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Evaluating the Viability of Pumpable Emulsion Explosives for use in Narrow Reef Mining Operations

A research report on a study presented to

The Department of Mining Engineering

Faculty of Engineering and the Built Environment

In partial fulfilment of the requirements

for the completion of an MSc(ENG)

By

SELWYN PEARTON

November, 2014

Declaration

I declare that "***Evaluating the Viability of Pumpable Emulsion Explosives for use in Narrow Reef Mining Operations"*** is my own work, that it has never been submitted for any examination or degree in any other university, and that all sources to which I have made reference or quoted have been indicated and acknowledged by complete references.

Selwyn Pearton
November 2014

Signed: .....

Abstract

Pumpable emulsion explosives have been available to surface and underground massive mining operations for decades and through their unique properties offer significant advantages through improved safety, reliability and performance. Despite their advantageous properties the benefits of pumpable emulsions have been unavailable to narrow reef mining operations due to the void in technology necessary for their successful implementation within the challenging environment.

The purpose of the following research report is to evaluate the viability of pumpable emulsion explosives for use in South African narrow reef mining operations. By approaching the problem from multiple perspectives, this research report aimed first to propose a theoretical framework and suite of equipment suitable for the implementation of pumpable emulsions within the narrow reef environment. Through the development of this equipment, controlled tests could be undertaken on the proposed narrow reef emulsion formulations and pumpable emulsion technology to obtain the necessary understanding of the performance of the system under controlled conditions. Once an understanding had been obtained for the controlled performance of the system, a field study could be undertaken in order to compare the performance of the new pumpable emulsion system with available explosives within the underground mining industry. Through the results obtained a comparison could be made of the blasting efficiency and feasibility of pumpable emulsions when compared to existing explosives within the narrow reef environment.

Given the vast difference in underground narrow reef and massive mining operations, fundamental changes were necessary with regard to emulsion technology prior to the successful implementation of the narrow reef emulsion system. Following the development of the system, controlled explosives tests allowed for optimisation of the pump technology as well as recommended daily practices. Blast results achieved in field trials with the narrow reef emulsion system allowed for improved blasting efficiency while reducing explosives costs per tonne broken. While blast results and efficiencies improved throughout pumpable emulsion trials, variances in daily blasting practices including the use of stemming, overcharging and varying priming practices influenced the results obtained. Further testing is therefore recommended in order to determine the influence of variables on the results obtained such that a specific increase in blasting efficiency can be determined.

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1 Introduction

This chapter will comprise an overview of the utilisation and advantages of bulk emulsion explosives in both large scale mining operations as well as narrow reef stoping environments. In addition, a brief rationale for the current study will be included.

Bulk emulsion explosives have been used in large scale mining operations across the globe for decades (Yeung, Player & Braddon, 1999). The reason for their extensive use lies primarily in the advantages of bulk emulsion explosives over alternative explosive technologies, both in terms of safety and blast performance (Yeung et al., 1999). Despite these benefits having been attainable for some time, the scale and cost of equipment required for the implementation of such technologies has resulted in their benefits being limited to large scale mining operations. This limitation to large scale mining has continued to exist, despite the ever growing demand for increased levels of safety and security in the narrow stope environment.

As it is not possible for a detonation front to propagate within an un-sensitised UN Class 5.1 pumpable emulsion, they are less sensitive than conventional explosives and are free from many of the regulations imposed on Class 1 explosives (Svard & Johansson, 2012). The increase in safety and security offered through this classification together with the physical properties of pumpable emulsions allow for significant advantages over available explosives technologies within the narrow reef environment. In addition to the improved safety during transportation, storage and handling, pumpable emulsions can be pumped between transport vessels, through underground pipelines and into the blasthole thereby reducing labour requirements. As pumpable emulsions have an oil matrix and are therefore insoluble in water, they have proven effective in wet mining operations reducing the occurrence of poor blast results in the presence of water as well as producing fewer noxious gases on detonation due to the increased intimacy between the fuel and oxidiser phases of the explosive (Svard & Johansson, 2012).

While this intimacy between the constituents of the explosive is beneficial to safety in that it reduces the emission of noxious gases, it is of equal importance in facilitating the comparatively high velocity of detonation (VOD) of pumpable emulsions. Through their high

VOD, pumpable emulsions deliver a greater percentage of their energy to the rock mass through shock energy as opposed to gas energy, increasing the fragmentation of ore and reducing overbreak and damage to surrounding rock strata (Rodgers, 1999; Yeung et al., 1999).

Despite efforts to develop a suite of equipment suitable for the implementation of pumpable emulsions in narrow reef stoping, little success has been achieved in developing the logistical and pump technologies vital to their successful implementation in the harsh South African narrow reef environment. Given the absence of suitable equipment, new technology was required in order to facilitate their successful implementation. While bulk emulsion technology has been successfully implemented in large scale underground operations, narrow reef operations face unique challenges different to those of larger operations. Despite efforts to improve levels of education within the work force, poor levels of education continue to impede productivity in narrow reef operations (Mosenthal, 1990). This burden coupled with harsh and confined conditions in the un-mechanised narrow reef necessitates a radical reduction in the cost and complexity of emulsion systems in both logistics and charging apparatus if they are to be successfully introduced. In addition to the fore mentioned technological challenges, limited research and data is available on the wide scale application of pumpable emulsion explosives in sub-36mm diameter blastholes, elevating the degree of uncertainty in the successful implementation of the proposed explosives technology.

As the above mentioned factors represent but a few of the challenges to be overcome, it was considered necessary to first undertake a theoretical study of the properties and performance characteristics of pumpable emulsions in order to evaluate possible downstream implications that could be introduced through their implementation. Given the possible benefits available through the implementation of pumpable emulsions within narrow reef operations, the research reported here aims to evaluate the practicality and performance of pumpable emulsions in stoping operations in order to gain an understanding of the factors essential to their successful implementation. Through this understanding the researcher aims to evaluate the feasibility of pumpable emulsions for use in narrow reef mining operations and in so doing develop a theoretical framework and suite of equipment suitable for their implementation within the South African narrow reef environment.

2 Literature Review

2.1 Explosives Selection for Narrow Reef Blasting Applications

Through recent decades, a range of 'new' commercial explosives has become available for use within the underground mining industry. In addition, the use of dynamite within South African operations has been eliminated and so the range of explosives now available is limited to ammonium nitrate (AN) based explosives. Considerable differences in physical properties and performance characteristics still exist between the various types of AN based explosives. As the results obtained in blasting operations are, in part, dependent on the physical properties and performance characteristics of the selected explosive, the optimal choice of explosive will differ depending on the blast design, the desired outcome of the blast and the geological and environmental conditions in which the blast takes place.

Areas in which physical properties have the greatest influence on blasting operations include:

- The resistance of the explosive to water. This will influence the possible use and reliability of an explosive in a wet blasting environment.
- The sensitivity and thus UN classification of the explosive. This will in turn influence operational safety as well as regulations applicable to the transportation, storage and handling of the explosive.
- The packaging of the explosive. Bulk transportation and handling of explosives allows for increased levels of labour efficiency, eliminates the costs associated with the packaging of explosives and increases explosive efficiency through increased coupling in the blasthole. However, the use of bulk explosives increases the propensity for waste and the overcharging of explosives if not adequately managed.
- The possible generation of noxious gases in the blast.

The performance characteristics of an explosive are dependent on the specific energy delivered by a fixed mass of explosive on detonation as well as the velocity of detonation (VOD), or the rate at which this energy is applied to the rock mass. The influence of these characteristics on blast results will be discussed later in the project report. The following section provides an understanding of the properties and performance characteristics of a range of commercial explosives applicable to this project report.

2.1.1 Dynamite

Despite the use of dynamites having been discontinued, a brief understanding of their performance characteristics is required for the purposes of comparison within this project report.

Dynamite was first patented by Alfred Nobel in 1864 after combining nitroglycerin with kieselguhr to form an explosive that was safer to handle than liquid nitroglycerin (International Society of Explosives Engineers (ISEE), 2011). While straight dynamite became accepted as the predominant explosive for use within commercial blasting applications, continued development by Ohlssen and Norbin led to the first patented use of ammonium nitrate in an explosive formulation in 1867 (de Graaf, 2010). As ammonia dynamite and ammonia gelignite were less expensive, safer to handle and had lower fume emission than straight dynamites, they replaced the use of straight dynamites in underground mining operations (ISEE, 2011).

Though the VOD of straight dynamite is comparatively high at 6000 m/s (Dyno Nobel, 2006), the inclusion of ammonium nitrate in Ammonia Dynamite and Ammonia Gelignite formulations reduced the intimacy of the fuel and oxidiser phases of the explosive, significantly reducing the VOD of the explosive (ISEE, 2011). VODs of typically used 'Ammon Dynamite' and 'Ammon Gelignite' formulations such as those used by Saffy (1961) measured 2200 m/s and 2400 m/s respectively. Though Ammon Gelignite was water resistant, Ammon Dynamites had poor water resistance and rigid stick packaging was used to protect them from the environment (ISEE, 1998). This in turn reduced the coupling of the explosive column within the blasthole (Saffy, 1961). As these explosives still required the presence of nitroglycerin in the formulation to allow for detonation, their sensitivity to external sources of energy remained comparatively high (Hustrulid, 1999; Canadian Industries limited (CIL), 1968).

2.1.2 ANFO

Though ammonium nitrate was first added to explosives formulations in 1867, the breakthrough in ammonium nitrite based explosives technology was only achieved in the latter part of the 1950's by Dr. Melvin Cook (ISEE, 1998). This included the invention of

Ammonium Nitrate Fuel Oil explosives (ANFO) and the later development of slurry explosives (ISEE, 1998).

ANFO was first introduced to South African underground tabular mining operations in 1963, and by 1975 accounted for approximately 60% of the commercial explosives consumption within the sector (Mosenthal, 1990). Not only did ANFO increase the level of safety of commercial explosives, but it was less expensive than dynamites, 'less arduous' to handle and as a result of its bulk form it allowed for 100% coupling within the blasthole, improving the efficiency of energy transfer from the blasthole into the surrounding rock mass (Mosenthal, 1990; Saffy, 1961). While the bulk nature of ANFO initially appeared to be beneficial to mining operations, it allowed for the uncontrollable overcharging of blastholes. This high mass of fully coupled low VOD explosive increased the extent of damage to the hanging wall and increased levels of overbreak experienced within operations (Cunningham, 1991; Mosenthal, 1990). Due to the hygroscopic nature of ANFO it is also less sensitive to detonation when exposed to humid conditions and its use will result in poor and inconsistent blast results in wet mines. The combination of these fore mentioned factors reduces the efficiency of blasting operations and increases the overall cost of mining due to undesirable blast results and high levels of explosives waste (Mosenthal, 1990).

Though ANFO is manufactured at an approximate bulk density of $0.8 g/cm^3$, the blow loading of ANFO in underground operations increases the density of the explosive thereby increasing the relative bulk strength (RBS) of the explosive. As the air pressure available at the time of loading determines the force at which the ANFO granules are propelled through the charging lance and into the hole, the crushed particle size and compaction of the prill within the blasthole will vary depending on the available air pressure, the strength of the prill and the loading technique used. Loaded densities achieved through the use of pneumatic loaders commonly range from 0.94 to 1.1 g/cm^3 depending on the abovementioned variables (Brinkmann, 1994; CIL, 1968). This high blow loaded density further exacerbates the problem of the overcharging of blastholes due to the increased energy within the blasthole (Kabongo, 1995). As ammonium nitrate crystals undergo a phase change at 32° Celsius, the control of shelf life is important in order to limit the degradation of ANFO through temperature cycling (Mulke, 1966). Repeated cycling of ANFO across 32° Celsius results in the degradation of the original prill and significantly increases the density achieved through pneumatic loading, as well as the quantity of ANFO blown into the air during loading operations (Mulke, 1966).

As these factors affect the density of blow loaded ANFO and this in turn affects the VOD, a broad and inconsistent range of VOD results will be experienced when using ANFO. Given the physical properties and high variability in performance characteristics of ANFO, its use is difficult to manage and its effects on the rock mass difficult to control. In an attempt to reduce the extent of damage caused by the overcharging of blastholes with ANFO, explosive manufacturers have attempted to reduce the relative bulk strength of ANFO (Kruger, 2010). Despite these efforts, limited success is evident and ANFO has been largely excluded from consideration in mines with poor ground conditions (Kruger, 2010).

2.1.3 Slurry Explosives

Watergel or Slurry explosives as they are also known, were first invented by Melvin A. Cook in 1958 and first implemented in South African open cast mines in 1968 (African Explosives Limited (AEL), 2012). Watergel explosives constitute 'a colloidal suspension of solid AN [Ammonium Nitrate] particles suspended in a liquid AN [Ammonium Nitrate] solution and gelled using cross linking agents' (Aimone, 1992). Gelling agents such as guar gum are used to thicken the explosive matrix while fuel oils are added to the matrix for detonation to take place (Aimone, 1992). In order to increase the sensitivity of watergel explosives, sensitising agents such as TNT, nitrostarch, Composition B, ethyl alcohol and glass micro-bubbles are added to the formulation, while aluminium is added to increase the energy released during detonation (Aimone, 1992).

Due to the comparatively poor intimacy of the fuel and oxidiser phases of watergel explosives, they have a lower VOD and thus a lower detonation pressure than emulsion explosives (Spiteri, 1998). Given the lower detonation pressure, the strain wave induced through the detonation of the charge will be lower than that of high VOD emulsion explosives, while the period of time in which high pressure gases act on the rock mass will be greater. Typical VOD values for small diameter watergel charges fall within the range of 3200 m/s to 3700 m/s (Brinkmann, 1990), while explosive densities may be as high as 1.35 g/cc allowing for a high energy concentration during loading (Spiteri, 1998). Due to the presence of water within the Watergel formulation, the resistance of watergel explosives to accidental initiation is significantly greater than that of dynamites (ISEE, 1998).

2.1.4 Emulsion Explosives

Emulsion explosives were first patented by Richard Egly and Albert Neckar in 1961 and first entered South African underground tabular mining operations in cartridged form in the early 1980s (AEL, 2012). Emulsion explosives are composed of two immiscible liquids with an aqueous oxidiser phase and a fuel oil phase making up the explosive. During the manufacturing process, the aqueous ammonium nitrate phase of the emulsion is divided repeatedly through a blending process forming microscopic droplets of oxidiser suspended within the oil matrix (ISEE, 1998). As a result of the microscopic size and even distribution of oxidiser droplets within the matrix, the intimacy of the oxidiser and fuel within emulsion explosives is higher than in ANFO and slurry explosives.

During the detonation of emulsion explosive, the high degree of intimacy between the two phases of the explosive allow for a faster reaction between the fuel and oxidiser thereby resulting in a higher VOD. As this allows for a more efficient reaction, smaller volumes of noxious gases are released during the detonation process (Svard & Johansson, 1999). Another benefit of the high intimacy between the fuel and oxidiser phases of emulsions is that the addition of mechanically or chemically induced gas bubbles is sufficient to sensitise the base emulsion to allow for detonation. For this reason no other sensitising chemicals need be added to an emulsion to allow detonation to take place and as a result, the resistance of emulsions to accidental initiation is substantially less than even that of watergel explosives (ISEE, 1998). Typical VOD values for emulsions in small diameter blastholes range from 4500 m/s to 5100 m/s (Brinkmann, 1990), with average density values in the region of 1.15 g/cc, to upper limits as high 1.35 g/cc (Spiteri, 1998). As emulsions are insoluble in water, they are ideal for use in wet mining operations where they are able to displace water within wet blast holes due to their high initial density.

2.2 The Theory of Explosives

2.2.1 Detonation Theory

In order to understand the way in which various explosives deliver their energy to the rock mass it is important to understand the theory of ideal and non-ideal detonation. Ideal detonation theory was originally developed by Chapman and Jouguet in the late 18th century (Brinkman, 1990). Chapman and Jouguet proposed that an ideal detonation front is a planer shock wave that travels through an explosive column heating the explosive at the detonation front to induce an exothermic chemical reaction. As all chemical products react within a short distance of the detonation front, all energy released in the detonation supports the detonation front. The reaction zone is therefore bounded at the back by the Chapman-Jouguet plane (CJ plane) as depicted in Figure 1. Behind the CJ plane it was assumed that stable gases at high pressure and temperature exist in the volume initially occupied by the explosive charge. As the rate of energy release and the confining media are not taken into consideration all energy released acts within the detonation front and no energy losses are taken into consideration (Uludag, 2005; Brinkman, 1990).

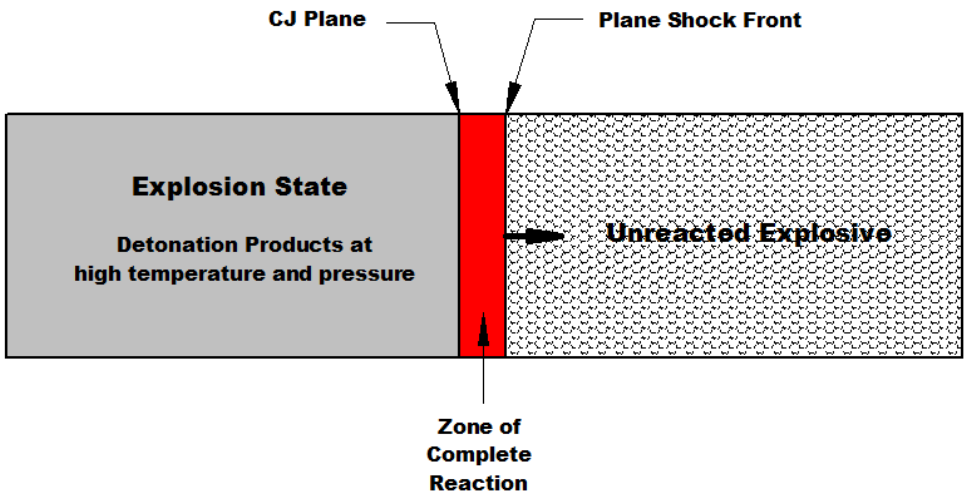


Figure 1, Ideal Detonation Front (Brinkmann, 1990)

In commercial explosives however, the intimacy of the oxidiser and fuel phases of the explosive formulation is often poor and as a result the chemical reaction continues for a much longer distance behind the detonation front. This type of detonation is termed a non-ideal detonation and factors such as the intimacy of the oxidiser and fuel phases, confining media, charge diameter, coupling ratio and density all play a role in the determination of the VOD and detonation pressure of the explosive (ISEE, 1998; Brinkmann, 1990). In non-ideal detonation theory, the reaction zone behind the detonation front is divided into a primary reaction zone (PRZ) and a secondary reaction zone. Energy released in the primary reaction zone supports the propagation of the detonation front while energy released in the secondary reaction zone prolongs the duration of high pressure gases within the blasthole (Brinkmann, 1990). In non-ideal detonation the shock front is curved due to reduced detonation pressure and reaction rates along the edge of explosive column as depicted in Figure 2. Energy release in the primary reaction zone can be as low as 10% or as high as 90%, depending on the properties of the explosive and the confining characteristics of the rock mass in which it is located (Brinkmann, 1990).

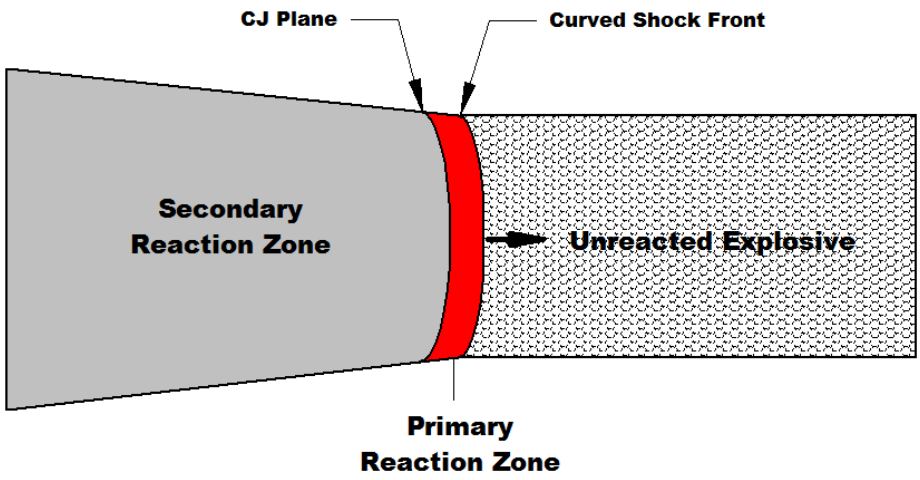


Figure 2, Non-ideal Detonation Theory (Brinkmann, 1990)

2.2.2 Principles of Explosive and Rock Interaction

According to Lownds (1991), the interaction of explosive energy with the rock mass through the detonation of an explosive charge can be divided into three distinct energy zones. These zones include the zones of shock energy, the zone of heave or gas energy and that of wasted energy. Figure 3 depicts the energy zones within a pressure volume graph where point A depicts the maximum explosion pressure on detonation of the explosive column, B represents the quasi-static equilibrium point at which the pressure in the blasthole is equal to the stress acting within the surrounding rock mass to return the blasthole to the original state and C, the point at which the products of detonation are released into the atmosphere. The gradient of line OB and thus the position of point B is dependent on rock stiffness and response of the rock mass to explosive loading. From the graph it is possible to calculate the energy delivered through the rock breaking mechanisms of shock (ABDO) and gas (BCED). Together, these regions represent the total energy acting on the rock mass through the area ACEO. As higher VOD explosives have a higher detonation pressure a greater percentage of the explosive energy will be delivered as shock energy and less as gas energy, while the opposite is true of lower VOD explosives (Lownds, 1991).

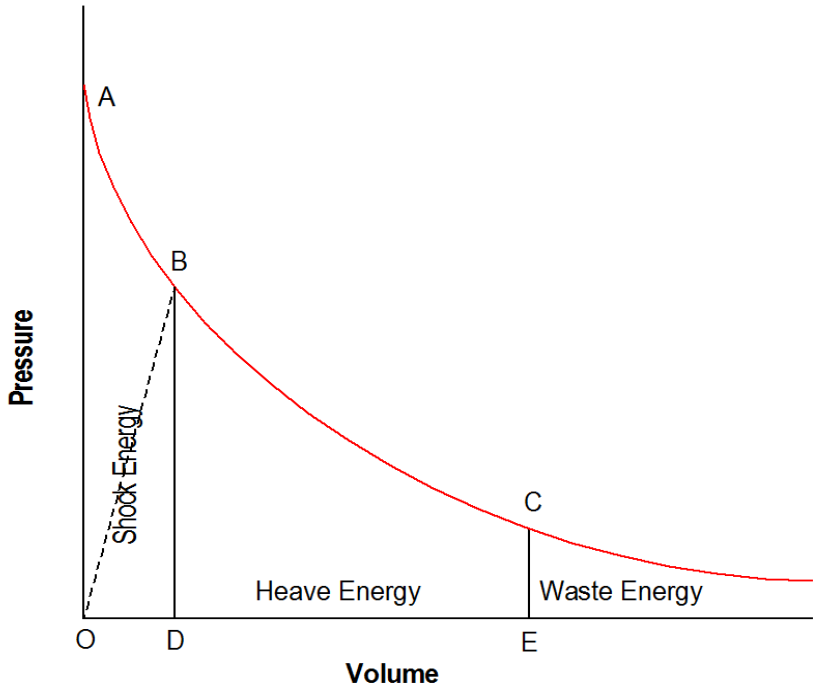


Figure 3, Energy partition in blasting (Lownds, 1991)

Through the rock breaking mechanisms of shock and gas, three zones of damage are formed around a blasthole prior to the interaction of the blasthole with the free face. These three zones include the *zone of crushing*, the *zone of radial cracking* and the *zone of crack extension*.

Zone of Crushing

Given the extreme shock radiated from the blasthole immediately on detonation, a zone of crushing is generated immediately around the blasthole. The diameter of the crushed zone is dependent on the magnitude of the radial strain emitted on detonation of the explosive column and the elastic limit or yield stress of the rock. When the maximum compressive stress no longer exceeds the elastic limit of the rock, crushing will cease to occur (Hustrulid, 1999; Uludag, 2005).

Zone of Radial Cracking

The zone of radial cracking occurs around the central zone of crushing. Given the high compressional stress generated in the radial direction away from the blasthole, a resultant tensile stress is induced tangential to the direction of the radial compressive stress. As rocks are weaker in tension than in compression, the high radial compression leads to the formation of tensile fractures leading radially away from the blasthole. These radial tensile fractures protrude outward from the blasthole (Hustrulid, 1999; Saffy, 1961). The cracks formed in this zone will extend outward until the tensile stress induced by the radial compressive stress is no longer large enough to overcome the tensile strength on the rock mass. As the magnitude of the radial compressive stress is proportional to the detonation pressure, the diameters of the 'zone of crushing' and the 'zone of radial cracking' will vary depending on VOD and density of the explosive (Cruise, 2010). Detonation pressure is described by Equation 1 below.

Equation 1 $P_d = \rho D^2 / 4$

Where: P_d is the detonation pressure

ρ is the density of the explosive (g/cc^3)

& D is the VOD measured in m/s (Persson et al, 1994)

Equation 2 $P_b = P_d / 2$

Where: P_b is the borehole pressure (Persson et al, 1994)

Zone of Crack Extension

Once the shock wave has acted on the rock mass, the high pressure gases remaining in the blasthole begin to work on the radial fractures induced by the radial compressive stress as well as pre-existing discontinuities within the rock mass. This borehole pressure prior to expansion of the blasthole can be calculated from Equation 2 above. This forms the 'zone of crack extension' (Cruise, 2010). As the pressure in the cracks increases, stress concentrations are formed at the protruding ends of the radial fractures causing them to extend further. During this gas phase of the explosive and rock interaction, the high stress field induced by the expanding explosive gases cause the displacement of the remaining unbroken burden toward the free face. The resulting strain in the direction of the free face induces bending moments in the burden causing flexural rupture of the rock against the free face (Uludag, 2005). Once the breakout of the burden is complete, the diameter of the affected area ranges between 40 to 50 multiples of the blasthole diameter (Cruise, 2010; Aimone, 1992).As different explosives have different VODs and densities, the detonation pressure produced on detonation of the explosive will differ according to Equation 1. As the detonation pressure is proportional to the square of the VOD, higher VOD explosives will generate significantly greater levels of shock than lower VOD explosives, maximising the radius of the zone of crushing and the zone of radial cracking. Low VOD explosives on the other hand, release less

shock energy into the rock mass, reducing the extent of damage in the zone of crushing and zone of radial cracking. The extreme gas pressure present in the hole at the time of crack extension dominates the breaking process, causing longer cracks to protrude away from the blasthole.

2.2.3 The Effect of Time on Rock Breaking Mechanisms

In order to gain a better understanding of the rock breaking mechanisms, it is necessary to consider the effect of time on the shock and gas mechanisms acting on the rock mass through detonation of a blasthole. Figure 4 demonstrates the pressure time curves for both low VOD and high VOD explosives. As the borehole pressure achieved in the 'explosion state' is proportional to the VOD of the explosive and equal to approximately half the detonation pressure, the borehole pressure for high VOD explosives is significantly greater than for low VOD explosives (Brinkmann, 1990). Through calculation, Brinkmann (1990) was able to determine that the borehole pressure achieved by high VOD explosives could exceed 5 times the borehole pressure of low VOD explosives for consistent mass and energy explosives. As the rate of flow for gases between two points is proportional to the differential pressure between the points (Serway, 2004), the significantly higher borehole pressure reached through the use of high VOD explosives will result in rapid ejection of the gases from the blasthole. Due to the high rate of gas ejection through elevated borehole pressures, the period of time in which gases remain in the blasthole is significantly shorter for higher VOD explosives than for lower VOD explosives.

