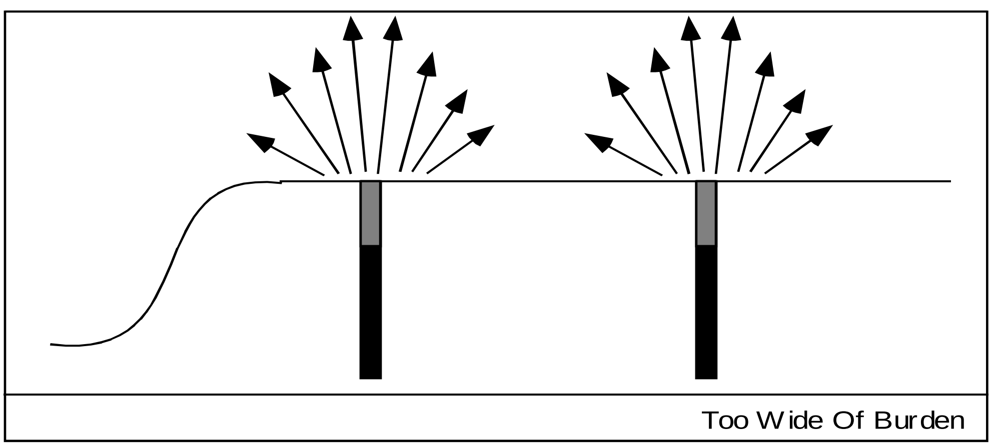


Figure 4 Burden Too Large Resulting in Cratering after J Floyd Efficient Blasting Techniques 2005

8. Excessive powder factor - results in the path of least resistance for the high pressure explosive gases either being the top of the shot via cratering and/or via face burst, figure 5.

The causes of excessive powder factor are:

- incorrect design

- incorrect pattern layout
- incorrect hole charge
- incorrect explosive loaded
- hole collar position accuracy
- hole deviation

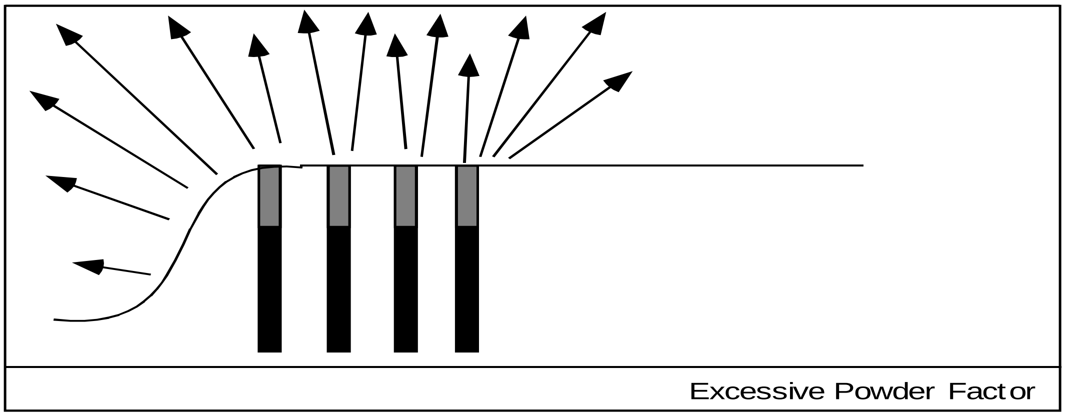


Figure 8.15 Flyrock Generated by Excessive Powder Factor after J Floyd Efficient Blasting Techniques 2005

The remedial actions for minimising the potential for flyrock from excessive powder factor are:

- verify design against previous designs in the area
- verify design against rules of thumb
- conduct a pre blast design inspection of the area noting any changes in rock type etc that may affect the design
- verify pattern layout against design
- ensure field controls are in place for column rise measurement, explosive density measurement and that the correct explosive type is being loaded and audit these
- conduct a pre blast loading briefing session with the shot firer covering the design, explosive loading and the field controls required
- audit the hole collar position standard
- conduct bore tracking of all face and a selection of main body holes and compare results against the standard