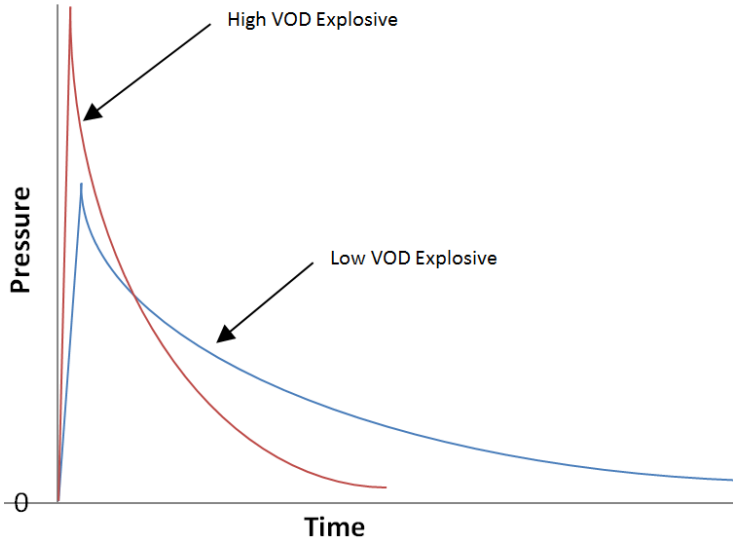


Figure 4, Comparative borehole pressures for higher and lower VOD explosives from time of detonation

As the VOD of an explosive is proportional to the detonation pressure, and as the detonation pressure is proportional to the strength of the shock wave emitted in the rock mass, the VOD of the explosive will influence the magnitude of strain induced fracturing in the rock mass. This was confirmed by Brinkmann (1994) who found through experiments that the magnitude of the strain wave in the rock mass was proportional to the VOD of the explosive in use. Results obtained in the strain tests undertaken by Brinkmann (1994) indicated that higher VOD explosives experienced a shorter rise time and higher peak amplitude more than 150% of those achieved by lower VOD explosives. In addition the wave period for high VOD explosives was significantly less than that experienced with lower VOD explosives.

Prior to Brinkmann, an experiment was undertaken by Saffy (1961) on the effect of explosive coupling on the magnitude of the strain wave generated in the rock mass. From the experiment, Saffy plotted the strain wave induced in the rock mass at increasing distance from the hole depicting the initial positive or radial compressive strain present in the rock mass, prior to the negative or tensile strain induced as the wave passed. The simplistic wave form is depicted in Figure 5.

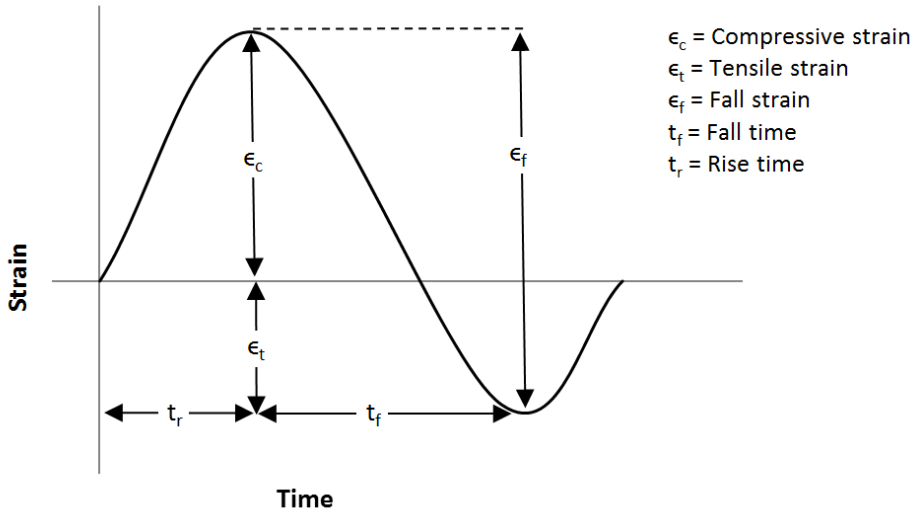


Figure 5, Shape of a measured strain wave generated through the detonation of a blasthole (Saffy, 1961)

During experimentation, Saffy (1961) measured the period and amplitude of the radial compressive strain present in the rock mass at set distances from the blasthole. By increasing the diameter of the blasthole while maintaining a fixed mass of explosive and charge diameter, Saffy was able to conclude that as the coupling in the blasthole decreased, so did the amplitude and period of the radial compressive strain wave induced in the surrounding rock mass. Also evident in the results was the exponential decay of maximum radial compressive strain as the distance from the blasthole increased, together with an increase in the period of the wave with distance from the blasthole (Rorke, 2010; Saffy, 1961). Hino (1956) as cited in Saffy (1961) proposed that the degree of fragmentation of the panel was a result of the amplitude of the strain wave, while the volume of rock broken or the ability to break out a burden was a dependent on the period of the strain wave or energy below the pressure time curve.

As varying rock types have varying p-wave velocities, the degree to which constructive interference can take place will depend on the ratio between the VOD of the explosive and the p-wave velocity of the rock mass. As the p-wave velocity of a specific rock is fixed, it is theoretically possible to maximise the strain pulse from the explosive column by selecting an explosive with a VOD similar to the p-wave velocity of the rock. As the tensile strength of a

specific rock type is lower than the compressive strength of the same rock type, less strain is required for rock to fail in tension than in compression. For this reason, a greater volume of rock adjacent to a free face will be broken by the reflection of a tensile strain wave, than by compressive strain (Cruise, 2010).

From the above discussion it is evident that despite similar quantities of gas and energy released through the detonation of both high VOD and low VOD ammonium nitrate explosives, the breaking effects of the explosives differ considerably. The breaking mechanisms of shock and gas could better be described as products of the efficiency of detonation, resulting in higher or lower detonation pressure and borehole pressure. These two pressures are the determining factors in the magnitude of the strain wave in the rock mass due to the rate of pressure increase and the rate at which high pressure gases are expelled from the blasthole as a result of borehole pressure. Borehole pressure together with resistance to ejection therefore determine the period of time available for the high pressure gases within the blasthole to act on the rock mass.

Ammon Dynamite represents the lower end of the spectrum for lower VOD commercial explosives, while emulsions represent the upper end of the spectrum for higher VOD commercial explosives (Brinkmann, 1990). The lower borehole pressure realised through lower VOD explosives results in slower ejection of gases from the blasthole, allowing greater time for heave or gas energy to act on the rock mass surrounding the blasthole. As proposed by Brinkmann (1990) and depicted in Figure 6, this may allow lower VOD explosives the ability to break out larger burdens in specific rock types owing to longer exposure of the rock mass to high pressure gases within the blasthole. While this may prove advantageous in certain mining practices, it can also present problems in poor ground conditions where gas energy is able to penetrate the hangingwall (Kruger, 2010; Brinkmann, 1994).

Higher VOD explosives generate significantly greater detonation and borehole pressures on detonation, increasing the amplitude of the strain wave in the rock mass but reducing the period of time available for high pressure gases to penetrate the rock mass. As a result, higher VOD explosives produce finer fragmentation than lower VOD explosives and allow for less gas penetration of the rock mass owing to accelerated decay in borehole pressure. This may in turn reduce the ability of high VOD explosives to break out larger burdens (Kruger, 2010; de Graaf 2010) and proposed by Brinkmann (1990). It has also been proposed that a larger '*zone of crushing*' around higher VOD blastholes may be a contributing factor to the

prevention of high pressure gases penetrating the rock (de Graaf, 2010). Where ground conditions are a concern, higher VOD lower energy explosives can be used to prevent the penetration of high pressure gases into existing fractures in the rock mass. This should be done in conjunction with reduced burden size and smaller blasthole diameters.

Tests undertaken by Brinkmann (1994) on the influence of high and low VOD explosives on the fragmentation of rock, confirmed that the higher the VOD of the explosive, the finer the fragmentation produced in the blast. Further testing by Brinkmann (1990) confirmed that while shock was a significant factor in the fracture of the rock mass, it was unable to assist in the breakout of larger burdens that could be broken with lower VOD explosives. In the experiment Brinkmann compared a high VOD cartridged emulsion with a VOD range of 4500 to 5100 m/s with a low VOD Dynamite with a VOD range of 1800 to 2500 m/s. In order to allow for comparable energy values, the cartridged emulsion was loaded with a coupling ratio of 80% to 90% while the high density dynamite was loaded with a coupling ratio of 70% to 80%. The results of the test are indicated in Figure 6 below where each data point represents the average of at least five blasts.

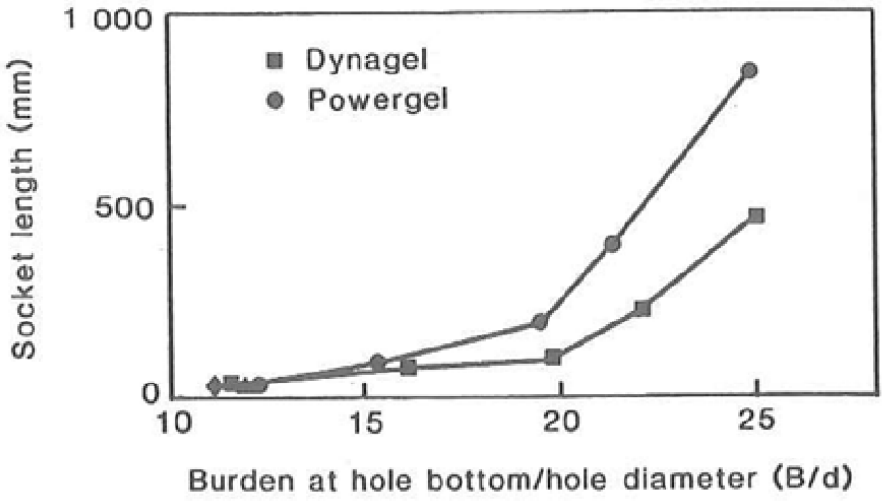


Figure 6, Comparison of socket length blasted with higher and lower VOD explosives with consistent energy (Brinkmann, 1990)

2.3 The Effect of Loading Practices on Explosives Performance

2.3.1 Influence of Explosive Properties on the Velocity of Detonation

As discussed in section 2.2.2, the VOD of an explosive plays a significant role in determining the detonation pressure and borehole pressure reached on detonation of a blasthole. One of the most significant factors dictating the VOD of an explosive is the physical intimacy of the oxidiser and fuel phases of the explosive (ISEE 1998). As ANFO and dynamite have a larger particle size, the intimacy of the fuel and oxidiser phases of the explosive is relatively poor, resulting in a lower VOD (de Graaf, 2010; Brinkmann, 1994). As emulsion explosives on the contrary have droplet sizes only microns in diameter, the intimacy of the reacting components is high, resulting in a higher VOD (ISEE, 1998; Brinkmann, 1994).

A second consideration in the physical make-up of an explosive is the distribution of gas voids within the explosive column. As commercial explosives are comparatively insensitive when compared to primary explosives, gas voids are required within the explosive to allow for the propagation of 'hot spots' during the detonation of the explosive (Rorke, 2010). As adiabatic compression of these gas voids is required for the detonation of the explosive, the smaller the size of the voids and the greater the number of voids (within a constant density explosive), the higher the VOD will be (Pathak, 2010). This is due to the increased rate of compression of the individual voids to reach the required reaction temperature, as well as the increase in surface area of the voids in contact with the surrounding explosive.

Additional factors that increase the steady state VOD of explosives include an increase in the density of the explosive, an increase in the confinement of the charge and an increase in the diameter of the explosive column (Chapman & Jouguet). As the diameter of a non-ideal explosive column approaches zero, the reduced pressure and reaction rate at the edge of the column begin to have a greater influence on the VOD of the explosive. This results in a critical diameter below which the explosive is unable to detonate. As the level of confinement increases, the critical diameter decreases. This is due to the prevention of lateral expansion of gases at the edge of the explosive column maintaining the level of temperature and pressure driving the detonation front (Chapman & Jouguet).

As no data could be found in the public domain for the VOD of pumpable emulsions in small diameter holes, the current research question was approached from two angles. As cartridged emulsions exhibit a comparatively high VOD, the probability of the pumpable emulsion

exhibiting a similarly high VOD was good. Differences that needed to be taken into consideration however include the higher water content of pumpable emulsions when compared to cartridged emulsions as well as the non-ideal manufacturing conditions under which pumpable emulsions are sensitised. As cartridged emulsions are manufactured under controlled manufacturing conditions and pumpable emulsions sensitised at the blastface, the consistency of the pumpable emulsion may be influenced by operating conditions. As the only other bulk explosive system in use in small diameter holes is ANFO, a theoretical consideration is included of its application and consistency for comparison with pumpable emulsions later in this project report. Figure 7 depicts a range of VOD values measured for ANFO by Thomas, Mulke, Brinkmann and Rorke under varying confinement and density.

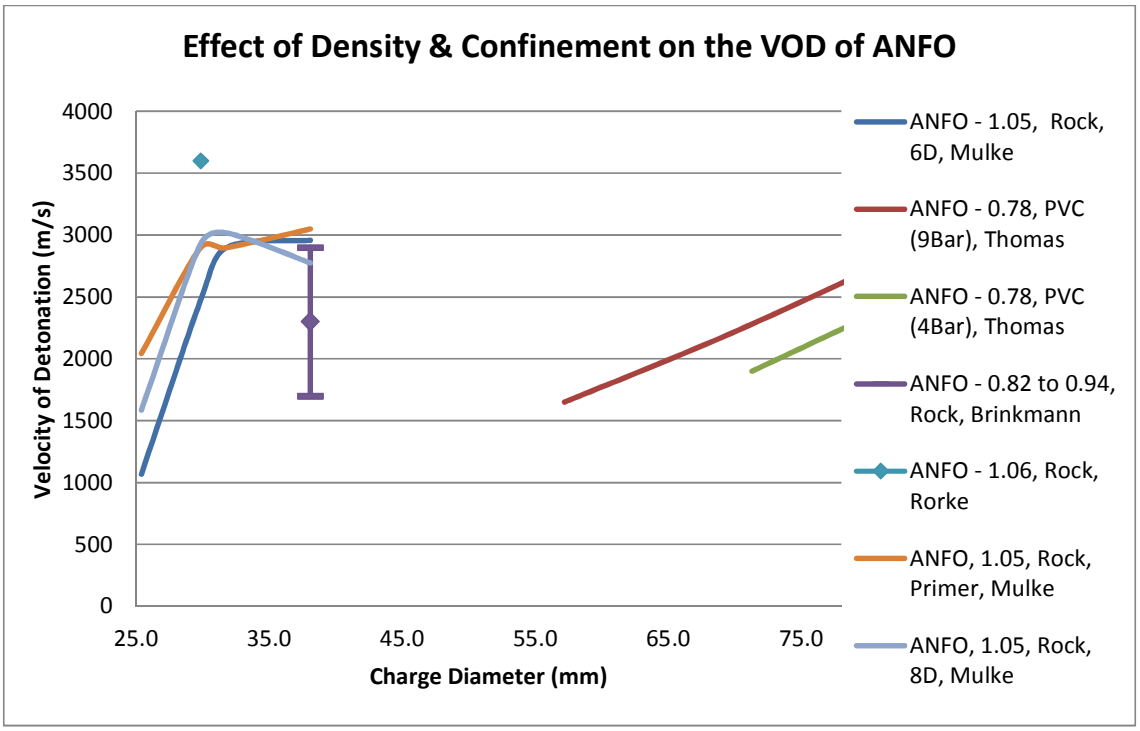


Figure 7, Effect of density and confinement on the VOD of ANFO (Thomas, 2005; Rorke, 1996; Brinkmann, 1990; Mulke, 1966)

It is evident from Figure 7 that the degree to which changes in the loading density and confinement of ANFO influence the VOD. This is a challenge to the use of bulk explosives systems as the in-hole density of the explosive can be influenced by a range of variables

present prior to and during loading. As VOD tests are often undertaken in a controlled environment, the ability to achieve constant density charges within the production environment is likely to be even less consistent than those depicted in Figure 7. Results obtained by Mulke (1966) indicate the difficulty in controlling the loaded density of ANFO in small diameter holes. This is illustrated in Figure 8 through a plot of the coefficient of variance for a range of small diameter blastholes. This was later supported by Rorke (1996) who achieved a greater standard deviation for small diameter test pipes than for larger pipes loaded with ANFO under controlled loading conditions. As the diameter of an explosive charge nears the critical diameter, the edge effects of non-ideal explosives and inconsistencies in loading begin to have a greater influence on VOD. As such, a greater variation in results will be noticed with decreasing charge diameter.

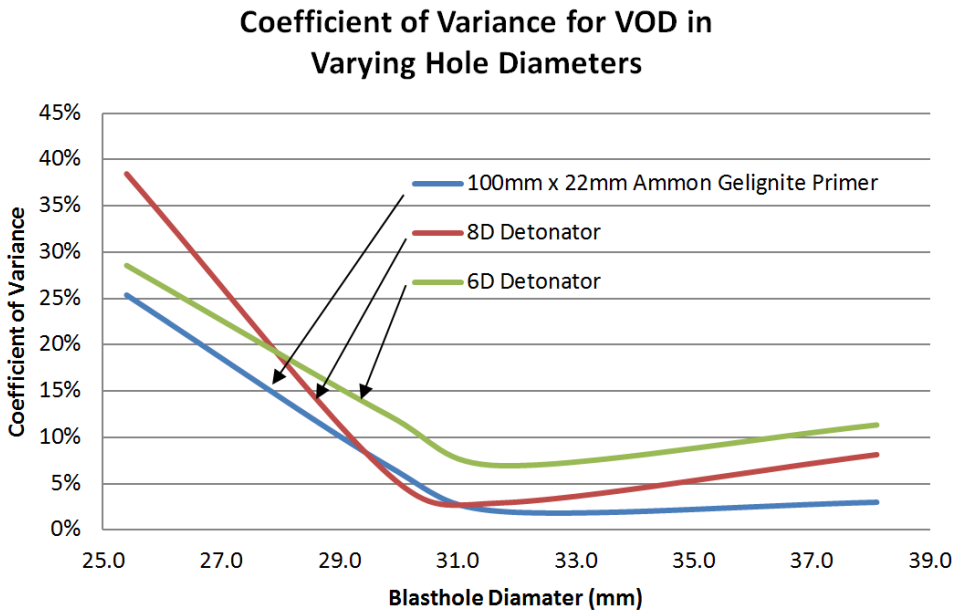


Figure 8, Coefficient of variance for VOD in increasing hole diameters (Mulke, 1966)

2.3.2 The Effect of Priming on VOD

In short blastholes such as those used in underground narrow reef operations, the effects of priming are an important consideration in the VOD achieved in a blasthole, as the initial run-up in VOD occurs at the toe of the blasthole. As the toe of the blasthole is the most confined area within a blast design (Brinkmann, 1994), the shock and gas energy released at this position of the blasthole is crucial to the advance of the blast. As poorly initiated holes require a number of borehole diameters to build up to the steady state VOD, the desired VOD will only be achieved toward the end of short explosive columns (de Graaf, 2010; ISEE, 1998). Conversely, should high VOD and mass priming charges be used, a period of overrun will be experienced in the VOD before it falls to steady state velocity (ISEE, 1998). This trend is depicted in Figure 9 with increasing primer mass.

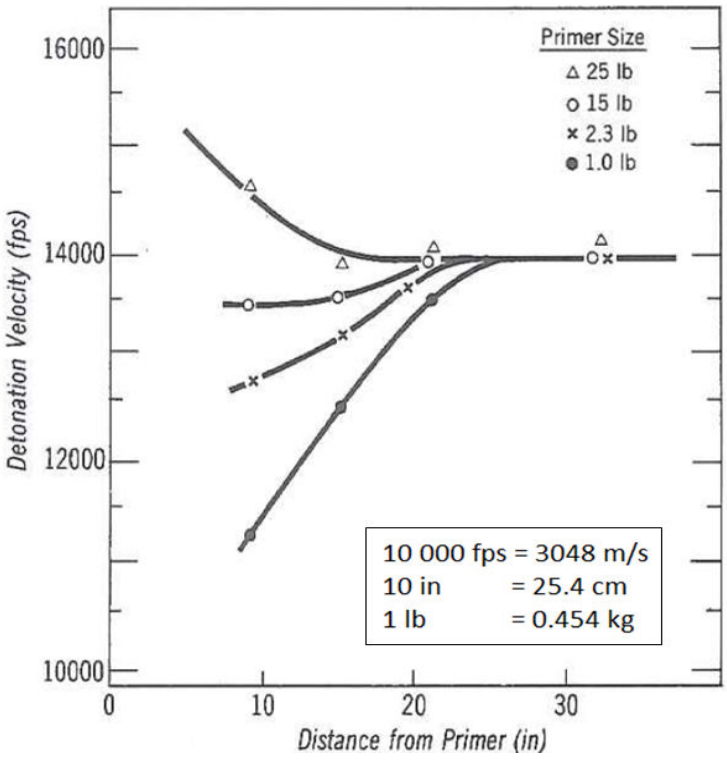


Figure 9, Effect of primer size on the run up of VOD in ANFO (230mm pipe diameter) (ISEE, 1998)

In modern commercial priming applications, the use of pentolite boosters (composed of PETN and TNT) is common practice. As pentolite has a VOD of 7000 m/s, the VOD of the explosive surrounding the booster is driven higher than would have occurred with a lower VOD primer (de Graaf, 2010). In order to achieve the maximum possible VOD, the diameter of the booster should be as close to the diameter of the blasthole as possible (de Graaf, 2010; Brinkmann, 1990). During the placement of an explosive charge, the detonator used to initiate a charge should point in the direction of the column, such that it imparts the greatest possible energy in the direction of detonation (Conradie 2006; ISEE, 1998). Experiments undertaken by Arnold (1954) as cited in Mulke (1966) support this principle as he found that the time required for the explosive to reach maximum detonation velocity was of a shorter duration through the implementation of this practice. In the South African narrow reef mining industry blastholes are seldom primed toward the centre of the charge due to the dangers of reverse priming blastholes. Should a misfire occur with the detonator reverse primed at the toe of the blasthole, special implements may be required in order to extract the detonator from the blasthole should it become wedged in the hole. For this reason, detonators are forward primed at the base of the blasthole.

In an attempt to minimise the length required for the build-up in VOD with the use of pumpable emulsions, this current research investigation considers the plausibility of collar priming such that the detonator could be directed toward the centre of the charge. While collar priming would allow for the detonator to point in the direction of the explosive column, three problems were identified with regard to its use:

- As the explosive column begins detonating at the collar of the hole, the period of time for which the explosion gases would act on the rock mass would be reduced due to early ejection of gases from the blasthole (Mulke, 1966).
- Secondly, constructive interference of strain waves produced on the detonation of the explosive column would compound in magnitude toward the toe of the blasthole. As rock is more easily fragmented by strain waves reflecting from a free face, less fracturing of the rock would occur due to the lack of a free face at the toe of the blasthole (Saffy, 1961).
- Lastly, as discontinuities in the blastface can lead to ‘back break’ across the collar of sequential blastholes, the possibility of extracting detonators from blastholes prior to detonation is greater if the detonator is positioned in the collar of the blasthole. This would lead to misfires in the blast and result in the loss of advance.

For these reasons the possibility of collar priming in short blastholes was eliminated and no attempt was made to use this practice in underground emulsions trials.

2.3.3 The Effect of Coupling on Explosive Performance

The coupling ratio of an explosive column is described as the average cross sectional area of the explosive column divided by the average cross sectional area of the blasthole. Bulk explosive products achieve 100% coupling within the blasthole as they are poured, blown or pumped into the blasthole and fill the hole in entirety from the toe of the hole. Rigid and semi-rigid explosives on the other hand are unable to adapt their shape to allow for 100% coupling within the blasthole. The coupling ratio for a rigid explosive cartridge will decrease with increasing hole diameter, while the coupling ratio for semi-rigid cartridge products is dependent of a number of variables within the manufacturing and loading of the explosive. These factors include the manufactured stiffness of the cartridge, the thickness and strength of wrapping or sleeving material of the cartridge, the elevation of the operation where it is used, the temperature at time of use, the ratio of the manufactured cartridge size to hole size and specific operator practices such as the force applied to tamp the cartridge and the number of cartridges inserted in a hole prior to tamping (ISEE, 1998; Saffy, 1961). In the majority of blasting applications, a high coupling ratio is desirable as it increases the mass of explosive at the toe of the blasthole. This is important as the toe of the blasthole experiences the greatest confinement in the blast and as such requires a higher level of energy to be broken out than does the collar of the round (Brinkmann, 1990).

An experiment undertaken by Saffy (1961) on the effect of coupling ratio on the strength of strain waves emitted from a blasthole revealed that the loss of mass placement per unit length was not the only inefficiency caused by a reduced coupling ratio. In the experiment, Saffy (1961) used a constant charge mass of 25.4mm (one inch) diameter Ammon Dynamite stick explosives while steadily increasing the diameter of blastholes such that the coupling ratio in each test was steadily reduced. In order to record the readings in the rock mass, strain gauges were grouted into the rock at fixed distances from the blasthole. As a baseline for comparison purposes, Saffy used equivalent strength and similar VOD semi-rigid Ammon Gelignite explosive cartridges. As this is a semi-rigid explosive, it could be tamped to achieve near full coupling within the blasthole as the diameter of blastholes were increased. The results obtained by Saffy (1961) are plotted in Figure 10.

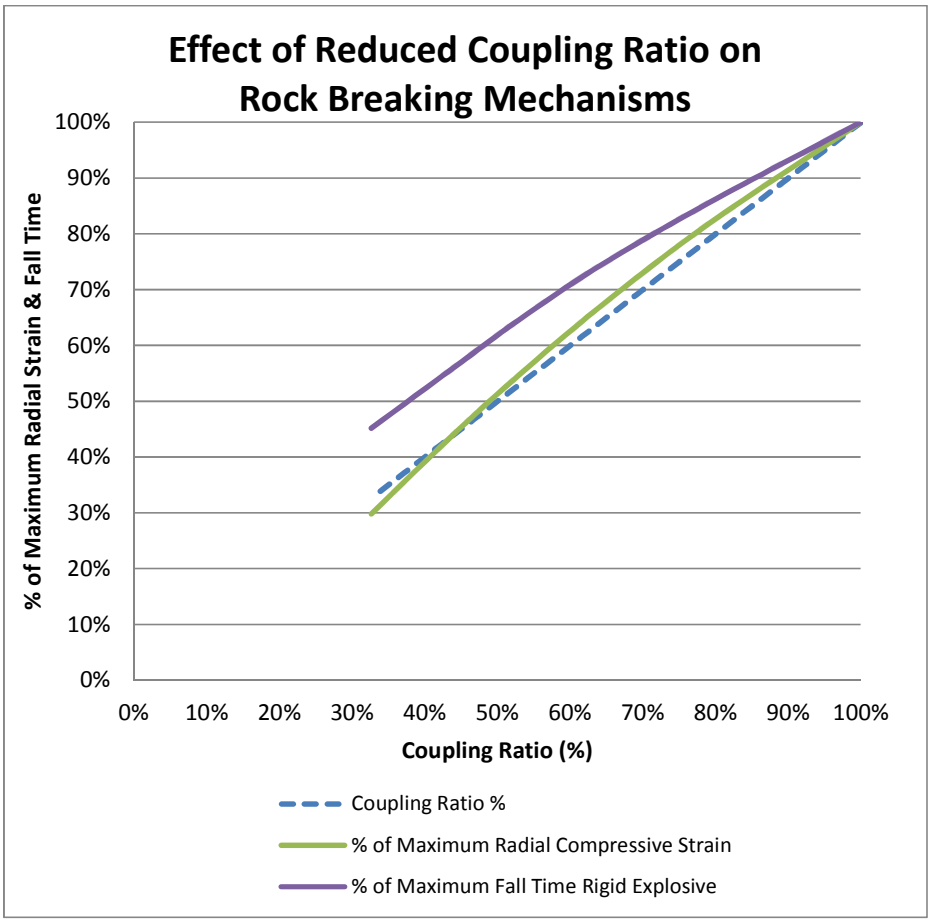


Figure 10, Effect of coupling ratio on Radial Compressive Strain and Fall Time (Saffy, 1961)

From the test results, Saffy (1961) was able to conclude that the coupling ratio of an explosives column had a significant impact on the ability of the explosive to transmit a compressive strain wave to the rock mass. In the case of radial compressive strain, Saffy (1961) found that the decrease in the maximum amplitude of the radial compressive strain was directly proportional to the coupling ratio of the explosive. In the base case semi-rigid gelignite cartridges were tamped to fill the hole and no reduction in radial compressive strain was noted with increasing blasthole diameter.

In non-ideal detonation theory, energy release within the primary reaction zone contributes to the VOD and detonation pressure of an explosive, while the energy within the secondary reaction zone is responsible for continued gas pressure in the blasthole (Brinkmann, 1990). Brinkmann (1990) approximated the energy released in the primary reaction zone through the following equation:

Equation 3 $Q_p = Q_t \times (Da/Di)^2$

Where: Q_p is the energy released in the primary reaction zone (MJ)

Q_t is the total explosive energy (MJ)

D_a is the measured VOD (m/s)

D_i is the ideal VOD (m/s)

As the detonation pressure of an explosive is largely responsible for the radial compressive strain (shock wave) in the rock mass (Brinkmann, 1990), the size of the strain wave is proportional to the quantity of energy released in the primary reaction zone. As Saffy (1961) demonstrated that coupling ratio is directly proportional to the magnitude of the radial compressive strain in the rock mass, Equation 3 can be rewritten to approximate the energy responsible for inducing the radial compressive strain in the rock mass:

As: $E_s = Q_p \times R$

Equation 4 $E_s = (Q_t \times (Da/Di)^2) \times R$

Where: R is the coupling ratio between the explosive column and rock mass

E_s is the energy used to induce radial compressive strain in the rock mass

From Equation 4, it is possible to approximate the relative energy utilised in inducing the radial compressive strain wave in the rock mass. Figure 11 below compares the E_s calculated for a pumpable emulsion (VOD 3600 m/s) with a relative bulk strength (RBS) of 132 and a high energy cartridged emulsion (VOD 4500 m/s) with an RBS of 153. As the coupling ratio

for the cartridged emulsion will depend on the variables discussed earlier, common coupling ratios for cartridged emulsions seldom exceed 85%. From Figure 11, it is evident that despite the 16% greater RBS and 25% higher VOD of the cartridged emulsion, the total shock energy acting on the rock mass will be similar for the two explosives. This is evident in the equivalent loaded energy per metre of blasthole due to the 'loss in mass', as well as the 'loss in shock' energy transmitted to the rock mass due to decoupling of the charge (Saffy, 1961). The overall loss in energy required for the generation of the strain wave in the rock mass is due to the compounding inefficiencies caused by the use of a decoupled charge.

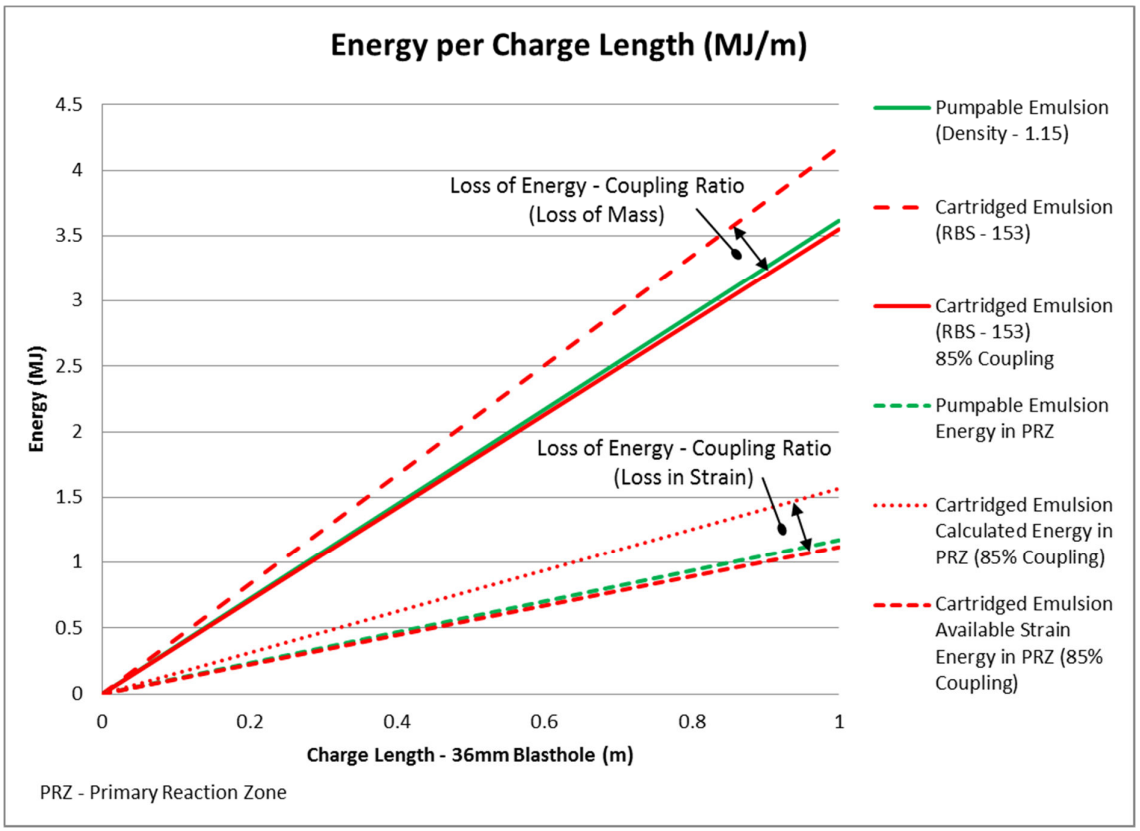


Figure 11, Theoretical energy comparison of a fully coupled bulk emulsion charge with a decoupled cartridged emulsion in 36mm blastholes.

- Cartridged emulsion mass loss is calculated with a coupling ratio of 85%
- Shock energy in the PRZ is calculated through Equation 3 (Brinkmann, 1990)
- Loss in shock energy delivered to the rock mass from the PRZ is calculated through Equation 4

From this comparison it can be seen that the efficiency of bulk emulsions is far greater than that of cartridged emulsions due to the inefficiencies of decoupled charges within the blasthole. In order to achieve equivalent blast results with both explosives systems, a cartridged emulsion of considerably greater strength is required.

2.3.4 The Effect of Stemming on Explosive Performance

Stemming products are used in commercial blasting applications in order to prevent the early ejection of high pressure gases from the blasthole, thereby allowing maximum work to be done by the high pressure gases produced on detonation of the explosive charge (Brits, 2010). The longer the period of time that high pressure gases are applied to pre-existing and blast generated fractures in the rock mass, the further the cracks will extend and the larger the burden that can be broken and finer fragmentation achieved through a fixed mass of explosive. This increases the efficiency of the blast and allows for a reduction in the mass of explosive required to break a specific burden (Saffy, 1961). Additional benefits of stemming suggested by Brits (2010) include a reduction in airblast, a possible increase in burden size, an increase in the fragmentation of ore and better throw from the blast improving muckpile profile and reducing cleaning time in stoping operations.

Tests undertaken by Brinkmann (1994) revealed that ‘up to 50% of the explosive energy [released in the blasthole] was carried away by gases venting from the collar when no stemming material [was] used’ (Brinkmann, 1994). Through the use of a single clay stemming plug, Brinkmann (1994) was able to increase the time to ejection of the stemming material by approximately 4.5 times from the time of detonation. This reduced the energy vented from the blasthole to as little as 15% of the energy generated in the blast (Brinkmann, 1994). This is evident in Figure 12 where peak pressures measured at the collar of the blasthole were reduced from 70MPa to 20Mpa, and the time between detonation and the expulsion of gases from below 0.5 milliseconds to in excess of 2 milliseconds. As the time required for the burden to be broken away from the solid is approximately 2 to 3 milliseconds (Brinkmann, 1994), the use of stemming is vital in achieving maximum blast efficiency.

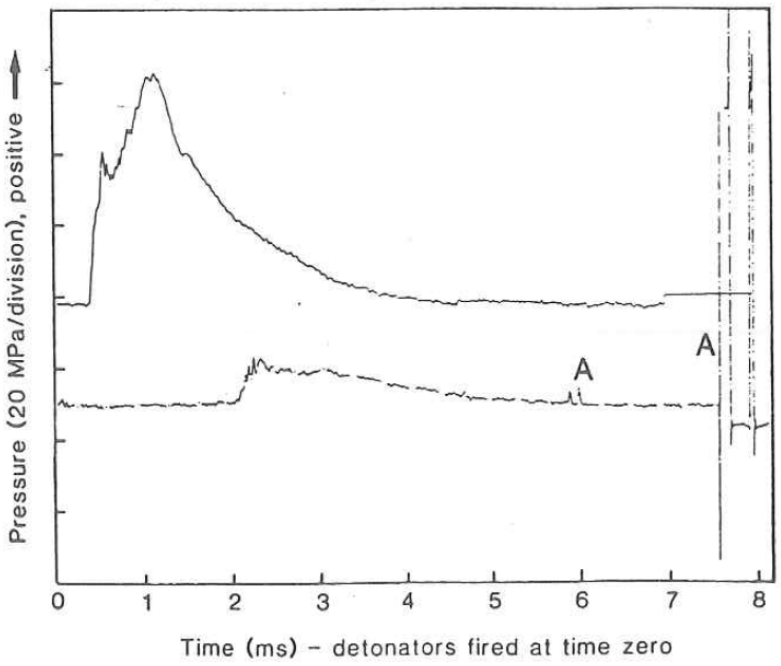


Figure 12, Exhaust pressure measured at the blast hole collar, single-hole blast with emulsion explosive. Trends on vertical axis offset for comparison (Upper un-stemmed; Lower clay stemmed.) (Brinkmann, 1994)

In order to evaluate the performance of different stemming materials, experiments undertaken by Brits (2010) measured the time to ejection for various stemming materials from small diameter blastholes. Brits (2010) loaded 1.4 kilograms of cartridged emulsion explosives into each 40mm blasthole prior to tamping with an increasing number of clay and plastic stemming products (6 different products were tested). Electronic detonators were fired simultaneously in and above each blasthole such that the time to ejection could be monitored on high speed camera. Figure 13 depicts a blasthole fired with no stemming material. It is evident from the picture that the ejection of high pressure gas and flame from the blasthole occurs almost immediately (Brits, 2010).

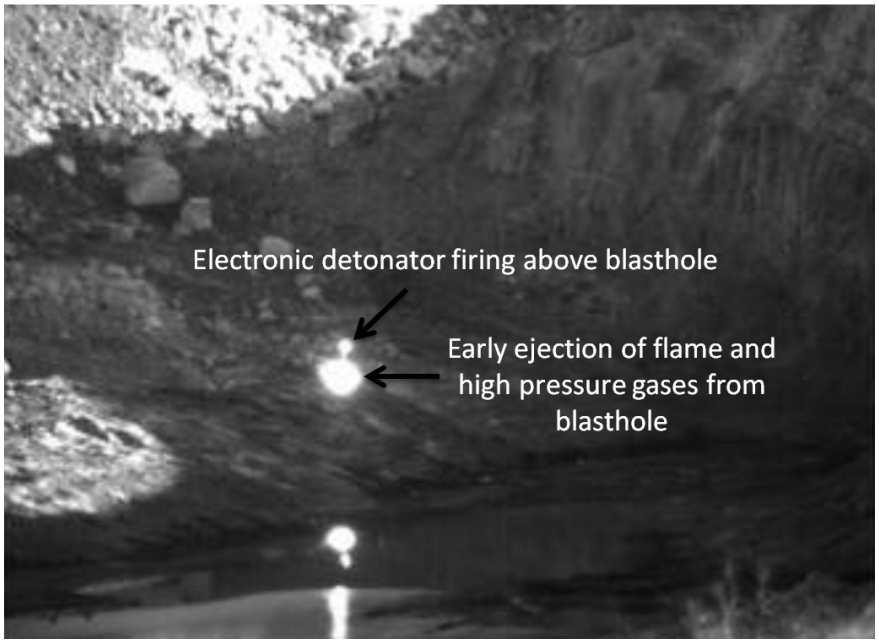


Figure 13, Blasthole detonated with no stemming material. Electronic detonators fired simultaneously inside explosive column and above the blasthole (Brits, 2010)

Results obtained by Brits revealed the following trends (Brits, 2010):

- A significant quantity of gas and flame is exhausted from un-stemmed blastholes immediately on detonation
- Single plastic stemming plugs had no noticeable effect with gas and flame being exhausted from the collar of the hole immediately on detonation
- The use of three or more plastic stemming plugs allowed for venting of the blasthole 2 milliseconds after initiation.
- The use of a single clay stemming plug prevented the emission of flame but the ejection of gases occurred almost immediately after initiation.
- The use of two clay stemming plugs prevented the venting of both flame and gas from the collar of the blasthole

Effect of Stemming on Radial Compressive Strain

In order to quantify the rock breaking mechanisms of shock and gas energy, Brinkmann (1994) used a stainless steel liner within the blasthole to prevent the gases produced in the blasthole from penetrating the rock mass. Figure 14 and Figure 15 indicate the radial compressive strain measured by Brinkmann at a distance of 600mm from a 38mm charge in lined and unlined blastholes. The blastholes were fired with dynamite and cartridged emulsion explosives respectively (Brinkmann, 1994).

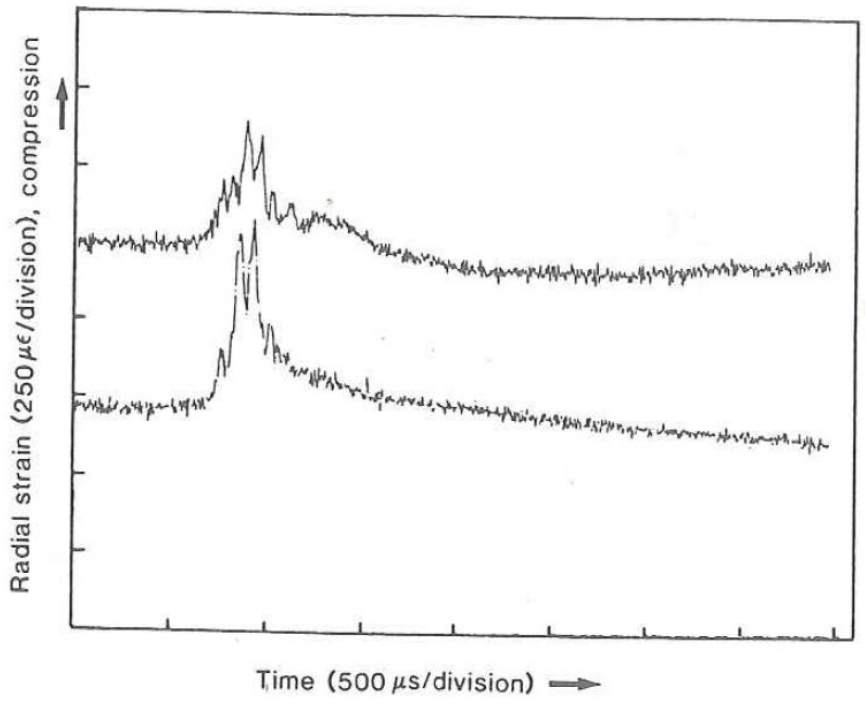


Figure 14, Radial strain measured at a distance of 600mm from the blasthole using dynamite. Trends on vertical axis offset for comparison (Upper - Normal blasthole, Lower - Lined blasthole) (Brinkmann, 1994)

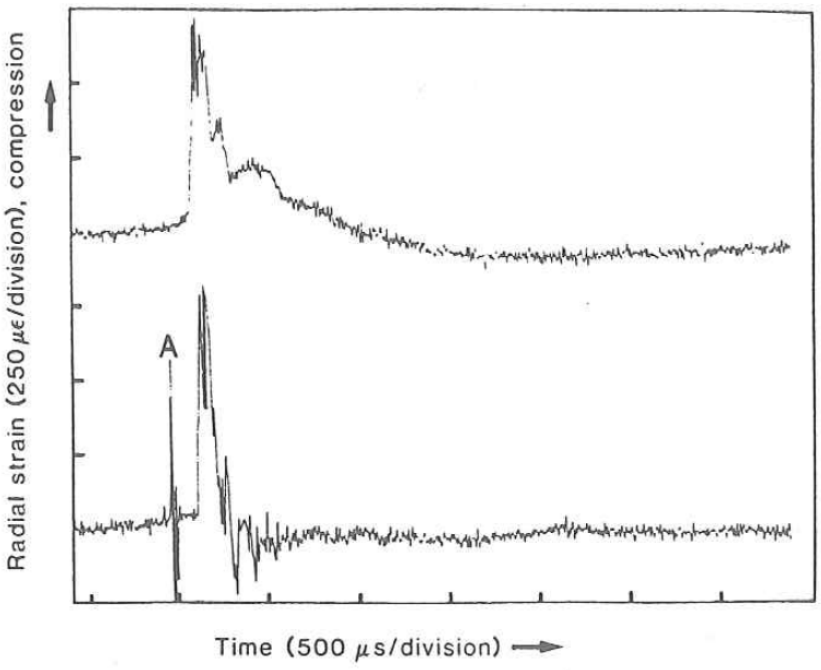


Figure 15, Radial strain measured at a distance of 600mm from the blasthole using cartridged emulsion. (Upper - Normal blasthole, Lower - Lined blasthole, A - Electrical noise) (Brinkmann, 1994)

From Brinkmann's (1994) results it is evident that the amplitude of the shock wave produced in the cartridged emulsion blast is significantly greater than that of the dynamite blast, while the period of the shock wave is significantly shorter. When comparing the lined and unlined trends of the same explosive type however, it is evident that the shock wave produced in the primary reaction zone is not the only mechanism driving the compressive strain wave in the rock mass. It is evident from the results for both dynamite and emulsion that when the blasthole liner is not used, a prolonged fall in the strain wave is experienced due to high pressure gases acting on and within the rock surrounding the blasthole. The results obtained by Saffy (1961) for a well tamped 'fully coupled' gelignite charge of fixed mass in increasing hole diameters supports this trend. During the baseline test, Saffy (1961) utilised a fixed 30cm (1 foot) length of 'dump sand' stemming material to stem blastholes. As the effectiveness of a fixed length of stemming is inversely proportional to the diameter of the blasthole, the level of confinement of the charge due to stemming will have decreased with increasing hole diameter. In addition, as the diameter of the hole continued to increase, the collar size or 'exhaust port' for the venting of high pressure gases from within the blasthole

would also have increased. Together, these two considerations would have allowed for faster venting of the high pressure gases from within the blasthole and therefore a faster decay in the pressure on the walls of the blasthole. Despite consistent readings for maximum fall strain and maximum radial compressive strain measured at a distance of 230mm (9 inches) from the blasthole, the period of the strain wave decreased with increasing hole diameter. This is indicated in Figure 16. From this discussion it is evident that through the use of stemming, the time from initiation to venting of a blasthole can be prolonged and the period of the strain wave lengthened.

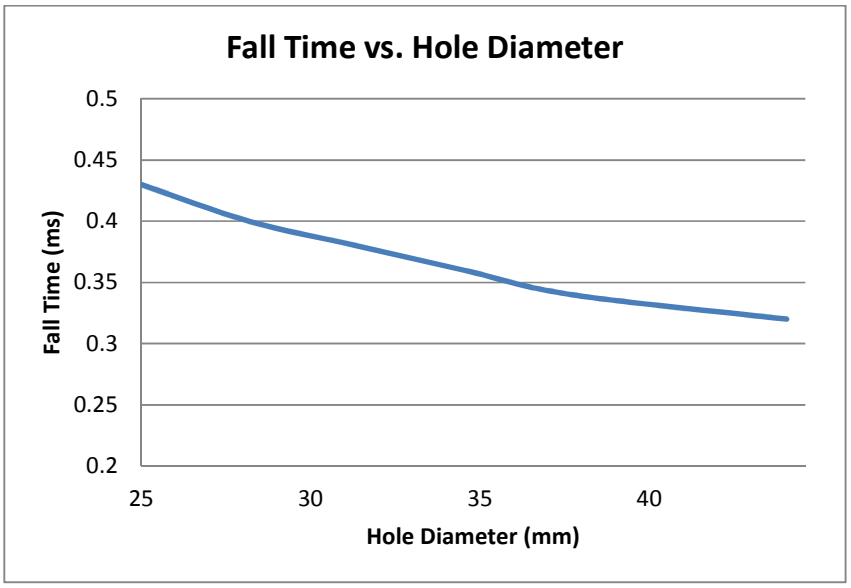


Figure 16, Fall Time measured by Saffy at 230mm (9 inches) from the blasthole for a fully coupled charge in increasing hole diameters (Saffy, 1961)

It is clear from the above discussion that the use of stemming materials will improve the efficiency and blast results obtained by mining operations, particularly where short rounds allow for the rapid ejection of high pressure gases from the blasthole. As all explosives break rock through the same simplistic breakage mechanisms, it has been suggested that irrespective of the type of explosive used, tamping should be applied to increase the length of time that gases are confined within the blasthole (ISEE, 1998). Tests undertaken by a 'large mining house' found that while tamping had a 'significant' impact on the advance achieved with high VOD cartridged emulsions, it had no definable impact on the advance achieved

with dynamite (Brinkmann, 1994). While it is likely that the efficiency achieved through the use of stemming for dynamite would have improved, the impact of stemming on the results obtained with high VOD emulsions was far more prominent.

It has been suggested that as higher VOD explosives are able to produce a greater strain pulse or shock wave than lower VOD explosives, it is not necessary to stem them as they will break ground through shock rather than gas energy (Saffy, 1961). Through tests, Brinkmann (1994) was able to demonstrate that while the magnitude of the strain wave is significant to the extent of fracturing within the rock mass, gas penetration is required for the breakout of the burden from the solid. As high VOD explosives generate a significantly greater borehole pressure than low VOD explosives, the gases formed by the reaction of explosive products are ejected from the blasthole more rapidly. This reduces the period for high pressure gases to penetrate the burden rock and in so doing facilitate the breakout of the burden. Low VOD explosives while fracturing the rock less and providing coarser fragmentation than high VOD explosives will allow for a slower build in pressure and a longer period within the blasthole thereby increasing gas penetration into fractures in the burden rock. While the use of stemming with low VOD explosives may be debateable, there is little doubt that in order to achieve maximum performance from higher VOD explosives, efficient stemming material is necessary to increase the period of high pressure gases within the blasthole.

2.4 The Impact of Poor Explosive Selection on Mine Profitability

Section 2.1 discusses a range of commercial explosives both historically and presently available within the South African narrow reef mining industry. As each of these explosives differ in physical properties and performance characteristics, the effect of each on the rock mass will differ. As underground operations and rock types are not all the same, the optimal explosive for each operation will differ according to the desired set of outcomes for each operation. Should an explosive and round design be chosen without consideration for the broader implications of the selection, the downstream financial implications of the decision could easily exceed the cost of explosives for the operation. As these downstream implications affect almost all activities within the mine, rock breaking is arguably the single most crucial and influential area of the mining operation and as such will have the greatest impact on the generation of profits (de Graaf, 2010).

According to Brinkmann (1994), the three most important considerations in daily blasting operations include the advance achieved per blast, the fragmentation of ore and the degree of overbreak experienced in stoping operations. These three considerations are influenced by geological and environmental conditions, the energy within the blast, the quality of drilling and blasting practices, and the performance characteristics and VOD of the explosive. While these factors may have the greatest influence on profits within a mining operation, they are often overlooked by production personnel due to lack of awareness, production pressure and the demand for direct savings on explosives for short term financial targets (Prout, 2010). As stressed by Beattie (1999), "the daily accomplishment of a blast of full advance, without mining excess waste and without damaging the hanging wall", is vital to the success of any mining operation.

2.4.1 The Importance of Advance per Blast and Blasting Rate

The advance achieved per blast is vital to the success of a mining operation as it is directly responsible for the liberation of payable ore from the solid (Cunningham & Wilson, 1991). Advance per blast is affected by multiple factors including the properties of the rock mass, geological considerations, blast design, drilling accuracy, explosives selection, the initiation system and timing of the round (Prout, 2010). Work undertaken by Brinkmann (1990) examined the effect of VOD on shock and gas breakage mechanisms within un-stemmed,

lined and un-lined blastholes. The results achieved by Brinkmann (1990) indicated that while shock energy was largely responsible for the fragmentation of the rock mass, gas energy was of greater importance in breaking larger burdens from the solid. As discussed in section 2.3.4, the use of stemming plugs with high VOD explosives is essential in maximising the period in which high pressure gases remain within the rock mass to act on the burden. This in turn increases the breakout of the toe of the blasthole and the overall efficiency of the blast.

Through calculation, Cunningham and Wilson (1991) proposed that for underground narrow reef operations, only a small increase in advance is required to justify the use of a more expensive explosive in order to improve the general results of the blast. Given current commodity prices, this increase in advance is often no more than millimetres in length. Table 1 compares the cost of explosives to the revenue generated through a single blast on a 30 metre production panel within a gold mining operation. It is evident from the calculation that with an average grade of 6 grams per tonne, the cost of explosives will be recovered within the first centimetre of advance. From this calculation it can be seen that the best suited explosive should be selected for a blasting application as the direct cost of explosives is negligible when compared to the financial implications of greater advance rates and improved blast efficiency. While it may increase the direct cost of explosives, the increase in expenditure will be offset by an increase of only millimetres in advance.

Table 1, Comparison of Revenue and the Direct Cost of Explosives for a Panel (Gold)

Panel Length (m)	30	Burden (m)	0.5
Panel Height (m)	1.2	Lines	2
Advance (m)	1.0 (83%)	Blastholes per face	120
Rock density (kg/m^3)	2.7	Mass per hole (kg)	0.8
Tonnes per blast (t)	97	Mass per face (kg)	96.0
Average Grade (g/t)	6.0	Powder factor (kg/m^3)	2.1
Gold Produced/blast (kg)	0.6	Cost per Kg Explosive	R 10.00
Gold Price ($/Oz)	$ 1,260	Cost per fuse	R 10.00
Exchange Rate (R/$)	R 10.20	Cost of Explosives	R 960
Rand Gold Price	R 413,248	Cost of Accessories	R 1,200
Revenue per blast	R 241,006	Total Cost of Explosives	R 2,160
Explosives Cost as % Revenue	**0.9%**	**Cost per Tonne Broken**	**R 22.22**

As the advance achieved in blasting operations is a function of the performance and reliability of an explosive, factors such as the physical properties of explosives also need to the taken into consideration during the selection process. The use of water soluble explosives such as ANFO in high humidity or wet operations presents the risk of blast failure and the loss of advance should the sensitivity of the explosive be reduced through the hydroscopic properties of the explosive (Mulke, 1966). Should only one in one hundred blasts fail as a result of poor explosive selection resulting in a loss in advance of only one metre, this would represent an equivalent loss in advance of one centimetre per panel per blast. Through the calculation above it can be seen that the loss of this single centimetre of advance per blast represents a financial loss greater than the cost of explosives for every blast and thus would have justified the use of an explosive at double the expense in order to prevent the failure of a single blast. From this it is evident that the direct cost of explosives is negligible when compared to the financial implications of daily blasting operations. Any loss in advance experienced within a mining operation represents a piece of ground that needs to be drilled and blasted for a second time in order for it to be broken from the rock mass (Brinkmann, 1994).