9. Inadequate timing between rows - results in the path of least resistance for the high pressure explosive gases being the top of the shot via cratering as the shot chokes up and does not move forward as designed, figure 6. The causes of inadequate timing are:
   - incorrect timing design or no formal timing design
   - actual timing inconsistent with the design
   - inaccurate delays

The remedial actions for minimising the potential for flyrock from inadequate timing between rows are:

- generate a formal colour coded timing design that meets the sites critical factors
- audit surface timing in the field to ensure compliance with the design

- request timing accuracy information from the explosive supplier to ensure that no overlaps can occur

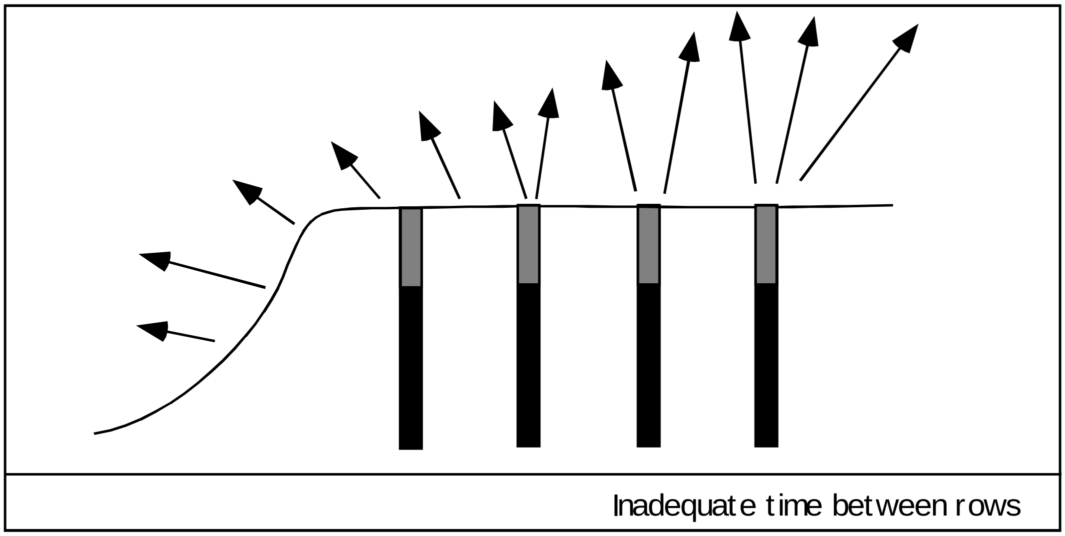


Figure 6 Flyrock Generated from Inadequate Timing Between Rows after J Floyd Efficient Blasting Techniques 2005

10. Pattern too deep - results in the path of least resistance for the high pressure explosive gases being the top of the shot via cratering as the shot chokes up towards the back of the shot, figure 7. The causes of the pattern being too deep are:
    - improper blast design
    - production requirements over rule good drill and blast design
    - poor planning
    - greater than 5 rows blast will choke leading to over confinement at the back of the blast

The remedial actions for minimising the potential for flyrock from the pattern being too deep are:

- only planning blasts a maximum of 5 rows deep or if the shots need to be deeper increasing the timing between the rows
- make pattern wider if more stone is required
- education of site personnel on the need to keep the blasts at a maximum of 5 or so rows
- always ensuring that the blasts are fully excavated to provide new free faces for blasts as this provides additional areas to drill.
- never let production dictate pattern design as deep patterns will lead to a loss in digging efficiency as well as the potential for flyrock