2.4.2 Fragmentation and Mine Call Factor

As with the advance achieved per blast, discussed above, multiple factors are responsible for the degree of fragmentation achieved in blast results. Of these the most important factors in the determining the fragmentation within a specific round design include the specific energy of the explosive, the powder factor within the round, the timing of the round, the quality of drilling and blasting practices and the VOD of the selected explosive (Prout, 2010; Lindsay, 1991). While the mass and VOD of the explosive within the blasthole determine the extent of fracturing surrounding the blasthole, Brinkmann (1994) found by experimentation that the specific energy within the round had by a significant margin the greatest effect on the size of fragmentation achieved in the blast. This is illustrated in Figure 17 where the specific energy per cubic metre is plotted against fragment size for 10%, 50% and 90% screen pass rates (Brinkmann 1994). Due to the increase in pressure within the blast, rock particles will be accelerated to higher velocities and the size of rock fragments will be further reduced on collision with the excavation walls (Brinkmann, 1994).

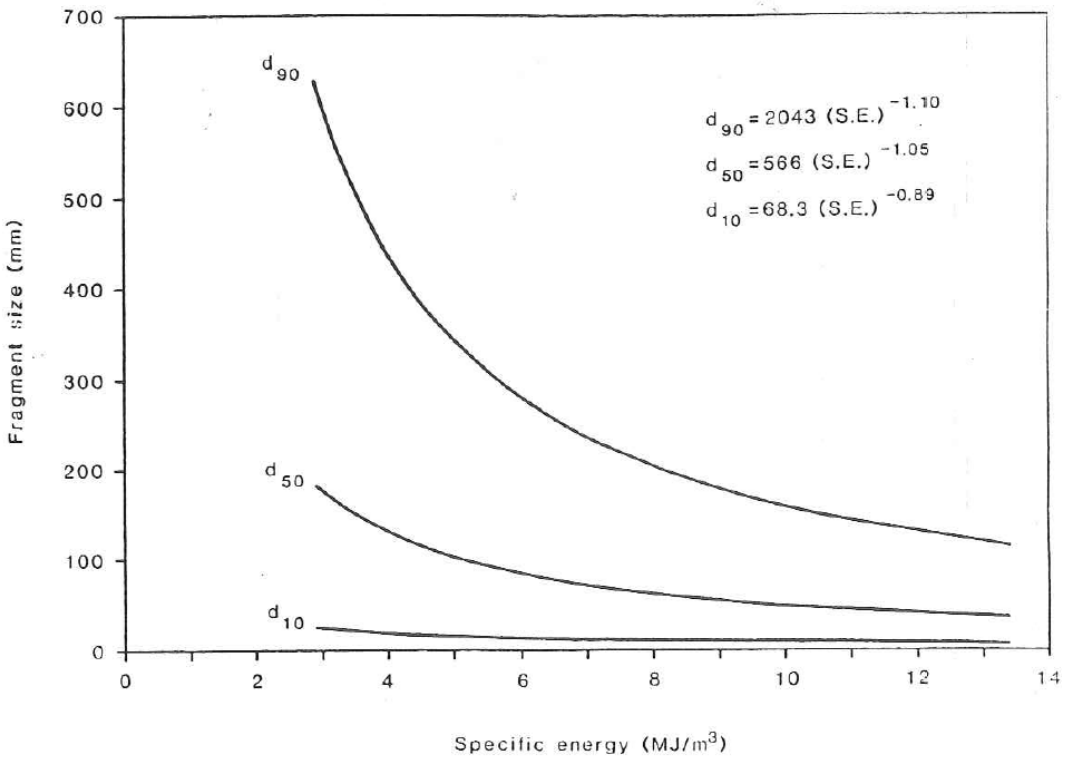
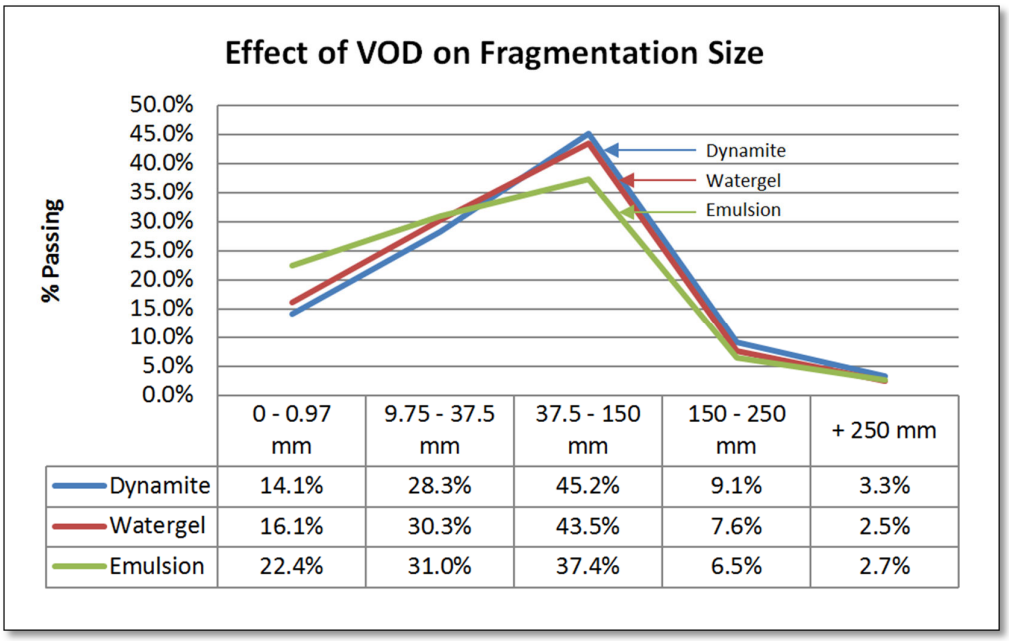


Figure 17, Influence of specific energy on fragment size in blast results (Brinkmann, 1994)

From his experiments Brinkmann (1994) noted a practical limit to the specific energy that could be applied in an attempt to reduce the maximum fragment size achieved in the blast. As indicated in Figure 17, only a small reduction in large particle size is experienced for increasing specific energy beyond approximately 8 MJ/m^3. While the VOD of an explosive is of lesser importance than specific energy in the size of fragmentation produced in the blast, Brinkmann found that in blasts with a specific energy of 10 MJ/m^3, higher VOD emulsion cartridges resulted in an average fragment size 30% smaller than that of low VOD dynamite explosives. Similarly, moderate VOD watergel explosives also experienced a reduction in average fragment size, however this only by 12%. These results are illustrated in Figure 18.



	0 - 0.97 mm	9.75 - 37.5 mm	37.5 - 150 mm	150 - 250 mm	+ 250 mm
Dynamite	14.1%	28.3%	45.2%	9.1%	3.3%
Watergel	16.1%	30.3%	43.5%	7.6%	2.5%
Emulsion	22.4%	31.0%	37.4%	6.5%	2.7%

Figure 18, Influence of VOD on fragment size in blast results (Brinkmann, 1994)

In determining the required fragment size for a specific mining operation, logistical considerations, geological properties of the rock mass and the distribution or mineralisation of the ore need to be taken into consideration (Brinkmann, 1994). Fragment size is of particular importance to carbonaceous gold reefs within the Witwatersrand due to the detrimental effect of excessive fragmentation on mine call factor. Several case studies have highlighted the severe financial implications that can result from the excessive use of high energy explosives on such operations (Kruger, 2010; Brinkmann, 1994). Another repercussion as a result of excessively fine fragmentation includes increased operating expenses for autogenous mills due to the requirement for additional steel balls (Brinkmann, 1994).

Excessively large fragmentation will similarly have an impact on the operating costs of the mining operation. As the powder factor within the blast increases, within acceptable limits, the average fragment size produced within the blast will decrease increasing the efficiency of handling and size reduction activities downstream.

Areas of low efficiency and increased expenditure as a result of oversize ore in the muckpile include (Prout, 2010; Cunningham & Wilson, 1991; Brinkmann, 1994):

- Damage to support and blasting barricades
- Secondary blasting activities
- Cleaning cycle times
- Efficiency and ware on scrapers
- Grizzly maintenance costs
- Ore pass blockages
- Maintenance of loading boxes
- Equipment running costs
- Crusher throughput and maintenance

From the above discussion it is clear that a fundamental understanding is required of the effect of fragmentation size on the profitability of a specific mining operation in order for the correct explosive and round design to be selected for the application. Given the financial implications of fragmentation on the profitability of an operation, it is essential to understand both the effect of excessive fines and oversize material on the profitability of the mine (Brinkmann, 1994).

2.4.3 Overbreak and Dilution

One of the greatest areas of leverage in narrow stope mining operations is the degree of overbreak experienced within stoping operations. Overbreak results from the penetration of high pressure gases into the rock mass surrounding the excavation, resulting in the breakout of excess rock from the hangingwall (Brinkmann, 1994). As with advance and fragmentation, the primary factors responsible for the degree of overbreak achieved include the energy of the explosive and within the round design, the quality of drilling and blasting practices, sequential firing of blastholes and the VOD of the selected explosive. As the powder factor within a round design increases, the magnitude of the shock wave and the volume of high pressure gases produced within the blasthole increase. On detonation, high pressure gases penetrate the hangingwall resulting in the breakout of the hanging. As rock type and geology play a significant role in the extent of overbreak, these factors need to be taken into consideration during the selection of explosives and blast designs.

A study undertaken by Cunningham and Wilson (1991) on the comparative overbreak experienced through the use of Dynagel and ANFO explosives on a gold mining operation revealed an average overbreak of approximately 18% for Dynagel in comparison to an average overbreak of approximately 33% for that of ANFO. Despite the similar VODs of Dynagel and ANFO explosives, a greater level of overbreak was experienced through the use of ANFO as a result of the overcharging of blastholes (Cunningham & Wilson, 1991). This comparison illustrates the importance in controlling the energy within the blast in order to prevent overbreak.

As higher VOD explosives produce a greater strain wave and have a shorter period within the blasthole, limited time is available for the penetration of high pressure gases into rock strata surrounding the excavation. Higher VOD explosives can thus be applied together with lower energy explosives in order to control the extent of damage to the hangingwall in stoping operations (Lindsay, 1991).

Given their narrow stoping width, narrow reef operations are particularly sensitive to overbreak due to the additional volume of rock that will be broken out as a result of only minor levels of overbreak (Brinkmann, 1994). As overbreak is largely responsible for the liberation of waste, payable ore is diluted decreasing the head grade and increasing all expenses associated with the downstream handling and processing of ore (Prout, 2010; Swart, Human & Harvey, 2004; Brinkmann, 1994). Through overbreak, mine infrastructure is indirectly allocated to the handling and processing of waste rock and the production of gold is therefore restricted. This increases expenses incurred through activities such as tramming, shaft costs and time limitations, mucking and support requirements in proportion to the level of overbreak and a limit to potential revenue through reduced gold production (Prout, 2010; Cunningham & Wilson, 1991).

Due to the inefficiencies in the processing of ore, an increase in the volume of rock processed for the liberation of a specific quantity of gold will also result in an increase in the total gold losses incurred in processing (Prout, 2010; Pickering, 2005). This is as a result of gold being trapped within the larger volume of rock and concentrate during the beneficiation process. As expenses related to the handling and processing of ore increase and the actual mass of gold extracted from the rock decreases, over break has a significant effect on the bottom line of mining operations. ‘If mining width is not controlled the profitability of the mining operation will suffer’ (Pickering, 2005).

2.5 The Impact of Poor Labour Practices on Mine Profitability

Given the importance of the leveraging effects of explosives as discussed above, the importance of the correct application of explosives through correct drilling and blasting practices is fundamental to the success of any blasting operation. As production personnel are often unaware of the broader implications of their decisions, significant financial implications often originate in daily blasting activities. The state of health of a mining operation is determined by the effectiveness of daily drilling and blasting activities on the operation (Cunningham & Wilson, 1991).

2.5.1 Poor Drilling Practices

According to Brinkmann (1994), the most common causes for poor blast performance in daily operations includes poor drilling practices together with the non-sequential detonation of blastholes due to the initiation system. The importance of these two factors is that together, they determine the burden that exists for a specific blasthole at the time of detonation (Brinkmann, 1994). During blasting, the most difficult area of a round design to break is the toe of the blasthole due to the geometrical constraints confining the toe of the hole. As the effect of drilling angle and elevation have a greater effect on the position of the base of the blasthole than the collar, any variation in drilling accuracy reduces the potential for the breakout of the toe of the hole. As a change of only 10˚ in drilling direction increases the burden at the toe of the hole by more than 30%, the advance achieved in the blast will be significantly affected (Brinkmann, 1994).

In order to compensate for the loss of advance due to poor drilling accuracy, miners decrease the size of burdens between blastholes, increasing the number of blastholes within the face (Brinkmann, 1994). Through this practice, blasting efficiencies are reduced to well below optimal levels. As this practice increases the powder factor within the blast, the average size of fragmentation produced in the blast will be reduced. However, due to the uneven distribution of energy within the rock mass, oversize and undersize fragmentation will still be produced. Overbreak and blast damage within the hanging wall also increase due to the penetration of high pressure gases into undesired areas as a result of overburdened and misaligned blastholes (Jager, 2010; Kruger, 2010). Through accurate drilling practices, Brinkmann (1994) found that the burden between blastholes could be increased by up to

34%, while maintaining full advance. This increase in burden size is similar to the 30% deviation in the size of toe burdens resulting through a 10˚ error in drilling direction.

2.5.2 Overcharging of Blastholes

In order to maximise the advance achieved per blast, underground charging personnel often load blastholes to the collar of the hole. While an increase in the powder factor within the collar of the blasthole increases overbreak and fragmentation in the blast, it provides no assistance to the breakout of the toe of the blasthole and the resulting advance. This practice is detrimental to the profitability of the mining operation as it results in an increase in the direct cost of explosives while increasing levels of overbreak, fragmentation and fly-rock. This in turn causes damage to infrastructure and support within the operation (Prout, 2010).

Table 2 tabulates the effect that the coupling ratio has on the Relative Bulk Strength of various explosives loaded within 36mm blastholes. As cartridged emulsions experience a coupling ratio of approximately only 85% in 36mm blastholes, a 15% reduction in RBS is experienced during the loading process. Given this loss in mass and thus energy together with the loss in shock transmission through decoupling discussed in section 2.3.3, cartridged explosives have a lower propensity for overbreak than fully coupled charges unless a cartridge of significantly greater strength is applied. Figure 19 illustrates the propensity for the overcharging of blastholes with low and high density cartridged emulsions, pumpable emulsions and ANFO.

Table 2, Effect of coupling on the energy within a 36mm blasthole

Explosive	RWS	Loaded Density	RBS	Coupling Ratio *	In-hole RBS (After Coupling)
ANFO (Low Density)	100	0.95(Rorke)	119	100%	119
ANFO (Std. Density)	100	1.05(Mulke)	131	100%	131
Cartridged Emulsion (Low Energy)	94	1.15	132	***85%***	***112***
Cartridged Emulsion (High Energy)	109	1.17	153	***85%***	***134***
Pumpable Emulsion (Min)	88	0.9	99	100%	99
Pumpable Emulsion (Max)	92	1.15	132	100%	132

* Average Coupling Ratio for 36mm hole diameter

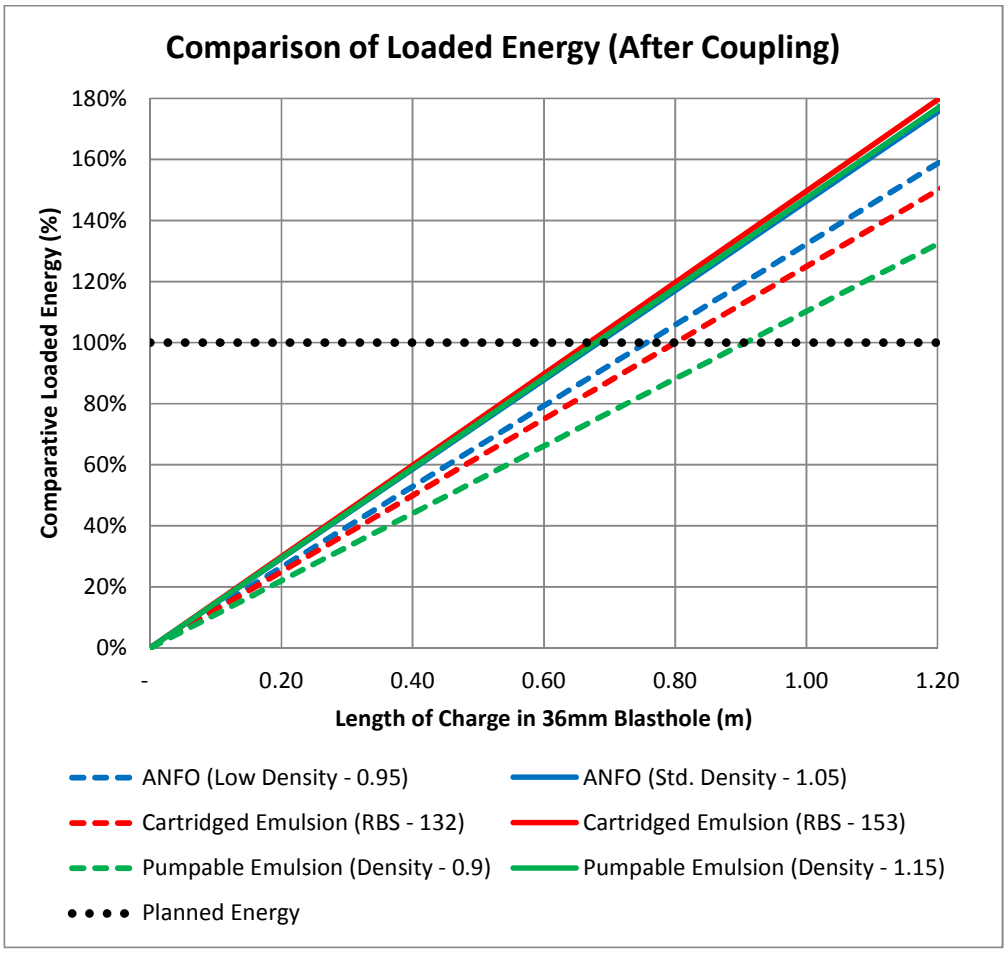


Figure 19, Propensity for the overcharging of blastholes with varying explosives. The planned energy within the 36mm blasthole represents a fully coupled 800 gram charge of ANFO at a density of 1.05, resulting in a charge length of 68cm.

The planned energy in Figure 19 represents a fully coupled 800 gram charge of ANFO at a density of 1.05. As depicted in the graph, the column length required for this explosive charge is 68cm or 57% of the blasthole. From the Figure 19 it is evident that due to the full coupling of the ANFO and pumpable emulsion systems, the energy loaded per metre of blasthole is similar to that of comparatively higher energy cartridged emulsions. From this comparison it can also be seen that the higher the RBS and coupling of the explosive, the greater is the significance of overcharging of blastholes (Van der Walt, 2004). As many mining operations operate on lower strength emulsion cartridges such as that represented by the cartridged emulsion with a RBS of 132, the energy loaded by ANFO can easily exceed that of cartridged explosives. Due to the propensity of operators to overcharge blastholes

when using bulk ANFO explosives it is common for them to exceed the recommended charge mass by in excess of 175%. As the density and mass of pumpable emulsion explosives delivered through the portable pump can be predetermined, a controlled mass of pumpable emulsion can be charged per blasthole while allowing for full coupling within the hole. One of the methods of reducing the energy within a pumpable emulsion round is by reducing the loaded density in the blasthole. As indicated in Figure 19, this reduction in density would reduce the total energy that could be loaded per hole, but in so doing will also reduce the energy available at the toe of the hole. This method of control is therefore not recommended for higher strength rock masses as it will reduce the break out of the toe.

As only 57% of the 36mm blasthole discussed with respect to Figure 19 should be loaded, the practice of filling blastholes to the collar will result in an increase of 75% in the powder factor of the blast illustrating the significance of overcharging on the energy within the blast. Figure 20 indicates the effect of overcharging (with ANFO) on the size of fragmentation achieved in the blast (Prout, 2010). From the trend it is clear that the increase in energy resulting from the overcharging of blastholes has had a significant effect on the size of fragmentation achieved.

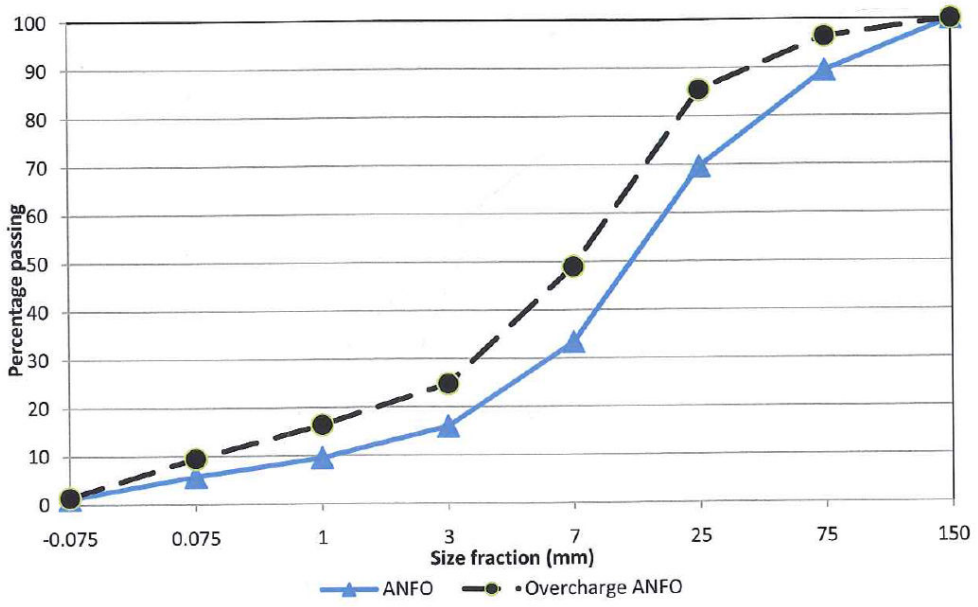


Figure 20, Effect of overcharging with ANFO on the size of fragmentation (Prout, 2010)

2.6 Logistical Requirements for Explosives Management

It is evident from above discussions that explosive selection together with the selection of appropriate blast design criteria will have a direct influence on the profitability of a mining operation through the blast results obtained. Though the financial leverage of the above mentioned factors is of greatest importance to the selection of explosive, additional factors such as logistical considerations and the control of explosives also need to be taken into consideration for their influence on the profitability of the operation.

Distribution models for commercial explosives fall within two broad categories, packaged or bulk (unpackaged) explosives. Packaged explosives originated with the development of dynamite, where individual sticks of explosive could be loaded into a blasthole prior to blasting. In the present day, cartridged explosives are available in both watergel and emulsion varieties.

While nitro-glycerine can be regarded as the first liquid or bulk explosive that could be poured into blastholes prior to blasting (ISEE, 1998), the greatest innovation in bulk explosive technology took place through the development of prilled ANFO in the 1950's. The development of pumpable watergels and emulsions followed as a matter of course and too grew in popularity within the mining industry. Though initially limited to use within surface operations and later mechanised massive underground operations, the confined nature of narrow reef stoping operations prevented the use of bulk or unpackaged explosives. In order to achieve a number of the advantages available through the use of bulk explosives, ANFO was introduced to the underground industry in pre-packed 25kg bags that allowed for ANFO to be blow loaded into blastholes through the use of a pneumatic loader once at the blastface (ISEE, 1998).

2.6.1 Packaged Explosives

The way in which an explosive is packaged has both practical and financial implications on mining operations. Two types of packaged ammonium nitrate based cartridges exist for use in underground blasting applications, namely watergel (slurry) and emulsion cartridges. As packaged explosives such as cartridges are pre-sensitised at a manufacturing facility, better equipment and a higher level of control can be exercised in achieving consistent quality

product. In addition, as cartridges are manufactured ready to load into the blasthole, they are also easy to handle within confined and difficult underground conditions.

Despite these benefits, significant disadvantages are brought about through the use of packaged explosives. Given the pre-sensitised nature of packaged explosives, including watergel and emulsion cartridges, most packaged explosives are classified as class 1 explosives and as such are subject to the regulations of the Department of Mineral Resources (DMR), Chief Inspectorate of Explosives (CIE) and the Department of Labour (DOL) throughout manufacture, delivery to and storage on mining operations. This is a significant disadvantage within the current South African environment given the high level of legislation regulating the transportation and control of class 1 explosives.

In addition to increased regulation, the manufacture of pre-packaged explosives necessitates increased investment and operating expenses on the side of the manufacturer in turn resulting in higher prices than for packaged explosives than for bulk products. As packaged products in turn require additional labour for offloading and handling on the shaft, transportation throughout the operations and during loading of the blastface, considerable additional labour expenses are incurred. As a reduction in coupling ratio and thus explosive efficiency is also experienced through the use of cartridged explosives, as discussed in section 2.3.3, an increased mass of explosive is required per blasthole further compounding the additional time and labour is required to load the blastface.

2.6.2 Bulk Explosives

As bulk and semi-bulk explosives systems deliver explosives to the blastface in a pumpable or pourable form, loaders or charging equipment are necessary for the loading of blastholes. Advantages of bulk explosives systems include a reduction in labour for the transportation and loading of explosives, a reduction in charging time due to the high charging rate of charging equipment and a 100% coupling ratio between the explosive charge and the blasthole wall. As full coupling increases the efficiency with which the detonation pressure and brisance produced in the detonation front is transmitted into the rock mass, the overall efficiency of the blast is improved (Saffy, 1961). As bulk explosives exhibit a coupling ratio of 100%, they are likely to achieve better blast results than equivalent strength cartridged explosives when used in comparable blasting environments. As cartridged explosives possess

a coupling ratio of only 80% to 90%, they are unable to achieve the same effective charge diameter at the base of the hole when compared to bulk explosives. Bulk explosives thus concentrate a greater proportion of the available explosives energy at the bottom of the hole facilitating better breakout of the toe and increasing the possibility of advance.

In addition to the opportunity for the use of bulk class 5.1 blasting agents (to be discussed in Section 2.7), an additional advantage limited to the use of bulk emulsions and other non-sensitised liquid explosives systems is the ability to adjust the density and thus energy available within the blasthole. This can be achieved through the use of a single base blasting agent coupled with a range of specified density sensitising agents.

Though significant advantages are available through the use of bulk explosives systems, the loading of bulk explosives at the blastface presents a degree of risk within daily blasting operations. The introduction of loaders or charging equipment necessary for the loading of the blastface increases the opportunity for equipment failure or poor labour practices thereby resulting in insensitive or incorrect density explosive that could in turn result in undesirable blast results or the complete failure of the blast. In order to reduce the possibility for error the training of personnel is of greater importance with bulk explosives systems than with cartridged explosives given the increased opportunity for failure of the blast. As bulk explosives require storage, transfer and loading equipment, capital requirements for bulk explosives systems can be greater than that of pre-packaged systems (should shaft head deliveries be available and magazine facilities not be utilised) (Prout, 2010).

2.7 Legal Classification of Blasting Systems

Class 5.1 blasting systems were first introduced with the onsite mixing and sensitisation of ANFO and watergels by Cook in 1958 (AEL, 2012). While available to open cast and large mechanised operations for decades, they have only recently become available to deep level tabular operations. As Class 5.1 blasting or oxidising agents are not classified as explosives, they are free from many of the explosives regulations imposed on Class 1 explosives. In South African legislation, the CIE accepts ‘Ammonium nitrate emulsions, suspension or gels’ (ANE) as Classification 5.1 or ‘non-detonable’ if they are able to pass the three United Nations (UN) tests for a ‘non-detonable’ blasting agent or intermediate. These three tests include (United Nations, 2009):

1. Thermal Stability Test for ANE (Ammonium nitrate emulsions, suspension or gels)

   A 500ml Dewar vessel is filled to 80% capacity and heated to 20˚c above the maximum possible transport temperature for 7 days. No sample may exceed the test chamber temperature by 6˚c during the test.

2. ANE Gap Test

   A 70mm thick PMMA (Perspex) spacer is positioned linearly between a pentolite booster (95mm in diameter by 95mm in length) and a confined sample (95mm in diameter by 280mm in length) in order to test the sensitivity of samples to a shock pressure between of 3.5 and 4GPa. A punctured steel witness plate (20mm thick) at the end of the intermediate column indicates detonation of the sample.

3. Koenen Test

   The Koenen test is used to establish the minimum confinement required for a tube of ANE to detonate when heated rapidly at a heating rate of 3.3 ± 0.3 K/s. In order to be classified as a class 5.1 oxidiser a sample must fail to detonate when burnt within the specified test vessel with a vent hole of 2.0mm or greater in size.

One of the greatest obstacles to the delay in the introduction of Class 5.1 blasting agents to underground tabular operations is the size and complexity required for the reliable 'manufacture' or sensitisation of blasting intermediates at the blastface. While this is comparatively easier to achieve and control in mechanised operations with fewer charging units and semi-skilled and trained operators, the confined nature of tabular stoping operations has presented challenges to the size, reliability and number of charging units required as well as challenges linked to the level of training of the workforce.

2.7.1 Transportation and Storage of Explosives

As Class 5.1 blasting intermediates are not detonable through shock, friction or fire, they represent the forefront of safe commercially available explosives technology within the mining industry. Improved levels of safety and reduced legislation available through the implementation of Class 5.1 blasting intermediates allow for increases in operational efficiency with relatively little change to daily mining practices. As the significant majority of explosives used in mining operations are either pre-sensitised ANFO or cartridged explosives, their substitution with a Class 5.1 blasting agent significantly reduces the level of explosives control required on the operation as only initiation systems and accessories will

require stringent control (Fourie, Zaniewski & Cross, 2010; Prout, 2010). This decrease in regulation will significantly reduce the need for storage facilities for pre-sensitised explosives thus allowing a considerable reduction in storage expenses from the possible use of magazine facilities to a reduction in the number of explosive boxes and control systems required for the operational control of explosives. These costs are often ignored when comparing the overall cost of explosives systems.

While these advantages are significant to opencast and underground massive mechanised operations, the potential benefits they offer to deep level narrow reef operations are arguably greater due to the poor flexibility and accessibility on vertical shafts and limited control over daily labour practices. Logistical considerations such as available shaft time, tramming restrictions and the storage of explosives have a greater influence on operational efficiencies of deep level operations than on mechanised or surface operations.

Due to the Class 5.1 classification of blasting agents, they are no longer classified as explosives and can be transported together with men and materials depending on standard mine practices. Due to the reduction in applicable legislation, blasting agents may also be transported underground and stored in larger quantities than class 1 explosives, thereby allowing the transportation of class 5.1 blasting agents underground on a weekly or biweekly basis. This increases the availability of locomotives for the transportation of materials to the workface as well as ore to the shaft (Fourie, Zaniewski & Cross, 2010), and reduces the demand on shaft time allowing for the transportation of additional materials underground (Prout, 2010). This in turn allows for a possible increase in production and will reduce the risk of lost blasts due to insufficient materials at the face (Prout, 2010).

Due to legislation governing the storage and handling of Class 1 explosives underground, the over ordering of explosives can result in explosives being concealed in worked out areas in the mine. The repercussions of this practice include the possibility of punitive legal measures by the DMR, the risk of the theft of the explosives for use in illegal mining activities and ATM bombings and the potentially fatal risk to proto teams in the event of underground fires. As blasting agents do not fall within the same legal classification, no risk or legal ramifications exist for blasting agents if conceal in a worked out areas. Also, as blasting agents burn rather than detonate when they exceed their minimum reaction temperature in a fire, the risk to proto teams is significantly less than that of Class 1 explosives.

During the sensitisation of pumpable emulsion explosives, a small quantity of sensitising agent is mixed with the base emulsion through the charging unit. As the percentage of sensitising agent required for the sensitisation of a blasting agent is only approximately 1% of that of the base emulsion, only 1 to 2 litres of sensitiser is necessary for daily blasting practices on a panel. As the mass of sensitising agent required is so small it can be carried underground on a daily basis by the responsible person before being returned at the end of the day to be secured at a central facility at the end of the day. For this reason the sensitising agent can be considered the 'key' to the use of the system, greatly reducing the level of control required for the entire mass of class 1 explosives used on a daily basis.

2.7.2 Theft of Explosives

One of the greatest factors driving the implementation of Class 5.1 blasting agents in mining operations is the theft of commercial explosives for use in criminal activities. As cartridged explosives are pre-sensitised and easy to handle, they have become the preferred explosive for use in illegal activities such as ATM bombings and illegal mining activities. In an attempt to reduce the extent of crimes committed with commercial explosives, the 'Inspector of mines', under the auspices of the Department of Mineral Resources (DMR), has targeted the control of explosives throughout commercial blasting applications in an attempt to prevent the flow of Class 1.1 cartridged explosives from mines into the community. Despite the continued focus for the police on the theft of explosives and the increase in arrests seen in recent years, present levels of ATM bombings are still high presenting an unacceptable level of risk to the community. This trend is evident in Table 3 below.

Table 3, Number of ATM bombings per year in South Africa

Fiscal Year	Number of ATM Bombings
2007/2008	431
2008/2009	387
2009/2010	247
2010/2011	399
2011/2012	251

(South Africa.info, 2012 and ISS, 2011)

As Class 5.1 blasting agents are not explosives until sensitised in the face they cannot be used in criminal activities without the correct components and equipment required for their sensitisation. As the use of this equipment in such applications is highly implausible, criminals will continue to use pre-sensitised cartridged explosives until they are no longer available. The viscous nature of pumpable emulsions both before and after the sensitisation process also prevents the relocation of the sensitised emulsion without disturbing the chemically induced gas bubbles within the sensitised emulsion. As the disturbance of these gas bubbles would return the emulsion to its non-explosive state it would again become un-detonable and not possible for use in criminal activities.

2.7.3 Section 54 Notices

Given the high occurrence of ATM bombings within South Africa on an annual basis, the DMR has acted decisively in issuing punitive action against mining operations that do not comply with legislation regarding explosives control. Table 4 and Figure 21 below indicate the total reported number of section 54s issued to a large platinum producer within South Africa between 2007 and 2013 together with the number of Explosives related Section 54s. Despite significant efforts and world leading innovations by the platinum producer to reduce the occurrence of section 54s linked to explosives control, a total of nine section 54s were still experienced due to explosives in 2012.

Table 4, Explosives Related Section 54s (Large platinum producer)

Year	Total Section 54s	Explosives Related Section 54s	Explosives Related Percentage %
2007	27	-	-
2008	35	-	-
2009	47	23	49%
2010	36	25	69%
2011	81	17	21%
2012	52	9	17%
2013	70	-	-

- No Data Available

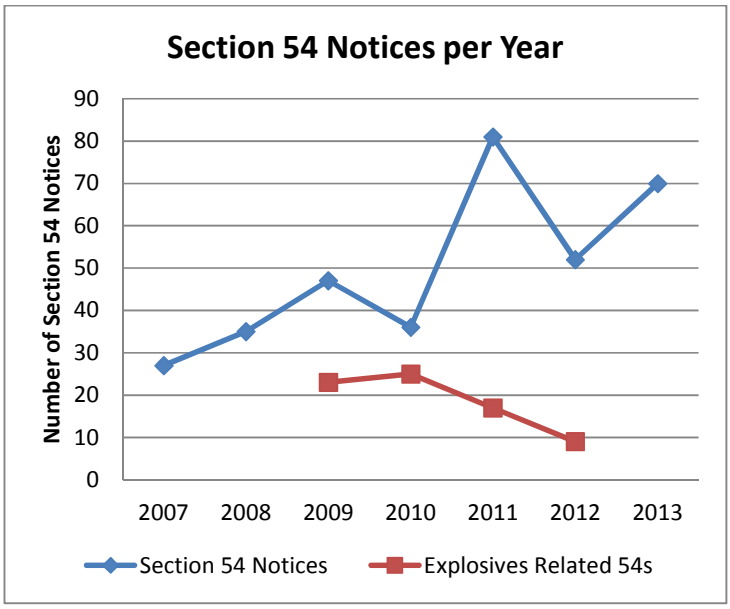


Figure 21, Reported number of section 54s per year against explosives related section 54s (Large platinum producer)

Table 5 lists the loss of production experienced by the producer in 2011 due to explosives related section 54s. Despite the emphasis on the reduction of section 54s and the declining number experienced each year, the loss in revenue experienced through explosives related section 54s is still in line with the total annual cost of explosives for the producer.

Table 5, Loss of Production due to explosives related section 54s

Financial Year	2011
Section 54 Notices (all operations)	81
Mine stoppage days in year (all operations)	312
Days lost / Notice	3.9
Lost oz.	138 000
Annual Production	2 410 000
Lost oz. per day standing	442.31
Loss of Production due to section 54s	5.7%
Loss of Production due to Explosives section 54s	1.2%
Lost Revenue due to Explosives section 54s	≈ R 400 mill

Figures reported for the Lonmin Middelkraal operation (Hossy & Saffy shafts) over the same period revealed higher production losses due to section 54 notices. Reported losses incurred over the first 6 months of the 2012 financial year amounted to 121'000 tonnes or 11% of planned production for the period (World Television, 2012), while reported losses at Impala Platinum amounted to 33'000 ounces within the final quarter of 2011 alone, more than 10% of the planned production for the period (Implats, 2012; JM, 2012). Gold operations were similarly affected by the increase in section 54 notices, with Anglogold Ashanti reporting losses of 73'000 ounces of gold over the first 6 months of the 2012 financial year. This loss in production totalled 4.5% of the planned 1'624'000 ounces for the period (Mineweb, 2012; Anglogold Ashanti Annual Report, 2011).

3 Methodology

Primary aim: To evaluate the viability of pumpable emulsion explosives for use in narrow reef mining operations.

Objectives:

1. To develop a suite of equipment and emulsion formulation suitable for the implementation of pumpable emulsions in conventional narrow reef mining operations.
2. To characterise the performance of the fore mentioned pumpable emulsion under controlled blasting conditions.
3. To evaluate the performance of the fore mentioned pumpable emulsion against existing commercial explosives in underground field trials.

Given the benefits achieved through the implementation of pumpable emulsions in opencast and mechanised underground mining operations, together with the success of cartridged emulsions within underground narrow reef operations, it was believed that if a suitable method could be developed for their implementation in underground narrow reef operations, a number of their advantages could prove beneficial to the sector. In order to test the performance and viability of pumpable emulsions within conventional underground narrow reef operations, a project was proposed that allowed for the development of the necessary emulsion formulations and charging equipment, prior to field trials with the new technology.

Research design

This project comprises both quantitative and qualitative aspects yielding both numerical data as well as theoretical frameworks. Data collection was conducted both in a controlled environment as well as within an underground mining context.

Data collection process

This project comprised three consecutive phases as outlined in the research objectives above. The first phase of the project included the development of a new emulsion formulation, suitable for use in small diameter blastholes, together with the development of the necessary emulsion charging equipment required for the implementation of pumpable emulsions within the narrow reef environment. The second phase of the project involved controlled

characterisation tests at BME's Losberg manufacturing facility on the new emulsion formulation in conjunction with the new suite of emulsion charging equipment. The purpose of this phase of the study was to gain an understanding of the performance and reliability of the new suite of charging equipment, as well as to characterise the performance of the new emulsion formulation, such that it could be compared to explosives currently in use within the underground narrow reef industry. The third and final phase of the project included the implementation of the new pumpable emulsion technology within the underground environment such that the performance of the new technology could be compared to existing explosives systems. Through this approach, the effects of pumpable emulsions could be studied for their suitability and performance in narrow reef mining operations. While the three phases of the project are described simplistically, the requirements of new product development necessitated a certain degree of repetition through phases one and two of the project. This was undertaken in order to achieve the desired performance in both the new emulsion formulation and the new suite of pumpable emulsion technology.

4 Phase 1 - Development of New Emulsion Technology

In order for pumpable emulsions to be considered suitable for use in conventional tabular operations, the areas of emulsion formulation and charging unit development were considered paramount to the success of the project. However, owing to intellectual property considerations, a detailed discussion of the design process and the final specifications of the new Portable Emulsion Pump have been omitted from this report. A brief discussion has however been included on the product requirements and performance for both the new emulsion formulation and pumpable emulsion technology. As it was proposed that technology was the foremost limitation in the implementation of pumpable emulsion systems, a comparison is also drawn between existing emulsion charging equipment and the new pumpable emulsion technology.

4.1 Development of a Pumpable Emulsion for Small Diameter Blastholes

As discussed in the literature review, the physical properties and performance characteristics of an explosive are of fundamental importance in the selection of an explosive for a specific blasting application. With the physical properties of pumpable emulsions to a large extent predetermined, the areas of greatest flexibility in the performance of new emulsion formulations exist in the specific energy of the formulation and sensitivity of the emulsion to initiation. These factors will in turn influence the performance characteristics of the emulsion within specific blasting applications.

Table 6, Requirements of the New Narrow Reef Emulsion Formulation

Sensitivity of intermediate	Class 5.1 oxidiser
Sensitivity of sensitised emulsion	8D cap sensitive
Period to achieve firing density	< 30 minutes
Critical diameter	< 25mm
Proposed applications	Development & Stoping
Viscosity	38000cp
Desired shelf life	> 6 months
Pumping cycles	Unrestricted (closed emulsion system)

In order for pumpable emulsions to be successfully implemented within narrow reef mining operations, a pumpable emulsion formulation was required that could detonate reliably in the small diameter blastholes typically present within narrow reef industry. As the pumpable emulsion with the smallest available critical diameter was limited to use in blastholes greater than 38mm in size, a new formulation was required that could be implemented in the 28mm to 38mm blastholes typically found within the industry. In order to achieve reliable propagation of the detonation front in small diameter blastholes, the sensitivity to initiation of the sensitised emulsion would need to be increased while maintaining the class 5.1 legal classification of the un-sensitised emulsion.

In addition to the requirement for the emulsion to be used in smaller hole diameters, a second reason for the required sensitivity was the need to eliminate the use of boosters traditionally used in the priming practices of pumpable emulsion systems. As round designs within conventional narrow reef operations contain a considerable number of small diameter blastholes, the ratio of detonators to the total mass of explosives is far greater for narrow reef operations than mechanised operations and thus requires the use of a greater quantity of detonators and boosters per blast. This trend is evident in Figure 22 as calculated in Table 7.

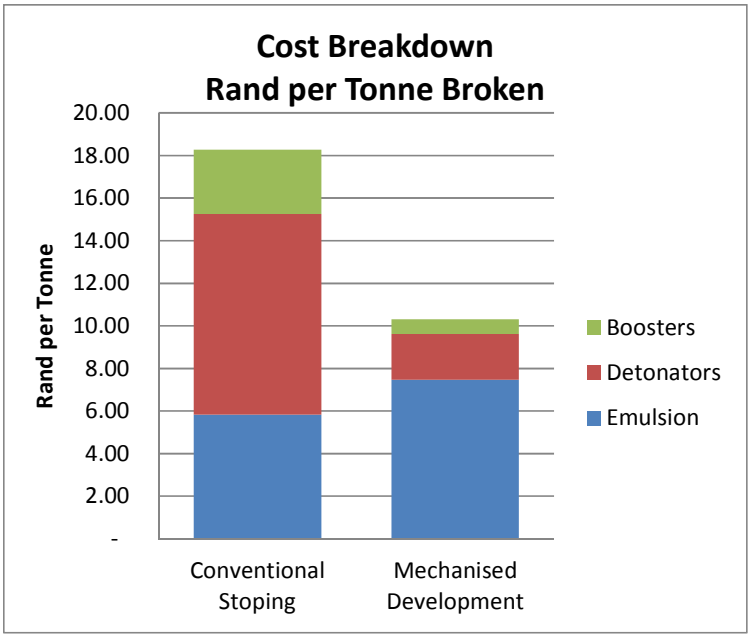


Figure 22, Itemization of blasting expenses – Rand per tonne broken

Table 7, Calculation of Blasting Expenses –Stoping and Mechanised development

Workplace	Conventional Stoping	Mechanised Dev. (Bord and Pillar)
Width/Length (m)	30	5
Height (m)	1.2	4.5
Effective Drilled Length (m)	1.2	3.2
Volume per blast (m^3)	43.2	72.0
Rock density (t/m^3)	2.7	2.7
Tonnes Broken per blast (t)	117	194
Burden (m)	0.6	0.6
Spacing (m)	0.6	0.6
Blastholes per face	100	38
Explosive per hole (kg)	0.8	4.5
Explosive per face (kg)	80	171
Powder Factor (kg/m^3)	1.9	2.4
Cost of Explosives (Rand)		
Emulsion Cost per tonne	8,500	8,500
Cost per Detonator	11.00	11.00
Cost per Booster	3.50	3.50
Cost of Explosives per Blast		
Emulsion Cost per Blast (Rand)	680.00	1,453.50
Cost of Detonators per blast (Rand)	1,100	418
Cost of Boosters per blast (Rand)	350	133
Total Cost per Blast (Rand)	2,130.00	2,004.50
Rand per tonne Broken	18.26	10.31

The ability to eliminate the use of a 12 gram booster when priming conventional stoping rounds with pumpable emulsions allows for an immediate reduction in the direct cost of explosives by more than 15%. Should the use of boosters be required, not only would there be an increase in direct cost of explosives, but their presence on the operation would impose an additional burden on the control of explosives, detracting from the advantages achieved through the implementation of the Class 5.1 pumpable emulsion system.

In the first year of the project on-going research and development led to the development and testing of two emulsion formulations that were proposed for use in small diameter holes within the narrow reef industry. For the purposes of this report these formulations are denoted ‘Emulsion A’ and ‘Emulsion B’. Though both formulations provided good results in testing, the results obtained with Emulsion B achieved greater consistency under varying operating and environmental conditions as well as better product stability thereby increasing the

possible shelf life and re-pumpability of the product. Following these observations Emulsion B was selected as a preferential explosive for continued testing with the Portable Pump and further development of Emulsion A was postponed indefinitely.

4.1.1 Emulsion Testing and Optimisation

In order to achieve the correct emulsion density for use in blasting operations, gassing tests were undertaken at the Losberg test facility in order to determine the effect of specific factors on the gassing rate and blasting density of Emulsion B. In order to sensitise the base emulsion and thus form an explosive, a small percentage of sensitising agent is mixed with the emulsion before exiting the charging lance and entering the blasthole. Depending on the specific characteristics of the emulsions and the quantity and concentration of this solution, the final density of the pumpable emulsion will differ. As no highly sensitive chemicals or explosives are added to Emulsion B during the sensitisation process, the increase in sensitivity of the emulsion is as a direct result of the generation of nitrogen gas bubbles within the emulsion. The formation of these gas bubbles decreases density from an initial density of 1.46 g/cc to within the blasting window between 1.15 g/cc and 0.9 g/cc.

Figure 23 depicts a set of Emulsion B samples taken to establish the rate of gassing under controlled conditions. Three emulsion B samples were pumped into 265ml sample cups in short succession through the use of the Portable Pump and the initial time recorded. At 15 minute intervals following the sensitisation of the samples, a straight edge was used to scrape excess emulsion from the above the rim of the sample cup before the weight of the cup was recorded. The mass of the cup was subtracted before the density of the sample could be calculated.



Figure 23, Sequence of three Emulsion B samples taken to establish gassing rate under controlled conditions.

One significant consideration in the sensitisation of emulsion is the length of time required for the emulsion to reach the desired blasting density. As the period of time between the loading and sensitisation of the emulsion and the initiation of the face can vary from 15 minutes to a number of days, it is important to ensure that the gassing rate of the emulsion is able to deliver the required density within the allotted time (taking into account the environmental conditions). If the gassing rate is excessively fast the size of bubbles formed within the emulsion may be too large leading to the collapse and desensitisation of the emulsion. If the gassing rate is too slow, the time required to reach the desired density may be too long and the emulsion may not be sensitive to detonation by the time the face is fired. In order to determine the effect of specific variables on the gassing rates and final densities achieved through the use of the Portable Pump, variables were monitored under controlled test conditions. Following analysis of the results, modifications were made to the trial equipment and formulation until repetitive tests delivered the desired results for the range of operating criteria expected within the underground environment.

Figure 24 depicts the first 60 minutes of one such set of sensitising tests where controlled changes were made to the temperature of the emulsion in order to determine the effect of temperature changes on the gassing rate of the emulsion. As a decrease in temperature results in an increase in the viscosity of an emulsion, as depicted in Figure 25, the turbulence required for efficient mixing of emulsion and sensitiser increases. This increase in turbulence necessitated an increase in the pressure ratio within the Portable Pump as well as an increase in mixer efficiency such that consistent density emulsion could be achieved under fluctuating

temperatures. Through a series of tests variables such as temperature, pressure, viscosity, acidity and the addition of active ingredients could be tested and optimised for the formulation and new emulsion technology.

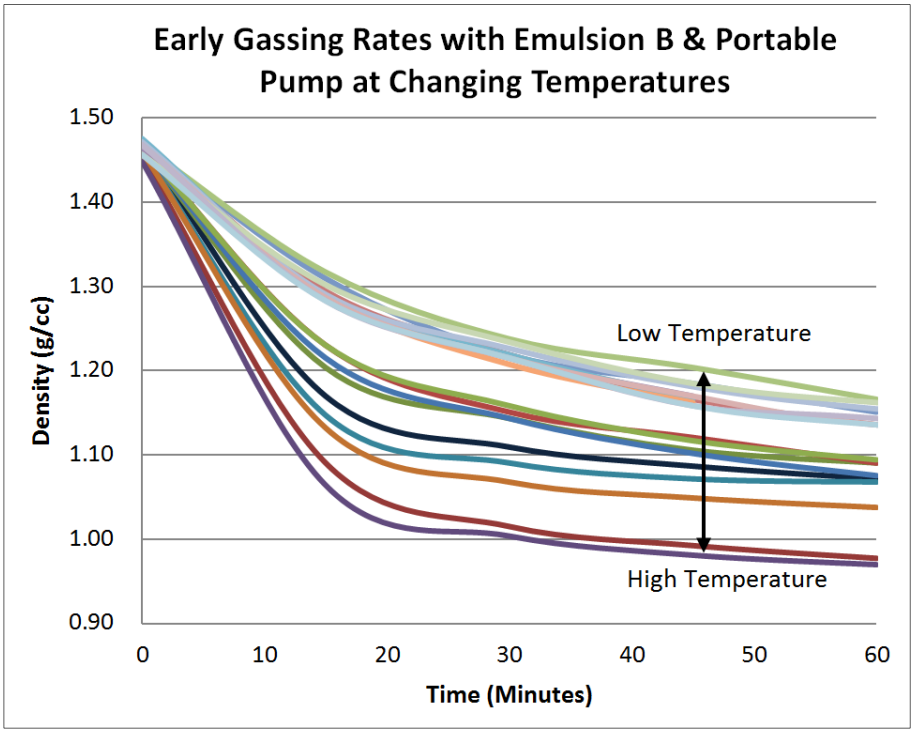


Figure 24, Effect of temperature variations on the gassing rate of an early Emulsion B formulation prior to optimisation.

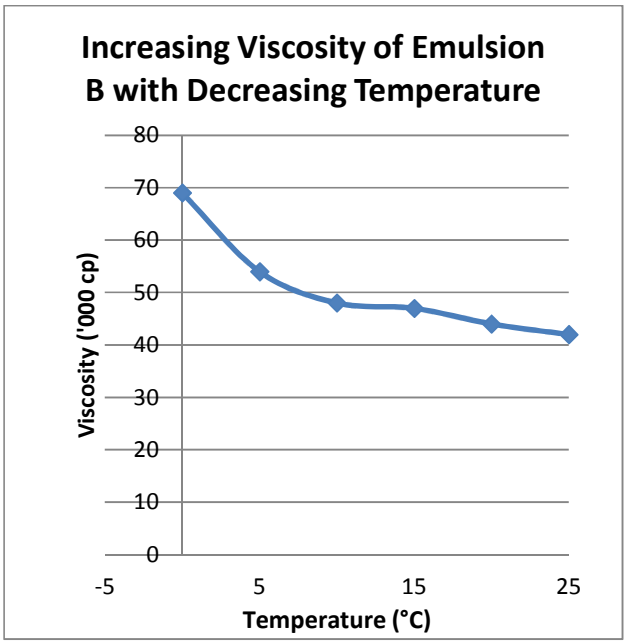


Figure 25, Effect of temperature variations on the viscosity of Emulsion B

Table 8 depicts a set of test results recorded for Emulsion A and Emulsion B under controlled operating conditions during the optimisation of the emulsion and sensitising solution. In order to obtain consistent and reliable performance from the new emulsion formulations the effects of variables such as temperature, viscosity, oxidiser acidity, sensitiser type and concentration and the addition of sensitiser enhancing catalysts were evaluated. Later tests also included the effects of pump flow rate, pressure and mixing technologies on product consistency.

Table 8, Cup sample density at 15 minute intervals through the use of the Portable Pump

Time (minutes)	0 min	15 min	30 min	45 min	60 min
Emulsion B – Sample 1, APX12, 29°C	1.46	1.15	1.1	1.06	1.05
Emulsion B – Sample 2, APX12, 29°C	1.46	1.16	1.1	1.06	1.05
Emulsion B – Sample 3, APX12, 29°C	1.46	1.14	1.1	1.06	1.05
Emulsion A – Variable A, Sample 1	1.48	1.39	1.34	1.29	1.28
Emulsion A – Variable A, Sample 2	1.49	1.39	1.34	1.29	1.28
Emulsion A – Variable A, Sample 3	1.5	1.39	1.34	1.29	1.27
Emulsion A – Variable B, Sample 1	1.49	1.42	1.38	1.36	1.33
Emulsion A – Variable B, Sample 2	1.49	1.44	1.4	1.38	1.36
Emulsion A – Variable B, Sample 3	1.48	1.42	1.38	1.35	1.34

4.2 The Development of Charging Equipment for Conventional Tabular Mining Operations



Figure 26, Portable Pump as implemented in the Rand Uranium narrow reef trial

Possibly the greatest challenge faced in the introduction of pumpable emulsion systems to narrow reef operations is the management of pump technology required for their implementation. As pumpable emulsions are transported underground as a Class 5.1 oxidiser, the ability to 'manufacture' explosives in the underground environment is determined almost entirely by the reliability and consistency of the equipment used in charging operations. While it is comparatively simple to guarantee the quality of explosives manufactured at production facilities, the ability to identify 'out of spec' sensitised emulsion in underground operations depends on the ability and training of the pump operator. Traditional charging units previously required for the implementation of pumpable emulsions on mechanised underground are complex machines and as such have necessitated the allocation and training of a skilled technician or artisan with each unit in order to ensure the correct performance of the charging unit.

Given the number of pumps required for the implementation of pumpable emulsions within the narrow reef environment, it is no longer possible for every charging unit to be accompanied by a trained technician to ensure the quality of explosive manufactured at the blastface. With this in mind, the reliability of the Portable pump and its ability to deliver consistent sensitised emulsion without continual calibration were deemed essential to the success of the project. Given the level of skills and training within the workforce, safe operation of the pump technology was paramount throughout all operating conditions and all possible failure modes. All possible risks were to be identified, and multiple fail-safe modes incorporated into the charging equipment design so as to eliminate the possibility of dangerous pumping conditions.

As technicians were no longer available during daily charging operations, it would no longer be possible to 'guarantee' the quality of explosive delivered to each blasthole during charging operations. For this reason a decision was made to remove the ability of individuals to adjust the manufacturing parameters of sensitised emulsion in the underground environment. In order to allow this to take place, the consistency and repeatability of the charging equipment was essential such that uniform settings applied to all charging equipment would produce consistent sensitised emulsion on all pumps in use throughout the operation. This outcome needed to be achieved despite variable operating conditions that included temperature fluctuations and changing air pressure throughout the operation based on both workplace and on time of day.