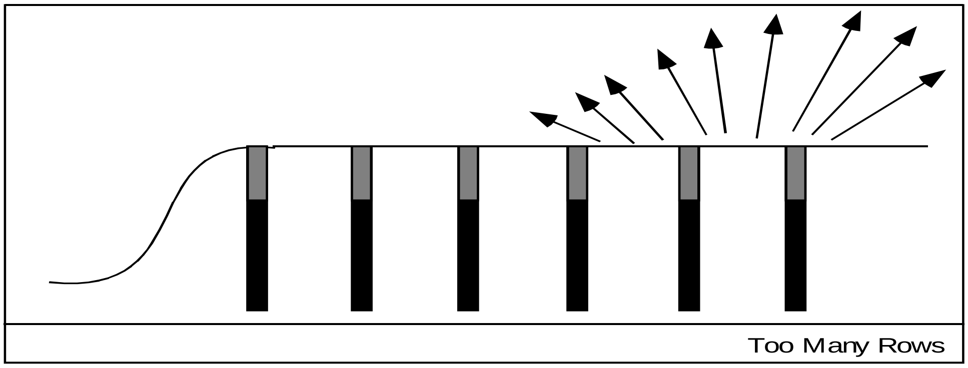


Figure 7 Flyrock at Back of Deep Shot after J Floyd 2005

11. Buffer of material on the face - results in the path of least resistance for the high pressure explosive gases being the top of the shot via cratering causing the face not to move, figure 8. The causes of a buffer of material being left on the face and the shot being fired are:
    - poor face clean up
    - production constraints
    - poor planning
    - inability to convince key personnel to provide a clean free face

The remedial actions for minimising the potential for flyrock from a buffer of broken material being left on the face are:

- insist on the blast having a free face prior to drilling commencing
- insist on the key personnel signing off on the risk assessment indicating that firing with a buffer is acceptable even though the risk of flyrock is not being mitigated
- increase stem lengths in shots by a minimum of 3 hole diameters for dry shots and 6 hole diameters in wet shots with a buffer clearly indicating that this will lead to a reduction in fragmentation at the crest of the shot

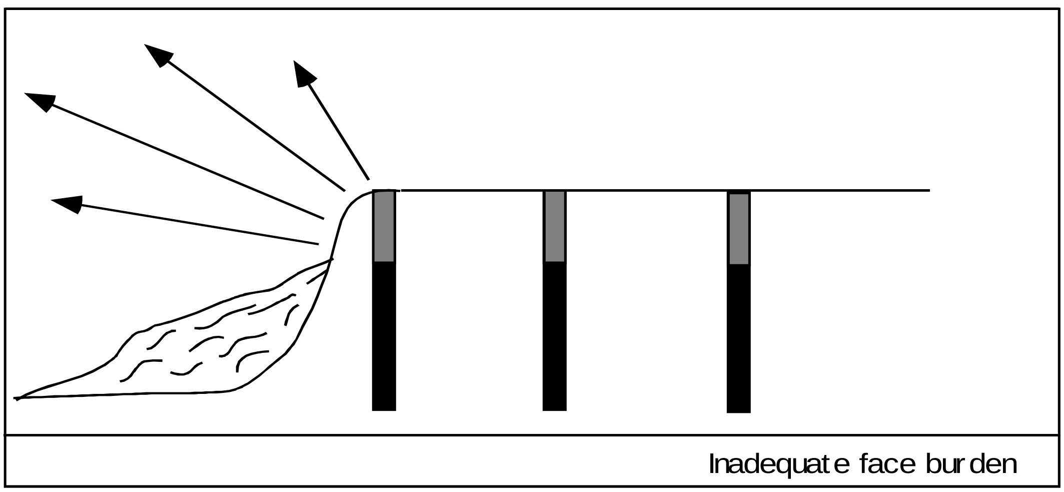


Figure 8 Flyrock Potentially Generated from Buffer of Material Left on the Face after J Floyd Efficient Blasting Techniques Course 2005