A significant learning during the development phase of the project was the requirement for a simple and reliable user interface. As equipment operated within the underground environment is typically simple to use and extremely robust, mine personnel have become accustomed to solving problems through the use of force, rather than through a methodical thought process. While traditional charging equipment allows for a more complex layout owing to the higher level of skill of the operator, the interface of the Portable pump needed to be simplified to make available only the essential components for the operation of the unit. As overcharging is widespread throughout the underground industry, it was deemed vital to include a function facilitating the delivery of a predetermined mass of emulsion per blasthole such that overcharging could be prevented.

4.3 The Influence of Equipment on the Feasibility of the Pumpable Emulsion System

Before considering the leverage of the system on other aspects of the operation, an understanding needs to be obtained for the difference in the explosives related expenditure for an operation. When considering the overall cost of an explosives system, three broad areas of costs need to be taken into consideration. These three areas include the direct costs of explosives, the logistical, capital and operating expenses for the system and the downstream implications of the explosive system on daily mining activities and revenue generation.

While bulk pumpable explosives systems are able to offer a reduction in the direct cost of explosives, logistics and storage requirements, and allow for increased levels of efficiency throughout the mining cycle, initial capital is required for the procurement of charging equipment and storage facilities. In order to justify the increase in capital expenditure required for the implementation of the pumpable emulsion system, an adequate level of equipment utilisation needs to be achieved in order to offset the costs incurred through the implementation of the system. The importance of utilisation to the overall cost of the explosives system is illustrated in Figure 27. As the capital and maintenance expenses required for the implementation of a charging unit increase, the utilisation of charging unit also needs to increase in order to offset the increase in fixed and operating expenditure. Though costs will vary for different mining operations, Figure 27 illustrates the importance of maximising the utilisation of charging equipment. Manufacturing and maintenance data used in the calculation of the cost curves was obtained from original equipment manufacturers (OEMs) and comparative labour and logistics costs include the cost of unit operators and technicians for mechanised equipment as well as the equipment necessary for the transport of explosives throughout the operation. A detailed discussion of costs regarding the narrow reef emulsion system and Portable pump is included in Section 7 of the report.

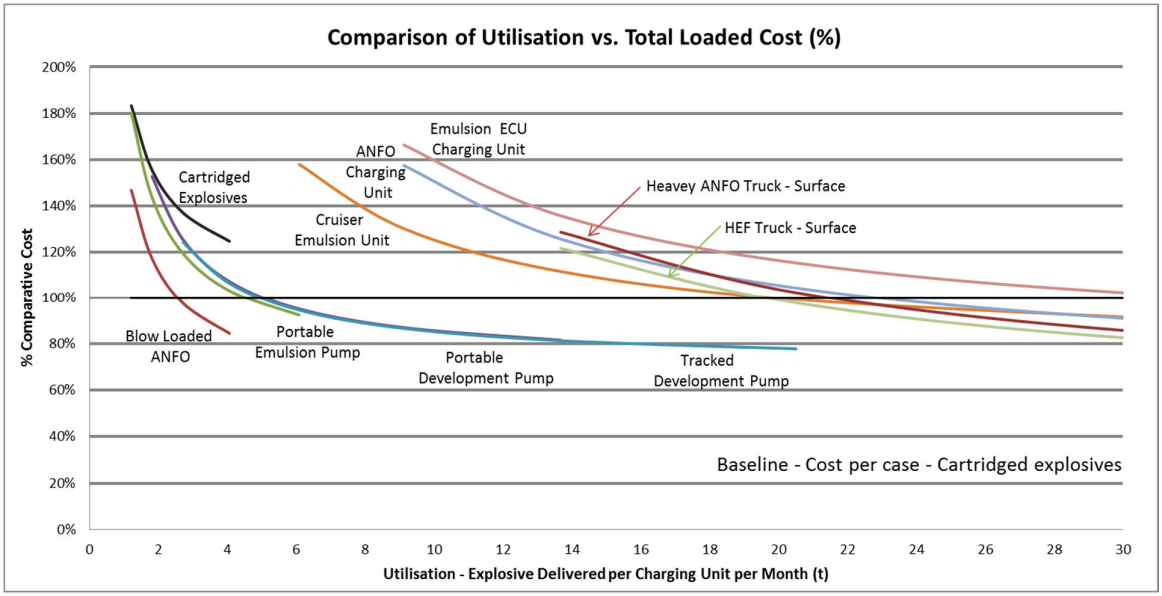


Figure 27, Significance of the equipment utilisation on the overall cost of explosives (Calculation excludes explosives waste, initiation systems & magazine facilities)

As charging equipment cannot easily be moved within the narrow reef environment and blasts are limited in size and undertaken only once per day, the utilisation of equipment within the narrow reef environment will be inherently poor. Under such conditions, the quantity of explosive that can be pumped through a charging unit will be limited to a range of 1 to 5 tonnes per month depending on the requirements of the panel. In order to allow for the feasible implementation of the narrow reef emulsion system, the system not only needed to achieve the technical requirements discussed previously, but also needed to cost magnitudes less than traditional charging equipment. For this reason design of the Portable pump was optimised on an on-going basis such that manufacturing and maintenance expenses could be reduced to within acceptable levels.

5 Phase 2 – Losberg Controlled Emulsion Tests

5.1 Losberg Test Procedure

In order to optimise the performance of the new emulsion formulations and emulsion charging equipment (Portable pump), tests were undertaken at BME's Losberg manufacturing and test facility on a regular basis throughout the development phase of the project. Tests undertaken during the development phase investigated the reliability and consistency of sensitised emulsion manufactured through the charging equipment, together with the sensitivity to initiation and gassing rate of the new emulsion formulations under a range of operating and environmental conditions. Once an acceptable level of performance had been achieved by both the charging equipment and the selected emulsion formulation, Emulsion B, the range of tests were expanded to include tests aimed at characterising the performance of the formulation. During these tests a range of diameters, degrees of confinement and priming practices were tested to assess their impact on the performance of Emulsion B.

5.1.1 Determining the Velocity of Detonation

Of primary importance in the performance characterisation of the new emulsion formulation was the determination of the velocity of detonation (VOD) so that the quality and consistency of detonation could be recorded. As VOD is influenced by the diameter and confinement of the blasthole, both confined and unconfined tests in increasing pipe diameters were required such that they could be compared to existing results for explosives within the underground environment. As the diameter of blastholes within underground narrow reef operations typically range between 28mm and 38mm in size, and underground mechanised operations typically blastholes 45mm in size, a range of pipe sizes representative of these hole diameters were selected for VOD tests. A minimum internal diameter of 28mm, and a maximum internal diameter of 45mm were selected as extremes and as the blasthole diameter most commonly used on narrow reef gold mines appeared to be 36mm this was selected as the intermediate diameter for VOD tests.

In order to compare the effects of higher and lower confinement on the VOD of the new formulation and to allow for the evaluation of priming practices as discussed below, steel and PVC pipes were used during the VOD tests. Specific pipe diameters selected for the trial are listed in Table 9 together with their respective wall thicknesses. While PVC pipes with a wall thickness between 2 and 2.5mm were available in 27.8mm, 36mm and 45mm internal diameters (S.A.N.S 967, rated for atmospheric pressure), steel pipes were not as easily available in these diameters. The internal diameter of steel pipes selected for the test included 27.8mm, 37.1mm, and 45.1mm with a greater range of wall thickness ranging from 1.65mm (Sch5) to 3.05mm (Sch10).

Table 9, Comparison of Internal Pipe Diameter and Wall Thickness

Comparison of Internal Pipe Diameter & Wall Thickness		
Nominal Pipe Size	PVC (S.A.N.S 967)	Steel
28 mm	27.8 x 2.1 mm	27.8 x 3.05 mm (Sch10)
36 mm	36.0 x 2.0 mm	37.1 x 2.95 mm (Sch10)
45 mm	45.0 x 2.5 mm	45.1 x 1.65 mm (Sch5)

Due to the use of steel pipes in the confined VOD tests, heavy machinery was required for the tests in order for the pipes to be buried before firing. In this way shrapnel can be prevented. The use of this technique added to the expense and time required in order to determine the confined VOD results. As the Losberg facility has a test cell for VOD tests of unconfined samples, PVC pipes could be detonated within the cell without the need for heavy machinery. It was therefore proposed that should it be possible to define a relationship between the confined (steel) and unconfined (PVC) test results it may be possible to avoid confined (steel) VOD tests in future research. Use of the test cell for VOD tests could be undertaken faster and at a lower expense than confined VOD tests, simplifying the initial comparison of new small diameter emulsion formulations.

5.1.2 Determining the Effect of Priming on the VOD of Emulsion B

As discussed in section 2.3.2 it is evident that the use of varying priming practices will have an effect on the VOD achieved within a blasthole. This was discussed by Conradie (2006) for

large diameter blastholes within the open cast environment. As it was determined that varying priming practices were required within the third phase of the project to prevent the detonator being extracted from the blasthole, an understanding was required of the effect of each priming practice on the VOD within the blasthole. This is discussed in further detail in Sections 6.1.1 and 6.2.2.

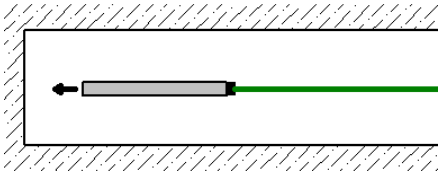
A second area of concern was that should detonators be positioned within 'sludge' or water at the toe of the hole without the assistance of a booster or cartridge, they may not provide sufficient energy for the detonation of the sensitised pumpable emulsion. For this reason both forward and reverse priming practices needed to be evaluated within a simulated blasthole of controlled confinement such that any additional effects brought about through priming practices could be analysed. As the condition and confinement of blastholes within the rock mass differ due to the presence of discontinuities in the rock mass, these variables would have influenced VOD results obtained within a test. In order to simulate consistent confinement steel pipes were prepared as indicated in Figure 28 with a single end welded closed (using a 2mm steel disk) in order to simulate the reflection of shock waves and pressure build-up from the toe of the blasthole.



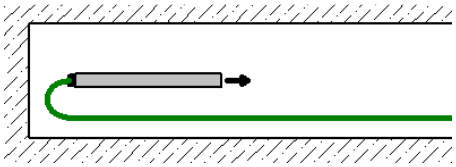
Figure 28, Steel pipes with a single end welded closed prepared for VOD Test 4

Given the priming practices utilised in underground field trials, priming practices within the controlled test evaluated the following priming practices:

- Forward priming with an 8D detonator



- Reverse priming with an 8D detonator



- Forward priming with a 12 gram stinger booster,
- Forward priming with a 200 gram, 29 x 270mm emulsion cartridge

(All holes were toe primed)

5.1.3 Test Procedure and Equipment used for the Determination of VOD

VOD Test 1- Emulsion A

Over the duration of the project, four separate VOD tests were undertaken for the evaluation of explosive performance and the determination of VOD. The first set of VOD tests, Test 1, was undertaken on Emulsion A early in the project in order to test the ability of the new Portable Pump to deliver consistent density emulsion into a pipe column such that it could be reliably detonated with an 8D detonator. The test procedure for the VOD test involved the use of the Portable Pump to load 1 metre 27.8mm internal diameter steel pipes with a single end taped shut, such that the pipes could be detonated under controlled conditions for the determination of VOD. An AEC Electronics Model VOD-3 VOD timer was used to record the VOD of each test pipe. Results were obtained by drilling two small holes in the test pipe a predefined distance apart before the two 2.2mm fibre optic sensor lines were taped into the

two holes. On detonation of the test pipe, light emitted by the detonation front passing the two fibre optic cables would be detected by the photodiodes within the VOD-3 VOD timer and in so doing start and stop the microsecond counter recording the time for the detonation front to travel the predetermined distance. The measurement and calculation of VOD was undertaken over a length of 30cm across the centre of the pipe in order to eliminate any effect that the 'run-up' and low confinement at the end of the test pipe may have on the measured VOD. The test layout is depicted in Figure 29 below.

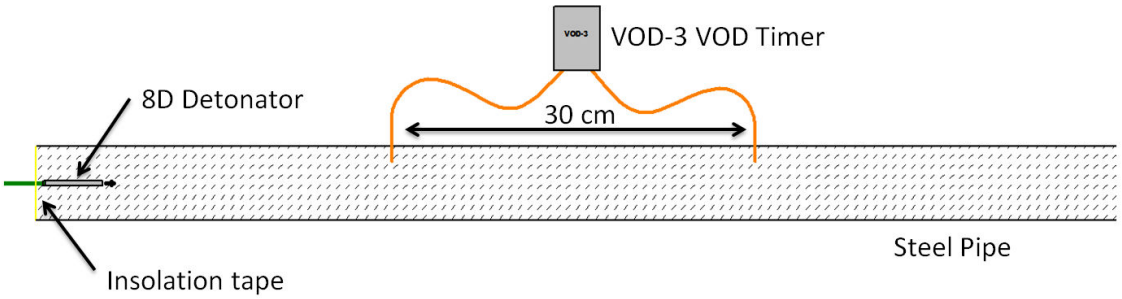


Figure 29, Test 1 – Recording of VOD in steel pipe with AECE VOD-3 VOD timer

While attempting to the load the first pipe it became evident that while the 19mm (internal diameter) charging lance could be inserted into the steel pipe, pressure build-up within the lance on activation of the pump caused the pipe to expand preventing the extraction of the lance from the pipe. In order to allow the pipes to be charged an air hole was punctured in the taped end of the pipe the emulsion was pumped through the test pipes during the filling process. Once blasting densities had been reached, the pipes were buried one at a time and fired with an 8D detonator inserted into the end of the pipe pointing toward the centre of the explosive column (simulating reverse priming). While all seven test pipes detonated, only 4 VOD readings were retrieved from VOD-3 VOD timer. As the continued development of Emulsion A was later postponed, Emulsion B was used for the remainder of the VOD tests.

VOD Test 2- Emulsion B

The second set of VOD tests undertaken during the project was conducted in order to determine the sensitivity of Emulsion B to initiation within unconfined small diameter blastholes. The test was undertaken in open ended PVC pipes with a wall thickness of 2mm and internal diameters of 27.8mm and 36mm in size. The planned measurement of data was undertaken utilising the available VOD-3 VOD timer as used in test one. During the test, the Portable Pump was used to load the Emulsion B formulation into the test pipes for testing. For the 27.8mm PVC pipes, the same loading procedure was used as that described in VOD Test 1 as depicted in Figure 30. As the charging lance could slide easily within the 36mm pipe, the end of the 36mm pipes was closed with tape and the pipes were charged from the bottom up as in standard mine practice. During the test, five 27.8mm pipes and four 36mm pipes were tested. All pipes where reverse toe primed prior to firing (from the end of the pipe with the 8D detonator pointing toward the centre of the explosive column).

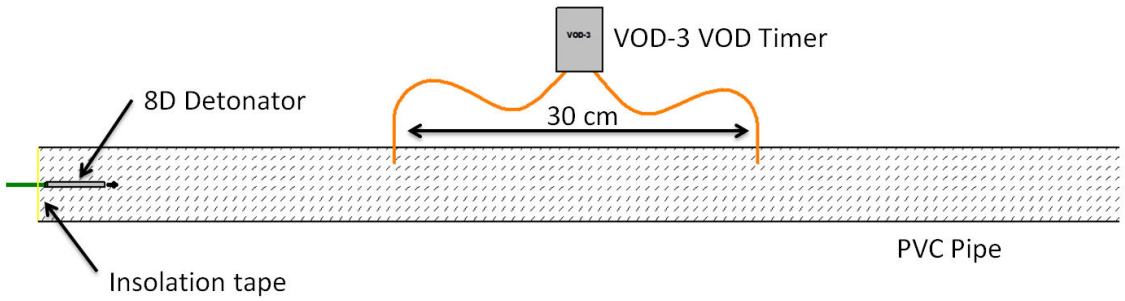


Figure 30, Test 2 – Recording of VOD in PVC pipe with VOD-3 VOD timer

VOD Test 3- Emulsion B

Test 3 was planned to compare the confined and unconfined VODs of Emulsion B in nominal 28mm, 36mm and 45mm diameter pipes and to gain an understanding of the effects of various priming practices on the 'run up' in VOD of the emulsion column. In order to achieve this a measuring instrument with greater accuracy was required than that utilised in VOD tests 1 and 2. The measuring equipment planned for use in test 3 was the TLC SpeedVOD that utilises 'time domain reflectometry' to determine VOD through continuous feedback on

the rate of cable damage at 5 microsecond intervals. On the day of the trial however, it was decided that the RG58 cable required for the measurement of the VOD was too large to be used within the small diameter emulsion pipes without negatively influencing the quality of results obtained. For this reason priming tests were postponed. As the confinement of the PVC pipes was considered lower than that of the steel pipes, a number of PVC pipes were measured by taping the RG58 probe cable to the side of the pipe. 27.8mm, 36mm and 45mm PVC pipes used for test and were prepared as indicated in Figure 31 before the RG58 cable was taped to the side of the pipe before detonation.

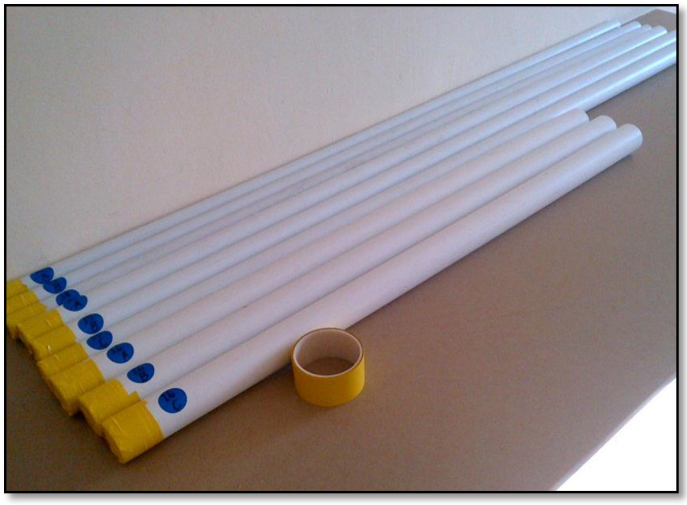


Figure 31, 27.8mm, 36mm and 45mm ID PVC pipes prepared for VOD Test 3

While VOD results were obtained for the PVC pipes, no direct correlation could be drawn between the lower VOD results obtain from outside the PVC pipe and the results obtained from within PVC pipes in VOD Test 4 for varying pipe diameters. This prevented the direct comparison of the results obtained in VOD test 3 with those of VOD tests 1, 2 and 4.

VOD Test 4- Emulsion B

The fourth set of VOD tests was planned to complete the VOD tests initially planned for the third set of VOD tests. Given the problems encountered with the use of the TLC SpeedVOD,

two MREL Blasting Instrumentation Handitrap II VOD Recorders were sourced for use in the fourth test in conjunction with MREL Probecable-HT. The MREL Handitrap uses continuous measurement (at a rate of 1 MHz) over a coaxial cable to record the rate of change in cable length during the detonation of the blasthole or pipe. As the 2mm Probecable-HT has a small cross sectional area, the volume of emulsion displaced from the pipe by the VOD cable would be negligible and the cable could thus be positioned within the explosive column for an accurate reading. In order to check that the two Handitraps were providing comparable readings, two Cartridge M emulsion cartridges were first placed end to end and fired over fixed lengths of probe cable from the two Handitraps as depicted in Figure 32.

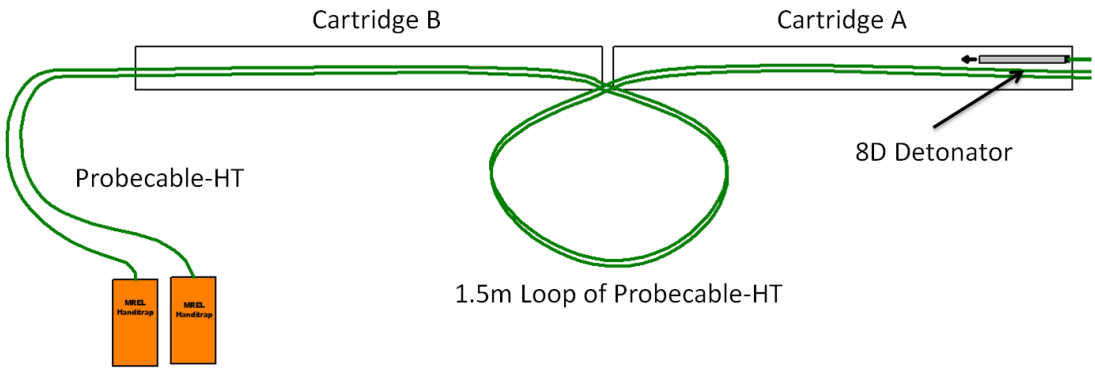


Figure 32, Test 4 – Calibration of MREL Handitrap VOD Recorders

During the fourth test, three 27.8mm, three 37.1mm and three 45.1mm steel pipes with welded ends were fired with reverse toe primed 8D detonators at the base of the pipe. These samples were fired for comparison with results obtained in previous PVC pipes and as a baseline for the effect of priming on VOD. In addition to the baseline, three 37.1mm steel pipes were fired with forward toe primed 8D detonators, three pipes with forward toe primed 12gram boosters, and three pipes with forward toe primed 8D detonators within a single Cartridge M (29 x 270mm, 200 gram) priming emulsion cartridge (as used for comparison of the Handitraps). Pipes were loaded according to mine practice with detonators and primers being pushed to the toe of the pipe through the use of the charging lance in either the forward or reverse direction and according to the prescribed priming practice for each test sample as

indicated in Figure 33. Once the detonator was positioned by the lance within the toe of the pipe the pump was activated to fill the pipe with Emulsion B.

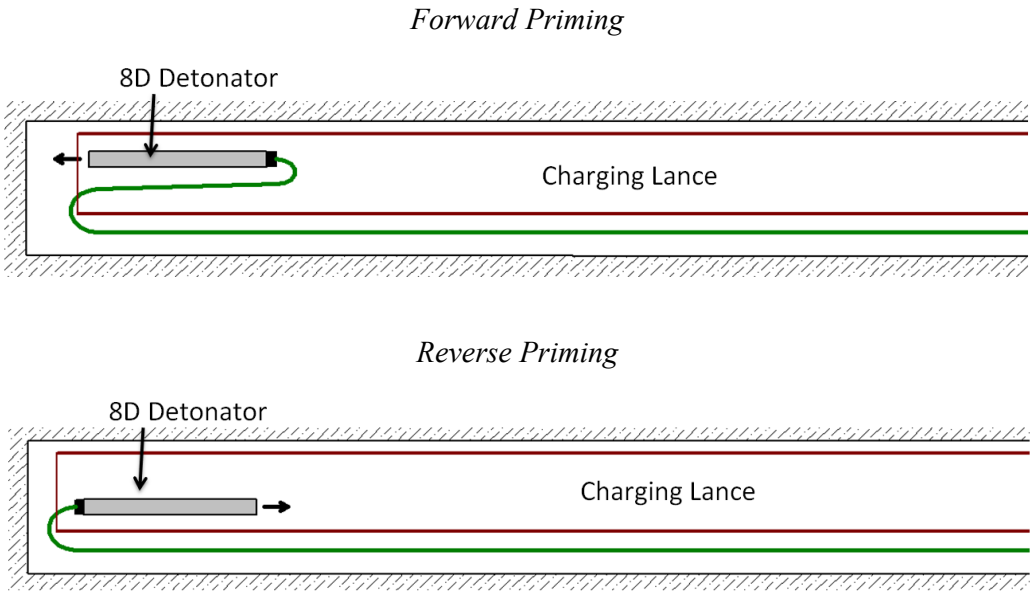


Figure 33, Method for forward and reverse priming of closed end VOD test pipes

In order to ensure the pipe did not get stuck within the 27.8mm steel pipe a marginally smaller charging lance was used in the test to allow for extraction of the charging lance while loading. Where Cartridge M was used for priming the detonator was forward primed within the cartridge prior to insertion into the test pipe as practiced in underground operations. The pipe was then charged with emulsion behind the cartridge. Cartridge M was selected for priming tests as it was an intermediate strength cartridge that had a similar VOD to both Cartridge L and Cartridge H that were to be used in trials on Rustenburg Minerals and Rand Uranium. A summary of the cartridged and pumpable emulsion explosives used in the latter half of the study is depicted in Table 10 below.

Table 10, Comparison of Cartridged Emulsions used in Losberg Tests & Field Trials

Explosive	Nominal Strength	Density (g/cm^3)	Relative Weight Strength	Relative Bulk Strength	
Emulsion B	-	1.15	0.92	1.32	
Cartridge L	Low	1.16	0.94	1.34	(Used in RMDC trial)
Cartridge M	Medium	1.16	1.03	1.45	(Used in Priming Tests)
Cartridge H	High	1.16	1.09	1.53	(Used in RU trial)

5.2 Losberg Test Results & Discussion

5.2.1 Correlation of MREL Handitrap VOD Recorders

At the outset of the fourth VOD test the two Handitrap instruments (instrument identification numbers 153 and 217) required for the test were tested simultaneously in order to correlate the results achieved by the two units. As Cartridge M was readily available due to its requirements for priming tests later in the day it was used for the comparison of the two Handitrap instruments. For the comparison, two unconfined 29 x 270mm emulsion cartridges were fixed linearly end to end before the two Probe cables for the two Handitraps were taped linearly to the side of the cartridges. During the placement of the HT-Probecable, a 5 metre tail was measured for both cables before the cables were taped to the first emulsion cartridge. Between the two cartridges, 1.5 metres of cable from both Handitraps was left in a loop before the cables were taped to the second cartridge as evident in the trial setup in Figure 32. Graphs from the two Handitraps are evident in Appendix 1 while a comparison of the two results is indicated in Table 11.

Table 11, Comparison of Results for Handitrap 153 and Handitrap 217

	VOD Test 1 Cartridge A	VOD Test 1 Cartridge B	Recorded Gap (m)
Handitrap 217	3937	3957	1.557
Handitrap 153	4084	3954	1.553
Comparison	3.7%	0.1%	0.3%

On comparison of the VOD results from Handitrap 153 and Handitrap 217 a difference in VOD of 3.7% is evident for cartridge A while a difference in VOD of 0.1% is evident for cartridge B. During the detonation of the explosive column, the 1.5 metre length of free probe cable between the cartridges was cut on the transition between cartridge A and cartridge B. The readings measured for the lengths of cable cut at the transition between the cartridges indicates a difference of only 0.3%, 4mm over the length of the 1.5 metre loop. It is apparent from the length of the VOD trends for cartridge B and the gap in the beginning of the VOD trend for the Handitrap 217 that the probe cable was cut by the end of cartridge A. This accounts for the 5 cm increase in the length of probe cable cut between the two cartridges. As the above results compared favourably, no further comparison was made between the results obtained from the two Handitraps.

5.2.2 VOD Test Results

The VOD test results obtained for Emulsion A and Emulsion B in VOD Tests 1, 2 and 4 are tabulated in Appendix 2, Appendix 3 and Appendix 4 respectively, while all graphs for VOD Test 4 are included in Appendix 5. Results obtained for each variant of pipe diameter, confinement and priming method recorded for Emulsion B are plotted in Figure 34, with the average VOD obtained for each variant of the tests indicated with a blue diamond. In addition to the average VOD, the range of VOD results for each test is demarcated on the graph with a red square indicating the highest VOD and the green triangle the lowest VOD obtained for each variant of the test. In order to allow for a comparison of the various priming practices within the confined steel pipes, the initial VOD over-run for each variant is demarcated with a purple dot. In initial VOD tests, up to 5 pipes were fired per variant in order to test the consistency of Emulsion B within a range of pipe diameters. Due to the number of variables tested in later tests however, the number of samples per test variation was reduced to only 3 test pipes. Of the 37 VOD test pipes fired with Emulsion B in VOD tests 2 to 4, 30 VOD readings were obtained, 6 readings were lost and 2 pipes failed to detonate. Of the 6 readings lost, 3 were lost due to electromagnetic interference resulting in the pre-trigger of the TLC SpeedVOD, 1 was due to a failed fibre optic pickup on the AECE VOD-3 VOD Timer and 2 due to insufficient trigger/tail length on the MREL Handitrap II.

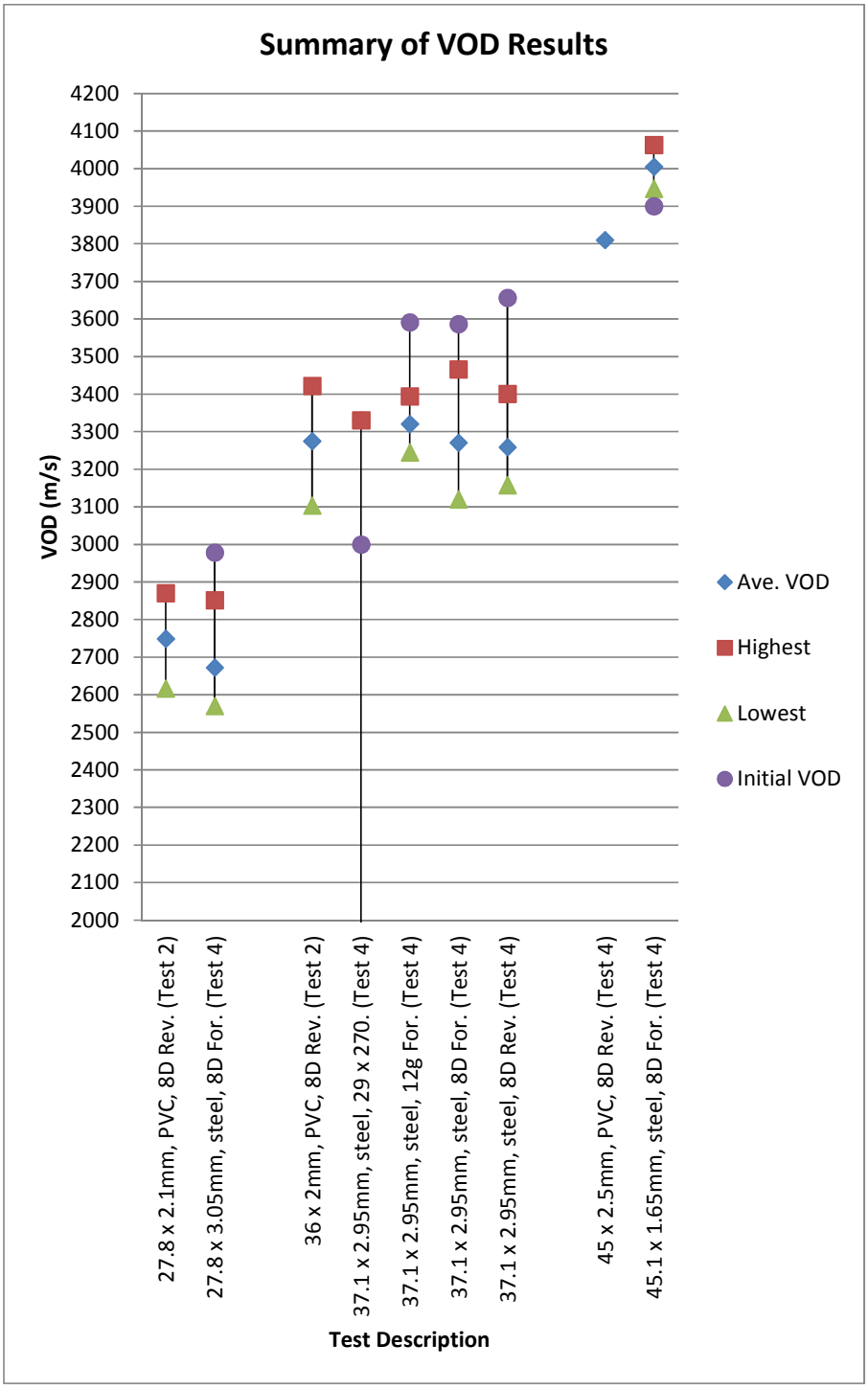


Figure 34, Summary of VOD Results for VOD Test 2 and VOD Test 4

Despite the small sample size, consistent VOD results were obtained for all tests undertaken with Emulsion B. As mentioned previously, 2 pipes failed to detonate within test sequence 4. Both of these were from the same variant, '35 x 2.5mm, steel, 29 x 270mm emulsion primer'. During VOD Test 2, blasting densities were allowed to gradually decrease through the test in order to establish an understanding for the change in VOD with decreasing density. No definable decrease in VOD was evident with decreasing density, but rather a marginal increase in VOD was evident with decreasing density in all 28mm PVC pipes tested in VOD tests 2 and 4. While this may be influenced by other variables, it may indicate an increase in the sensitivity of the pumpable emulsion with decreasing density due to the increased presence of bubbles in the emulsion (ISEE, 1998).

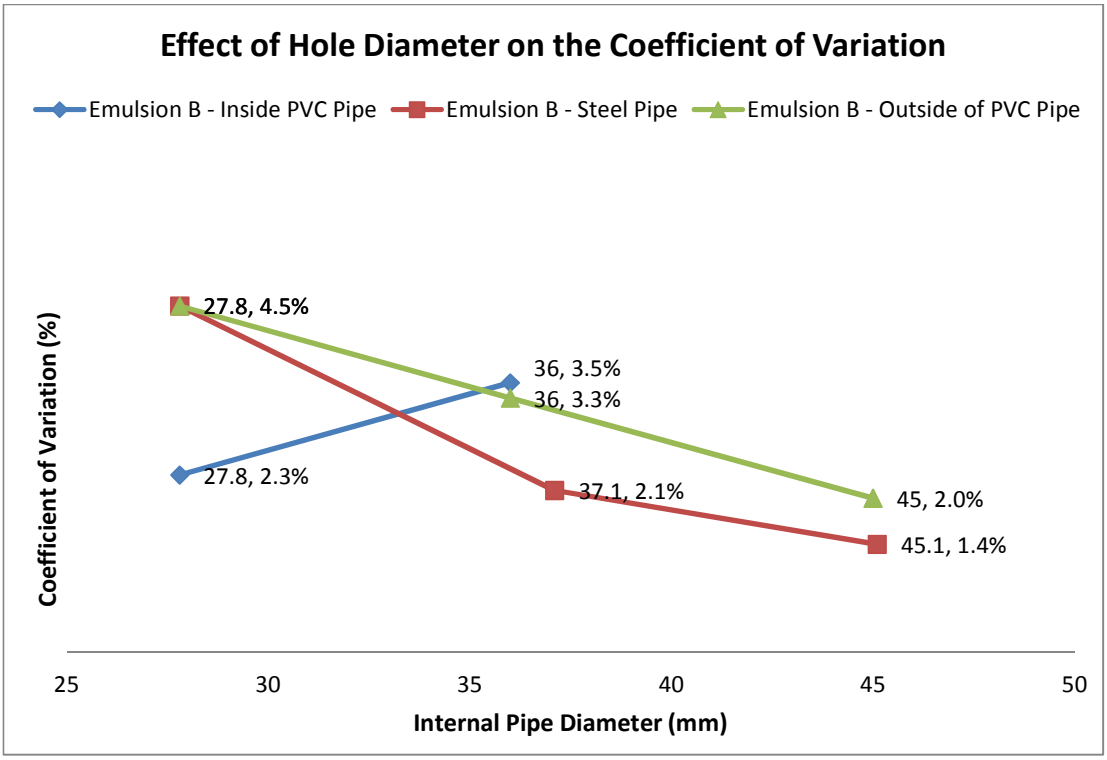


Figure 35, Coefficient of Variation for Emulsion B VOD results

The coefficient of variance for VOD results obtained in PVC and steel pipes is indicated in Figure 35. Both internal and external VODs for the PVC pipe tests are indicated in the graph. It is clear from the trends above that as the diameter of the explosive column increase, the

coefficient of variance for the VOD of test pipes decreases. The calculated coefficient of variance for VOD results measured in steel pipes reduced from 4.5% in 27.8mm pipes to 1.4% in 45.1mm pipes, while those measured on the outside of the PVC pipes reduced from 4.5% in 27.8mm pipes to 2% in 45mm pipes. While the coefficient of variance for VOD results inside and outside the 36mm PVC pipes is similar, the results obtained inside the 27.8mm PVC pipes indicated a lower coefficient of variance. This result appears abnormally low with three of the five samples in this test producing a VOD close to the average for the group.

5.2.3 Discussion of Losberg Test Results

- *Effect of Priming on VOD Results*

Amongst the results obtained in VOD Test 4 and depicted in Figure 34, are a range of VOD results obtained through various priming practices within 37.1mm steel pipes with a closed (welded) end. As varying priming practices had been undertaken in underground trials, an understanding was required of the impact of each method on the performance characteristics of Emulsion B in underground trials. Should a preferred priming method be identified, this method could be implemented in order to achieve the best possible results in underground trials. From the results obtained, no substantial benefit could be seen through the use of any of the four priming practices tested. Results obtained through the use of a 12 gram pentolite booster produced the highest average VOD, while the use of reverse primed 8D detonators indicated the lowest coefficient of variance at 2.1% and a slightly higher initial VOD than other priming methods. Results obtained through forward priming with an 8D detonator indicated a slightly lower initial over-run in VOD and a broader coefficient of variance than the tests with reverse primed 8D detonators. The marginal increase in average over-run and decrease in the coefficient of variance seen in tests undertaken with reverse primed 8D detonators is likely the product of reverse priming (Conradie, 2006; ISEE, 1998).

Through comparison of the above results it would appear that the highest and most consistent VOD results may be achieved through the combination of reverse priming and the use of a 12 gram booster. These results appear to be in line with the theory discussed in section 2.3.2, however cognisance must be taken of the limited sample size and the 6% variation in density in the fourth set of VOD tests. During underground blasting operations the use of a 12 gram

booster is of greater importance than in the ideal priming tests discussed above. Should contamination of emulsion occur at the base of a blasthole due to the presence of water or grit, the increased quantity of energy delivered by the 12 gram booster would be beneficial by providing the initial shock wave to a greater area at the toe of the blasthole, increasing the likelihood of detonation.

As priming during the trial required the use of a cartridge at the base of each hole (to hold the detonator within the blasthole), priming practices through the use of a cartridge were also included in priming tests. This test provided the only significantly unexpected result. Of the three 37.1mm steel pipes primed with a 29 x 270mm, 200 gram emulsion cartridge, two of the pipes failed to detonate completely. For both failed test samples a noticeably lower air pressure wave was experienced and on both instances the last 50cm of the 1.5 metre test pipe remained in good condition when dug from the test pit. The last of the three samples primed with the emulsion cartridge detonated with a VOD in line with the average results obtained for 37.1mm steel pipes, marginally above 3300 m/s. The initial VOD of this sample however indicated a lower than expected initial VOD of only 3000 m/s before building to steady state VOD. Given the mass of the primer and the comparatively high VOD of Cartridge M, an initial VOD of approximately 4000 m/s was expected for the test.

As the three pipes primed with emulsion cartridges were the last three pipes to be charged in the fourth VOD test it is possible that the sensitising agent prepared for the test may have run out prematurely or that an unforeseen error in the equipment may have resulted in incorrectly sensitised Emulsion B. As cup samples taken sequentially with test pipes were monitored for firing density, the error may have occurred after the cup sample for the three test pipes had been taken thereby concealing a possible error in test pipe density. Given the unexpected result no definitive conclusion can be drawn regarding the influence of Cartridge M on the VOD of Emulsion B. It would appear however from the single result obtained in the VOD tests together with the theory discussed in the literature review that no beneficial effect on VOD would be achieved through the use of cartridge emulsions in underground priming practices. Further testing is however required to obtain a greater understanding of the use of cartridged emulsions in priming applications under high and low confinement.

- *The Effect of Diameter and Confinement on the VOD of Emulsion B*

Figure 36 illustrates the VOD trends for Emulsion B in steel and PVC pipes at increasing pipe diameters. Also plotted on the graph are unconfined VOD readings at increasing for Cartridge M used to prime the three test samples in VOD Test 4, and the average VOD achieved for pumpable Emulsion A in a single pipe diameter in VOD Test 1. VOD results for the unconfined Cartridged M were obtained from quality control tests at the Losberg manufacturing facility (Bezuidenhout, 2012).

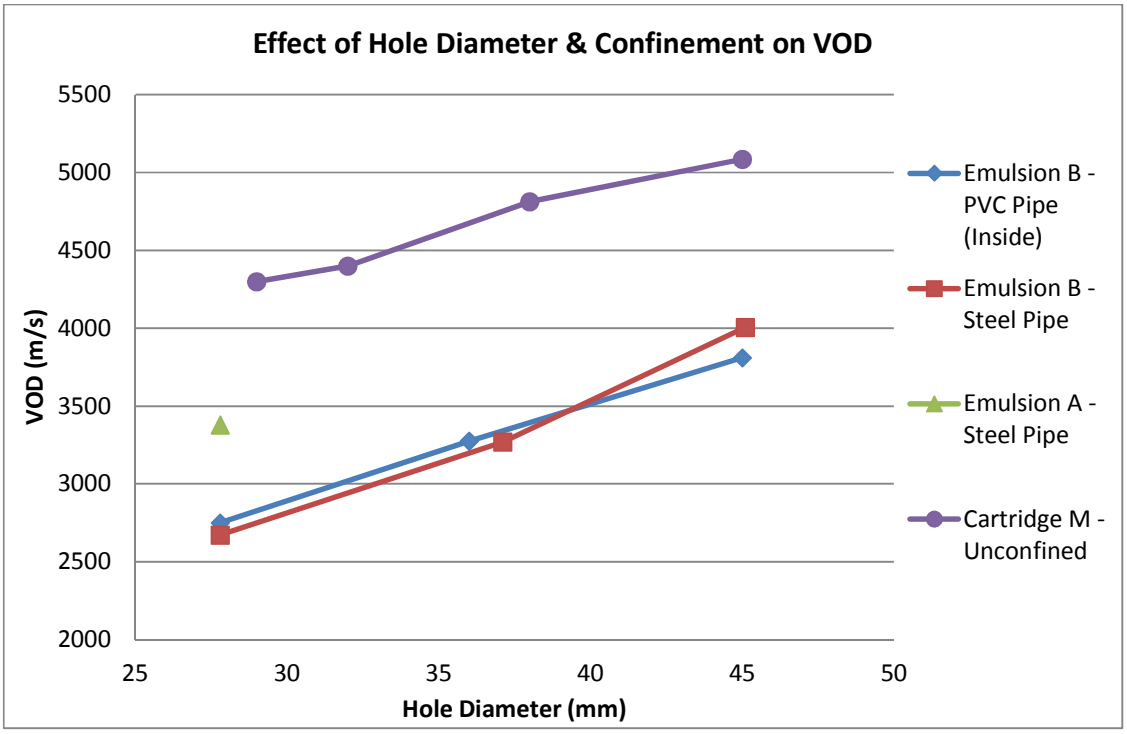


Figure 36, Velocity of detonation (VOD) for Emulsion B and cartridged emulsion plotted against pipe diameter

As expected, all three trends in the results indicate an increase in VOD with increasing charge diameter. Cartridge M achieved an average VOD of 4298 m/s for unconfined 29mm diameter cartridges and increased to an average VOD of 5085 m/s for 45mm cartridges. Results obtained for Emulsion B within PVC pipes begun with a comparatively low average VOD of

2749 m/s for 27.8mm diameter pipes, increasing to 3275 m/s for 36mm pipes and to 3810 m/s in 45mm pipe. Only a single reading was obtained for the 45mm PVC pipe. VOD results for Emulsion B in steel pipes begun at a similar average VOD of 2672 m/s in 27.8mm pipes increasing to 3270 m/s for 37.1mm pipes and 4005 m/s in 45.1mm pipes. Prior to testing, it was expected that due to the 'higher' confinement within steel pipes compared to PVC pipes, the theoretical critical diameter would be lower and the VOD within the steel pipes greater. The results depicted in Figure 36 were therefore unexpected as no apparent difference is evident in the VODs obtained for steel and PVC pipes in small diameter charges. Possible contributing factors to the VOD results include an increase in the sensitivity of the emulsion in the PVC pipes due to a marginally lower density, and the presence of the 3mm diameter shocktube inside the emulsion column in steel pipes and not within the PVC pipes (as the steel pipes were primed from a single open end).

In the test results obtained by Thomas and plotted in Figure 7, Thomas was able to demonstrate that class 4 (50mm x 1.5mm wall, 4 bar pressure rating) and class 9 (50mm x 2.2 wall, 9 bar pressure rating) PVC pipe were able to provide a degree of confinement to the explosive column and that VOD results increased with increasing confinement. While the SANS 967 PVC pipe used in the Losberg test was not rated for use in pressurised applications the 50mm pipe (used at Losberg) had a wall thickness of 2.5mm in comparison to the 1.5mm wall thickness of the class 4 pipe used by Thomas. This would suggest that the PVC pipe used in the Losberg VOD tests may have offered greater confinement than the pipes used by Thomas despite the absence of a pressure rating. As the loaded density of ANFO was only 0.78 g/cc in the tests undertaken by Thomas, a direct comparison of the results obtained by Thomas and the results obtained in the Losberg VOD tests is not possible.

From analysis of the VOD results and the data from similar studies it is evident that while PVC pipes where selected to obtain VOD results under poor confinement, a greater level of confinement was experienced than originally anticipated thereby increasing the VOD results achieved. Conversely, steel pipes selected to simulate VOD results under a higher degree of confinement proved to be a poor confining medium yielding lower than expected VOD results. Given the close similarity between the VOD results obtained in PVC and steel pipes of similar wall thickness, it is evident that steel pipes offer no definable increase in confinement over PVC pipes in small pipe diameters. In order to gain a better understanding of pumpable emulsions in small diameter blastholes, further testing is necessary to ascertain both the confined VOD and critical diameter of the Emulsion B. Plastic sleeving could be

used in future testing to obtain the unconfined VOD of Emulsion B (Thomas, 2005), while VOD tests in good condition blastholes of varying diameter are required to determine the confined VOD of the emulsion at small diameters.

A single set of VOD tests undertaken on Emulsion A in 27.1mm steel pipes are indicated with the single green point in Figure 35. As the sensitivity to initiation of Emulsion A was higher than that of Emulsion B, the average VOD achieved in the test was greater at 3440 m/s for 27.1mm pipes. It is clear from the VOD results obtained for Cartridge M and Emulsions A and B that the VOD of chemically sensitised pumpable emulsions is not as high as that of class 1 cartridged emulsions. Reasons for the difference in VOD include the reduced sensitivity, reduction in specific energy and lower detonation pressure of the explosive due to the elevated water content required for class 5.1 classification as well as the larger bubble size due to the non-ideal in-field sensitisation of the emulsion. The results indicate that it is however possible to achieve higher and lower VOD pumpable emulsions for small diameter blastholes by increasing or decreasing the energy and sensitivity of the formulation.

6 Phase 3 – Narrow Reef Trials with Pumpable Emulsion

6.1 Field Trial Procedure and Data Collection

Nearing the completion of phases one and two of the project, a suite of equipment was available for the implementation of field studies in order to determine the suitability of pumpable emulsions and the new Portable Pump for use on narrow reef mining operations. In order to compare the effects that pumpable emulsion systems would have on different types of mining operations, phase three of the project included a study of the implementation of the emulsion system within on two separate types of underground narrow reef operations. In order for the field studies to begin two separate sites were identified for implementation such that the performance of the pumpable emulsion system could be monitored on multiple faces within the operations.

The two initial sites approached for the trial studies included Rustenburg Minerals Development Company (RMDC), a chrome mine in the North Western Limb of the Bushveld complex, and a gold mine situated in the Freestate gold fields. Complications with the Freestate operation later led to its substitution by Cook 2 shaft in the West Wits Basin held by Rand Uranium (RU). As the Freestate trial was originally planned to follow the Rustenburg Minerals trial, the Rand Uranium trial began a number of months after the Rustenburg Minerals trial. Given the earlier start date of the Rustenburg Minerals Trial, a number of the lessons that had been learnt in the Rustenburg Minerals were implemented from the outset of the Rand Uranium Trial. Over this period of time revisions had also been made to the Portable Pump in order to allow for increased reliability and an easier user interface for the connection of emulsion and sensitiser bags. The increased ease of use and reliability of the system allowed for reduced maintenance and greater focus on the performance of the Emulsion B, as well as the implications of the pumpable emulsion system on the mine as a whole. As both operations considered the pumpable emulsion technology a possible replacement to their existing explosives systems, the results of the pumpable emulsion system were to be compared to the results of their existing cartridged explosives systems.

6.1.1 The Requirement for Stemming

During initial trials with the pumpable emulsion system it became apparent that despite successfully sensitising Emulsion B during the loading process, in excess of 10% of 1.2 metre stoping blastholes failed to fire during the blast. Through initial tests it became apparent that individual detonators within the panel were being pulled from blastholes before they were able to fire and therefore detonated outside the blasthole. It was noted that two thirds of the misfires occurred in the top row of blastholes and only a third in the bottom row of holes. As the use of Megadet SD detonators on stoping panels allow a length of loose shocktube to remain hanging from the collar of blastholes even after initiation, it was proposed that the detonation of preceding blastholes in the face may apply sufficient force to the loose length of shocktube to extract the as of yet undetonated SD detonator from within adjacent blastholes prior to detonation.

The use of Megadet Long Period Delay (LPD) detonators in Raises and Advanced Strike Gully rounds (ASG) did not experience the same problem and unfired holes/misfires were seldom encountered despite similar loading practices. Given the difference in results it was postulated that due to the longer 1.5 metre blastholes used in ASGs, detonators experienced more traction within the blasthole due to the longer emulsion column. In addition, as detonating cord was used to initiate the LPD detonators, the length of shocktube that remained hanging from the collar of each blasthole was short as the detonating cord would 'cut' the excess shocktube at the collar of each hole on initiation. This reduced the length of loose shocktube hanging from the blasthole that forces within the blast could act upon.

In order to resolve this problem and prevent the ejection of detonators from blastholes during the blast, development was undertaken with a stemming manufacturer to allow for the use of a stemming plug with both efficient stemming properties and a suitable application method for use with viscous pumpable emulsion explosives. As traditional stemming plugs cannot be tamped against the emulsion column within the blasthole (due to its viscous nature), a pull string was required that would allow the clay composite stemming plug to be locked within the hole without the action of tamping. As the stemming plug is applied soon after the blasthole has been loaded and prior to completion of the emulsion gassing process, the plug is applied inside the collar of the blasthole such that an air gap is available between the stemming and the gassing emulsion for column rise. As the emulsion column rises, air from

within the blasthole is forced through small spaces around the course clay composite stemming plug.



Figure 37, Sequence of photos showing operator inserting a Vala stemming plug while loading a winch bed

As all blasts from early in the project utilised stemming to eliminate the occurrence of misfires/pull outs, the performance of the pumpable emulsion system when compared to existing explosives systems would need to take cognisance of the effects of stemming on results obtained in the blast.

6.1.2 Methods for the Collection of Data and Comparison of Blast Results

Once the initial phases of the project were nearing completion and brief underground tests indicated positive results, the need arose to evaluate the performance of the pumpable emulsion system against existing explosives systems in the underground environment. In order to allow for the comparison, single panel trials were planned on the fore mentioned mining operations. In order to allow for comparison of performance, blast audits were undertaken on both the existing cartridged emulsion explosives and pumpable emulsion systems at the blastface. Blast audits were undertaken through the trial period as the emulsion system was moved between different faces on the operations. One raise line and three stoping panels were trialled over the duration of the Rustenburg Minerals and Rand Uranium Trials.

In order to ensure that relevant data was collected through the course of the trial, a blast audit sheet was introduced for the project as indicated in Appendix 6. In addition to specific data collected for qualitative purposes, pump operators were required to complete blast audit sheets throughout the trial period for quantitative analysis and quality control purposes. As energy content within the blast is of vital importance in the evaluation of blast results, blast design specifications were recorded to allow for the calculation of powder factor and energy factor for the comparison of blasting energies.

As the un-normally high presence of supervision within trial panels often leads to higher quality inputs from mine personnel leading to better blast results (white coat effect), the underground field trials were divided into two broad stages. The first stage of the trial included the presence of technical staff such that the emulsion system could be implemented and optimised within as short a period as possible. Following this period of focussed support, technical personnel withdrew from the trial and operational personnel were left in place to continue with the daily operation of the emulsion system and the daily recording of data. Throughout this period pump operators were required to complete blast audit sheets and record performance measurements at the face. This data was then analysed in an attempt to determine the performance of the system under standard operating practices.

6.1.3 Methods for Quantifying System Advantages

Once results had been obtained for each of the faces, the direct cost of explosives could be compared for the existing cartridged explosive used in the operation and for the implementation of Emulsion B. In addition to the direct cost of explosives, costs incurred through the requirement for charging equipment, logistics and transport infrastructure and storage requirements could also be compared for the two explosives systems. As the fore mentioned factors in turn have an effect on upstream and downstream activities on the mine, a comparison can then be made of the implications of the explosives systems on the broader feasibility of the operation. As the area of greatest importance in daily blasting practices is production of ore for the generation of revenue, the accomplishment of a daily quality blast of full advance and within dimensional compliance will be discussed with relevance to the direct cost of explosives.

Another area where the implementation of the pumpable emulsions could have a significant influence on the daily operation of the mine is through logistics and the daily handling of explosives. For this reason the third phase of the study included an analysis of the current logistics practices for class 1 explosives in order to determine the influence that the class 5.1 system would have on transport and handling requirements of the operation. As much of the focus in the Rustenburg Minerals trial lay in the integration of the cap sensitive emulsion and the new emulsion pump technology, the later section of the study was not included. Also, as the Groenfontein shaft on Rustenburg Minerals operation is a shallow decline shaft that is currently in a redevelopment and expansion phase, little data collected on the present logistics systems would be applicable to production operations. The moderate depth of the Cook 2 shaft at Rand Uranium was considered representative of shallower shafts within the Witwatersrand Basin at approximately 600 metres in depth. For this study, three time studies were undertaken on the handling of explosive through the shaft and the consecutive transportation of this explosive to the working areas. Once an understanding had been ascertained for labour and time requirements for the handling of explosives through existing mine practices, a theoretical time saving and increase in shaft availability could be calculated for the introduction of pumpable emulsions on the shaft. Also included in this phase of the study was the effect of section 54s, overbreak and tramming of class 5.1 blasting intermediates in place of class 1 explosives. As logistics and charging operations determine a significant proportion of explosives related labour requirements within the operation, a considerable saving was considered through a reduction in labour.

6.2 Rustenburg Minerals Field Trial

6.2.1 Rustenburg Minerals Blast Results – Development

Groenfontein shaft is a decline shaft on the outcrop of the LG6 chromite reef in the LCZ of the Critical Zone in the Western Bushveld Complex. Due to the shallow depth of the LG6 reef, partially weathered strata with filling occurs in the hangingwall, often resulting in poor hanging wall conditions. At the outset of the RMDC trial a request was made by the mine that the new pumpable emulsion, Emulsion B, together with the new pumpable emulsion

technology be tested in development prior to implementation within stoping panels such that planned production volumes would not be compromised.

Following the establishment of trial infrastructure on surface at Groenfontein shaft, RMDC, and the training of personnel responsible for the operation of the Portable Pump, the emulsion system was implemented in pre-development within the North 3 raise line. Given the limited exposure of the emulsion system at the outset of the trial, the initial focus during the implementation of the system was on achieving consistent blast results. As the aim of the trial was to compare the performance of the pumpable emulsion system to Cartridge L, the existing cartridge explosive, no initial changes were made to existing mine practices under standard operating conditions. The raise round design and timing sequence as used on the North 3 Raise at RMDC is depicted in Figure 38 while the powder factor and energy factors for the two explosives are listed in Table 12.