12. Misfires – result in a lack of burden as the misfire has been detected during digging so most if not all the burden has been removed. In addition, the blast surrounding the misfire has cracked the ground providing multiple paths of least resistance for the high pressure explosive gases to escape and create flyrock, figure 8.19. The causes of flyrock from misfires are:
    - the misfire
    - the lack of burden
    - cracked ground from blast induced fracturing

The techniques to minimise risk of flyrock from misfires are:

- reduce potential for misfires by using appropriately trained and qualified personnel who have high levels of attention to detail as they understand the ramification of misfires
- fully investigate each misfire and instigate remedial actions to minimise the potential for that type of misfire to occur again
- don't refire misfires but wash out if ANFO or use detergent loaded water to wash out Heavy ANFO or pumped product
- if refiring option selected then cover area with sand plus use blast mats and instigate a bigger than normal blast clearance zone

13. Redrilled holes - result in an ideal path of least resistance for high pressure explosive gases to vent plucking rock fragments from the collar region creating flyrock. The causes of redrilled holes are:
    - collapsed or partially collapsed holes
    - holes outside the standard for collar position accuracy or hole deviation
    - short holes

The techniques to minimise risk of flyrock from redrilled holes are:

- to minimise the number of redrilled holes by insisting the appropriate standards are adhered to with failure to conform being redrills at own expense
- filling all redrills and collapsed holes with correct sized crushed angular aggregate

14. Voids - result in localised explosive overloading and an explosive energy hot spot in the blast resulting in flyrock, figure 8.13. The causes of explosive overloading from voids are:
    - lack of drill logs
    - lack of discussion with driller
    - no monitoring of column rise
    - not following loading design

The techniques to minimise risk of flyrock from voids are:

- instigate drill logs and use these in designing the hole by hole explosive loads
- discuss how the ground is drilling when auditing drilling standards
- always have an after drilling the blast debrief

- deck through the void or fill the void with crushed angular aggregate
- insist on column rise being monitored
- insist on either the MIC or the designed load being used to load each hole

15. Blasthole deviation - results in under burdened face holes and potentially main body holes plus localised explosive overloading and explosive energy hot spots in the blast resulting in flyrock. The causes of hole deviation from are:
    - geology
    - hole angle
    - guide rods not used
    - excessive penetration rate
    - no hole deviation standard
    - no auditing of hole deviation
    - failure to load deviated holes accordingly – ie use of decking, packaged explosive, air decking etc

The techniques to minimise risk of flyrock from blasthole deviation are:

- develop a suitable hole deviation standard that is agreed to by the driller
- audit this by bore tracking all front row holes and a selection of main body holes
- redrill holes that do not meet the standard
- use laser face profiling to design front row holes with view of minimising angle if possible
- encourage the use of guide rods although hole deviation standard does this coupled with auditing and redrilling
- use laser profiling and borehole tracking to design hole by hole explosive loads to the site face burden standard
- pre loading briefing with shot firer where hole deviation results are discussed plus the effect on hole charging
- audit hole charging

16. Secondary blasting - results in explosives being detonated unconfined or with minimal confinement. As is the case with plaster shooting or popping which generates unwanted flyrock as an unfortunate by product, figure 8.19. The removal of toe using blasting can also result in flyrock as the hole depths are usually not large and even small charges can have insufficient stem length. The causes secondary blasting and the associated flyrock are:
    - oversize resulting from primary blasting
    - toe resulting from primary blasting
    - lack of confinement in boulder popping and toe removal
    - unconfined explosives detonating in plaster shooting

The techniques to minimise risk of flyrock from secondary blasting are:

- minimise the generation of oversize from primary blasting by using good blast design and minimising variation in the drill and blast process by controlling the drill and blast process using appropriate procedures and standards and auditing compliance to these

- minimise the generation of toe using the techniques mentioned for oversize
- use rock breakers to remove the resultant small amount of toe and oversize
- use blast mats and conservative blast designs (drill deeper than design to increase stem lengths) including crushed angular aggregate as stemming in toe shots
- use popping instead of plaster shooting.