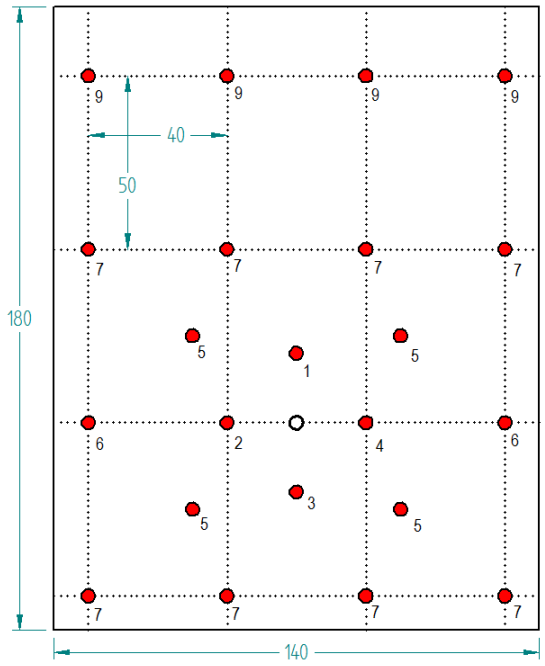


Figure 38, RMDC - Raise Blast Design and Timing Sequence

Table 12, Round Design Parameters for Cartridge L & Emulsion B, Raise N3, RMDC

Rustenburg Minerals	Raise Design	
Explosive	Cartridge L	Emulsion B
RWS	94	92
RBS	135	132
Explosive Density (g/cc)	1.16	1.14
Coupling (%)	85%	100%
Mass per hole (g)	1200	1200
Drilled Length (m)	1.5	1.5
Nominal Hole Diameter (mm)	36 ± 1	36 ± 1
Hole Volume (L)	1.53	1.53
Designed Face Width (m)	1.4	1.4
Designed Face Height (m)	1.8	1.8
Blastholes per Face	22	22
Column Length (%)	80%	69%
Unfilled Length (m)	0.29	0.47
Initiation System	Megadet LPD	Megadet LPD
Rock Density (g/cc)	4.1	4.1
Stemming Material	-	VALA (Bintech)
Powder Factor	7.0	7.0
Energy Factor	6.6	6.4

Appendix 7 is an account of blast results achieved with Emulsion B on the North 3 raise from the outset of blasting operations. These blast results are summarised in Table 13. Over the first four blasts of the trial, modifications were gradually made to the emulsion system to achieve a consistent average blasting density of between 1.12 and 1.15 g/cm^3. In order to check the density of loaded emulsion, quality control was undertaken on a daily basis through cup samples as described in Section 4.1.1 prior to the charging of the blastface. At the outset of the trial, supervision at the blastface was high and blasting practices and results were monitored on a daily basis. In order to allow for a comparison of the results obtained under normal operating conditions, supervision on the raise was gradually reduced until only a single visit was made to the raise on a weekly basis. In order to allow for the collection of data on days when supervision was not present on the face, the BME operator was required to complete the daily blast audit sheet in order to monitor blast results.

In total, 41 blasts were monitored on the North 3 raise line. Of the 41 blasts, 34 were drilled with a hole length of approximately 1.5 metres, while 7 blasts were drilled with a hole length

ranging between 1.2 and 0.8 metres due to time restrictions and general delays within the mining cycle. These short rounds are indicated by italics in Appendix 7. Of the 34 faces drilled with a hole depth of 1.5 metres, 6 blasts were undertaken through incorrect blasting practices or under abnormal operating conditions (highlighted in Appendix 7). Of these six, four faces were fired within 30 minutes of loading the blastface thus allowing insufficient time for Emulsion B to achieve blasting sensitivity, one face was charged under abnormally low air pressure resulting in poor quality mixing of the emulsion and one face was drilled with only 15 blastholes due to substandard drilling practices.

As emulsion within five of these faces would have deflagrated instead of detonated, the advance achieved on these faces was poor. Face advance achieved on correctly blasted 1.5 metre rounds averaged 96% for the duration of the North 3 raise trial, while faces blasted with a drilled length of 0.8 to 1.2 metres achieved an average face advance of 99%. Incorrectly blasted faces as discussed above achieved an average face advance of 76%, reducing the overall average advance achieved over the period of the trial to 93% for the 41 blasts.

Table 13, Summary of blast results obtained on the North 3 Raise from the outset of the trial

Blasting Practice	Blasts	Cumulative Drilled Length (m)	Cumulative Advance (m)	Average Hole Length (m)	Average Advance (m)	Average Advance (%)
1.5m Length Rounds	28	42.5	40.7	1.52	1.45	96%
0.8m to 1.2m Rounds	*7*	*7.9*	*7.8*	*1.13*	*1.11*	*99%*
*Incorrect Blasting Practices	6	9.1	6.9	1.52	1.15	76%
All Blasts	41	59.5	55.4	1.45	1.35	93%

* Incorrect blasting practices are discussed for individual blast results in Appendix 7

Figure 39 depicts the sequence of blast results achieved over the duration of the trial. In order to allow for the comparison of different hole lengths, the advance achieved per blast has been plotted as a percentage of drilled length for each individual blast. The period of density optimisation over the first 4 blasts of the trial is evident at the beginning of the graph. Between blasts 7 and 19 a period of inconsistent results are evident within the trend in Figure 39. Over this period, the poor and inconsistent gassing rate of Emulsion B (still requiring a

minimum of 30 minutes to reach blasting density) coupled with failure of the miner to use a 'stay-a-lite' starter when initiating the face allowed for insufficient time for Emulsion B to reach blasting density. Once 'stay-a-lite' starters became readily available and the sub-standard practice of the miner had been corrected, an immediate improvement in the average length of advance was evident.

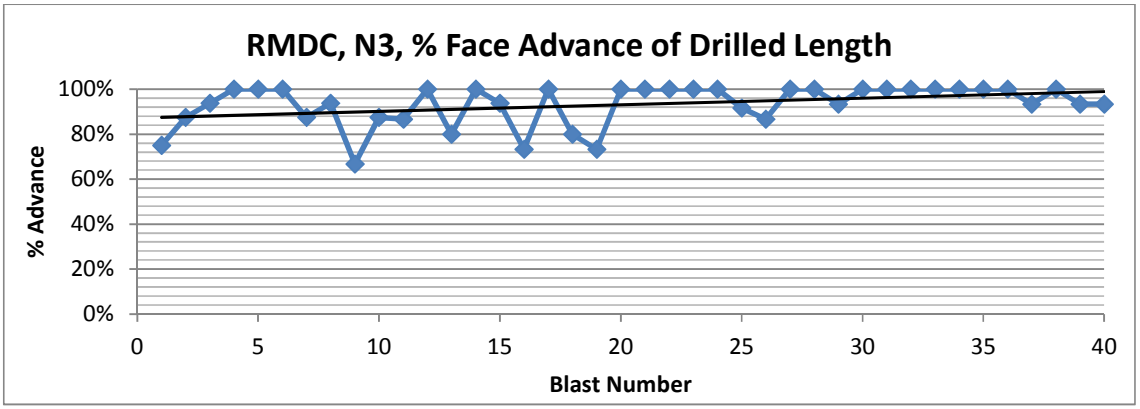


Figure 39, Emulsion B on North 3 raise RMDC – Percentage face advance of drilled length

As it was necessary to stem emulsion blastholes during preliminary trials with the emulsion system, Vala (Bintech) stemming plugs were utilised in all Emulsion B blastholes. This was undertaken to prevent detonators being pulled from the blasthole during initiation of the blast thereby resulting in misfires. A discussion of the need for stemming and the method for its application is discussed in Section 6.1.1 on page 98. As the North 3 raise broke through before comparative blasts could be undertaken with the pre-existing cartridged emulsion system, Cartridge L, a direct comparison of blast results was not possible. Advance rates achieved through the use of cartridged emulsions on parallel raises within the section typically ranged between 1 metre and 1.5 metres in length. Through the comparison of the socket lengths evident on blast faces within the section, the average socket length of approximately 30cm indicated an average advance of approximately 1.2 metres for equivalent 1.5 metre rounds. It was noted during the comparison that standard faces within the section appeared to have less accurate drilling when compared to the trial raise and despite the presence of stemming within the Cartridge L box charging personnel failed to stem blastholes.

6.2.2 Rustenburg Minerals Blast Results - Stoping

Following the success of the pumpable emulsion system on the N3 raise, a decision was taken to move the system to panel 3 South in order to test the performance of the system in daily blasting activities on stoping operations. At the outset of the stoping trial an immediate difference in temperature became apparent between the North 3 raise and the South 3 panel influencing the length of time required for Emulsion B cup samples to reach blasting density on the panel. As the North 3 raise had poor air flow the air temperature within the raise was high, while air and rock temperature within the shallow South 3 panel was cool due to the presence of water and good ventilation.

Initial blast results achieved on the South 3 panel were poor due to the slow gassing rate and the inability to achieve blasting density in the required time despite the use of a stay-a-lite. At this point the South 3 panel trial was postponed and blasting continued on raises and ASG's while controlled tests were undertaken on the emulsion B formulation to obtain a better understanding of the effects of temperature on the gassing rate of Emulsion B in varying operating and environmental conditions. A brief discussion of the test procedure followed during these tests is described in Section 4.1.1. Following a delay in the trial to allow for modifications to the Portable Pump and Emulsion B formulation the emulsion system was reintroduced to the South 3 panel. Figure 40 depicts the round design used on the South 3 panel while Table 14 compares the energy within the round design for Emulsion B and the pre-existing cartridged emulsion used on the panel, Cartridge L.

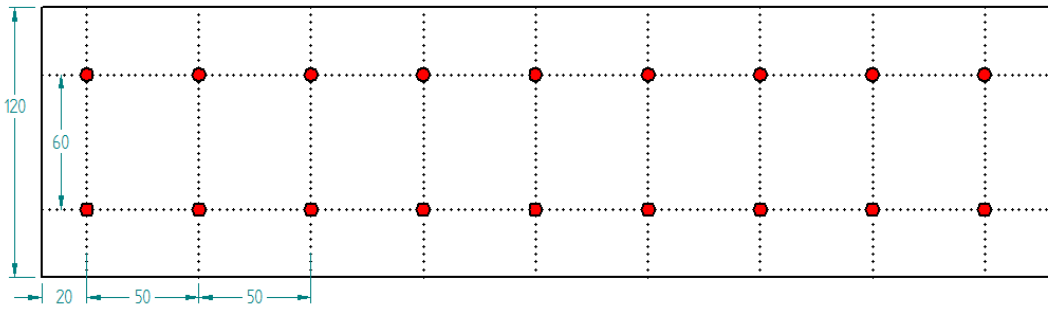


Figure 40, Round Design – Panel 3 South, RMDC

Table 14, Round Design Parameters - Panel 3 South, RMDC

Rustenburg Minerals	Stoping Panel 3 South	
Round Design	Cartridge L	Emulsion B
RWS	94	92
RBS	135	132
Explosive Density (g/cc)	1.16	1.14
Coupling (%)	85%	100%
Mass per hole (g)	1000g	800g
Drilling Angle (degrees)	90˚	90˚
Drilled Length (m)	1.2	1.2
Nominal Hole Diameter (mm)	36 ± 1	36 ± 1
Hole Volume (L)	1.22	1.22
Burden (m)	0.5	0.5
Spacing (m)	0.6 to 0.7	0.6 to 0.7
Design Stoping Width (m)	1.2	1.2
Column Length (%)	84%	57%
Unfilled Length (m)	0.19	0.51
Initiation System	Megadet SD	Megadet SD
Rock Density (g/cc)	4.1	4.1
Stemming Material	-	VALA (Bintech)
Drilling Pattern	Square	Square
Holes / line	2	2
Powder Factor	2.38	1.90
Energy Factor	2.24	1.75

On recommencement of the South 3 panel trial, it became apparent that while the problem of detonators pulling from blastholes had been largely resolved through the use of stemming plugs, it was still possible to experience a failure rate of up to 5% within the panel. These holes were evident as short round sockets in the blastface containing remnants of the sensitised emulsion column. As the use of 12 gram boosters had previously failed to resolve the problem, it was postulated that where discontinuities and weakness planes within the rock mass allowed for back break and spawling of the blastface, shocktube detonators were being pulled from blastholes through the detonation of preceding blastholes. In order to find a temporary solution to the problem the Megadet SD shocktube detonator was inserted into and looped around a single emulsion cartridge that was then used to prime the base of each blasthole. As the emulsion cartridges had an individual mass of 200 grams, the Portable Pump was set to deliver 600grams of emulsion in place of the original 800 gram charge. During daily operations drill operators changed drill bits at regular intervals and blasthole

diameter seldom deviated far enough to allow for a definable difference in measurement through the use of a clinorule. While a standard deviation 0.5mm from the nominal blasthole size was common, no measurements above 37mm or below 35mm were seen on the panel. As three drill operators drilled the panel daily and hole size continuously drifted over a small range, no noticeable average for oversize or undersize blasthole diameter could be accurately defined in each blast. Blast results obtained on the South 3 panel are tabulated in Appendix 8, summarised in Table 15 and plotted in Figure 41.

Table 15, Blast Results, RMDC Panel 3 South

Explosive	Number of Blasts	Cumulative Metres Drilled (m)	Cumulative Advance (m)	Average Hole Length (m)	Average Advance (m)	Average Advance (%)
Cartridge L	10	10.7	9.8	1.07	0.98	92%
Emulsion B - No Primer	14	15.9	13.9	1.14	0.99	87%
Emulsion B - Cartridge L Primer	7	7.2	7.2	1.03	1.03	100%

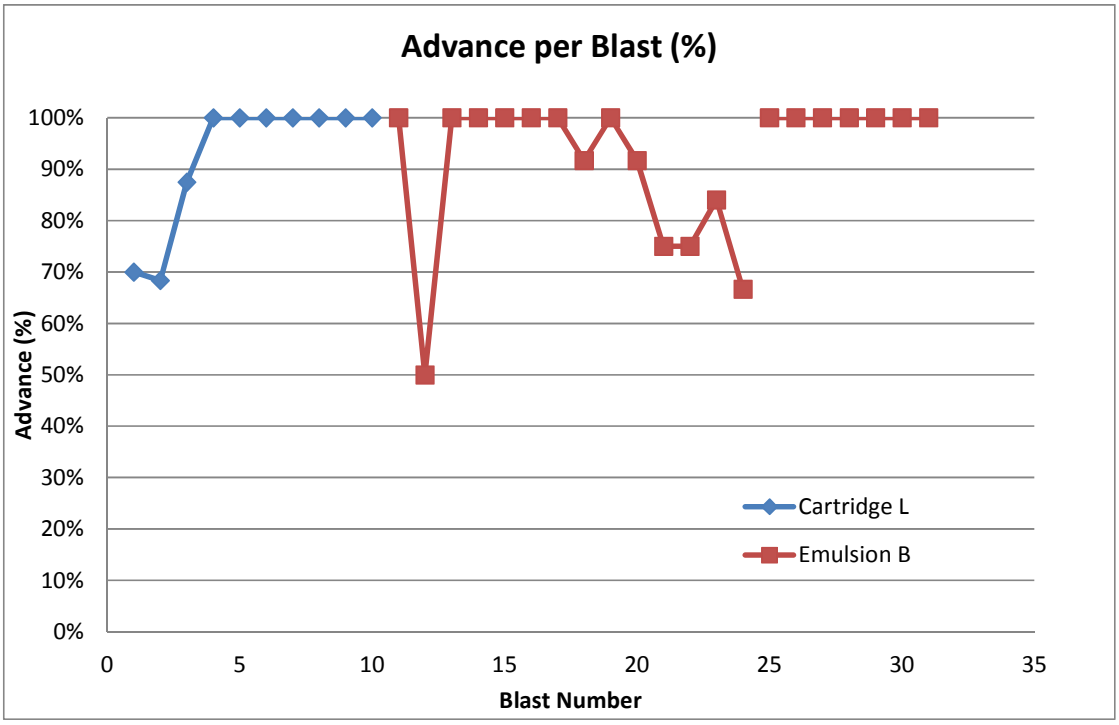


Figure 41, Face advance per blast as a percentage of drilled length, Panel 3 South

Prior to the implementation of the pumpable emulsion system on the panel, 10 blasts were monitored with the existing cartridged emulsion system (Cartridge L). Over this period, an average advance of 92% was achieved on the panel. Following the introduction of Emulsion B 21 blasts were monitored on panel S3. Of these, 14 blasts were undertaken prior to the implementation of the single Cartridge L primer at the base of each blasthole, and 7 blasts were undertaken following the implementation of the Cartridge L primer. Over the 14 blasts undertaken prior to the implementation of the cartridge primer, an average advance of 87% was achieved for the panel. During this period, the profile of the face began in a moderate condition and slowly deteriorated as blastholes that failed to fire due to pull-outs resulted in small to medium sized humps in the face. Following the implementation of the cartridged primer at the base of each hole, an advance of 100% was achieved in all 7 blasts.

Figure 41 indicates the sequence of blast results obtained for Cartridge L and Emulsion B on the S3 panel. During initial blasts with Emulsion B, full advance was achieved on the panel. It was however noted at this point, that up to 5% of blastholes on the panel misfired in the blast due to 'pull-outs'. As initial blasts on the panel were undertaken in LG6 chromite reef, the strength of the rock was comparatively low and the misfired blastholes were broken to approximately 5 cm in length by sequential blastholes breaking the double burden of 80 cm.

During the third week of the Emulsion B trial however, the LG6 chromite reef was lost and the strength of the rock increased as did the number and size of discontinuities within the face. Between the 8th and 14th blasts with Emulsion B, the number of blastholes that failed to detonate in the panel became more common and the sockets left in the blastface increased in length. Following the 14th Emulsion B blast where an advance of only 67% was achieved, the panel reverted to the use of Cartridge L for one week until a decision was taken to implement a single Cartridge L primer at the base of each blasthole. Following the implementation of the Cartridge L primer at the toe of each blasthole no further pull-out/misfires were observed on the face and Emulsion B achieved 100% advance in all subsequent blasts.

6.3 Rand Uranium Field Trial

6.3.1 Rand Uranium Blast Results – UE1A Stope

At the outset of the Rand Uranium emulsion trial, panel North 6 (N6) on 85 level of Cook 2 shaft was selected for the comparison of Emulsion B and the pre-existing cartridged emulsion, Cartridge H. The UE1A reef mined within the N6 panel lies at the base of the Gemsbokfontein Member at the base of the Elsburg Formation and at a depth of approximately 450 metres. Within the N6 panel, the channel width of the UE1A reef varied from 1cm to 10cm in height and was bounded in both the hangingwall and footwall by competent and massive siliceous quartzite (Stratigraphy column of the Turffontein Subgroup (modified after JCI Internal Reports, 2012) . Given the high strength of the quartzite and the comparatively low stress acting on the rock mass due to the shallow depth of the operation (Rorke 2012), high strength Cartridge H emulsion explosives were ordered by miners for production blasting. Panel N6 was a newly restarted short panel within the section and was selected by mine management in order for the emulsion system to be tested prior to its implementation on larger production panels. Figure 42 depicts the round design drilled on the N6 panel while Table 16 compares the design parameters for Emulsion B and the pre-existing cartridge emulsion, Cartridge H.

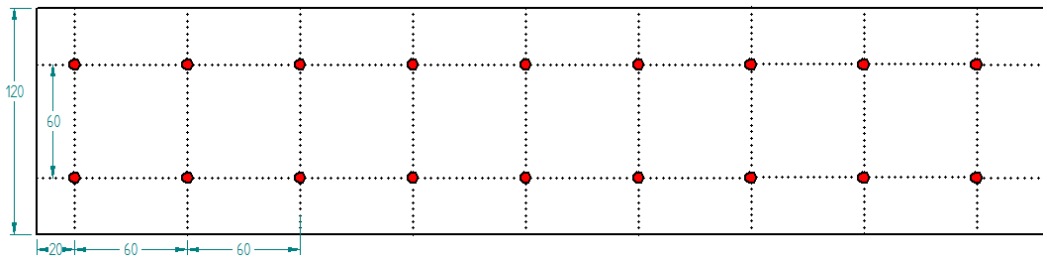


Figure 42, Round Design – Panel North 6, UE1A reef, Rand Uranium

Table 16, Round Design Parameters – UE1A Panel

Rand Uranium	85 Level UE1A Stope Design	
Round Design	Cartridge H	Emulsion B
RWS	109	92
RBS	159	131
Explosive Density (g/cc)	1.16	1.14
Coupling (%)	85%	100%
Mass per hole (g)	1200g	1000g
Drilling Angle (degrees)	70˚	70˚
Drilled Length (m)	1.2	1.2
Nominal Hole Diameter (mm)	36 ± 0.5	36 ± 0.5
Hole Volume (L)	1.22	1.22
Burden (m)	0.6	0.6
Spacing (m)	0.6	0.6
Design Stoping Width (m)	1.2	1.2
Column Length (%)	99%	72%
Unfilled Length (m)	0	0.34
Initiation System	Megadet SD	Megadet SD
Rock Density (g/cc)	2.7	2.7
Stemming Material	Clay / -	VALA (Bintech)
Drilling Pattern	Square	Square
Holes / line	2	2
Blastholes / metre	3.3	3.3
Powder Factor	2.8	2.3
Energy Factor	3.0	2.1

During the initial trial period, drilling quality on the panel was sub-standard and despite a prescribed burden of 60cm, burdens typically ranged from 55cm to 1 metres while drilling direction ranged from 50 to 100 degrees. The poor distribution of energy within the face as a result of this inaccuracy coupled with the high strength of the rock mass resulted in poor and inconsistent face advance for both Cartridge H and Emulsion B. While correctly sized burdens would advance, larger burdens prevented the breakout of the adjacent blastholes. Given the high variation in drilling direction experienced throughout the trial, toe burdens experienced an even greater variation in size significantly reducing the possibility of advance. While this trend was experienced with both cartridged and pumpable emulsion explosives, blasts undertaken with Emulsion B also experienced misfires as a result of detonators pulling from blastholes prior to the implementation of a Cartridge H primer at the base of the blasthole. As the burden left in place by a misfired hole could be greater than 1.2 metres,

significant humps were left in the face profile due to the strength of the massive quartzite. As a result of this inconsistency, measurements for face advance differed according to where they were taken on the face and were highly erratic and often not representative of the actual results achieved in the blast.

In addition to the challenges encountered in daily blasting practices was the repetitive refusal from the miner and team to use the pumpable emulsion system. Poor mine supervision on the panel allowed the miner to change back to the use of Cartridge H after each alternate blast until an instruction could be reissued on surface the following morning for the team to use Emulsion B. This resistance to change resulted in Cartridge H and Emulsion B being used on an alternating basis contaminating the data that was collected.

In order to allow for a comparison of the two explosives systems it was first necessary to correct the quality of blasting practices on the blastface so that meaningful results could be obtained from the trial. As blasts alternated between the Cartridge H and Emulsion B the corrections applied to blasting practices were applicable equally to both explosives systems. For the second half of the trial however, the majority of Cartridge H blasts were recorded first, followed by the results for Emulsion B. As training of personnel continued over this period and blast results continued to improve, it is felt that the results obtained in the second half of the trial are biased towards Emulsion B. The gradual increase in advance per blast is shown in Figure 43. Given the inconsistency in blast results in the first half of the trial it became necessary to average the advance achieved across the panel in each of the first 4 Cartridge H blasts and each of the first 5 Emulsion B blasts monitored. For the average the monitored length of sockets measured across the blastface was averaged to achieve an average advance for the face. Table 17 below indicates the average advance achieved for Cartridge H and Emulsion B for the initial averaged face advance and the average advance measured over the second half of the trial period.

Table 17, Blast Results - UE1A N6 Panel

Explosive	Number of Blasts	Cumulative Drilled Length (m)	Cumulative Advance (m)	Average Drilled Length (m)	Average Advance (m)	Average Advance (%)
Cartridge H (Average Face Profile)	4	4.5	2.5	1.13	0.64	56%
Emulsion B (Average Face Profile)	5	5.9	3.1	1.18	0.63	54%
Cartridge H	4	4.5	3.2	1.13	0.80	71%
Emulsion B	6	6.8	5.3	1.13	0.88	78%

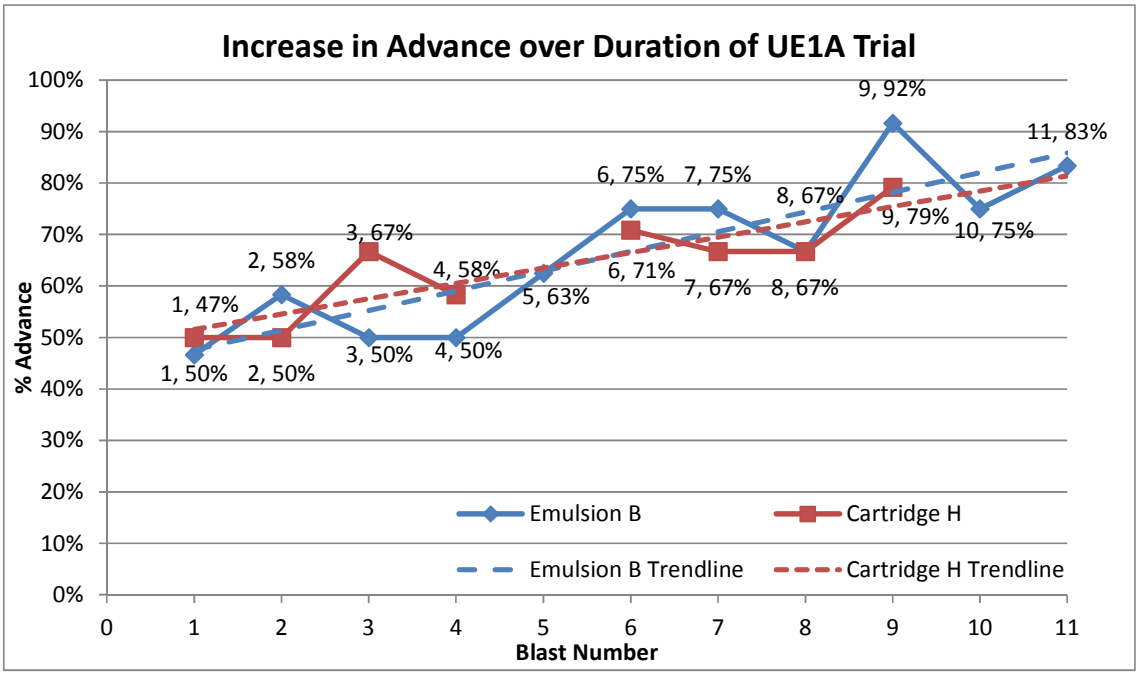


Figure 43, Increase in Advance over Duration of Trial

At the outset of the trial, the pumpable emulsion system was implemented with the use of a 12 gram booster at the base of the blasthole for reliable initiation of Emulsion B. During the first 5 blasts undertaken with Emulsion B on the UE1A panel it became evident that the use of the 12 gram booster and Vala stemming plug were not able to prevent detonators being pulled form blastholes thus resulting in misfires. Due to the strength of the massive quartzite in the hangingwall and footwall, oversize burdens caused by misfires would result in the formation of steps or humps in the face that would eliminate the free breaking point for the remainder of the panel. For these reasons the advance in the first 5 blasts undertaken with

Emulsion B achieved only 53%. Cartridge H blast results over this initial period indicated only marginally better results as a result of poor drilling direction and oversize burdens resulting in an averaged advance of only 56%.



Figure 44, Poor face profile and advance resulting from a pull-out/misfire in the fifth Emulsion B blast.

Following the implementation of a Cartridge H primer in Emulsion B blasts to which the shocktube could be tied, no further misfires were evident in the blast results and a significant improvement was noted in face profile. On-going correction of drilling practices also allowed for consistently better blast results on the panel in comparison to the start of the trial. Over the second half of the trial the four blasts were taken with Cartridge H achieved and average advance of 71% while the final six blasts on the panel undertaken with Emulsion B indicated an average advance of 78%. As stemming was used in conjunction with Emulsion B, a mass of 1 kilogram of emulsion was charged per hole (including the mass of the 200 gram primer). Despite efforts to reduce the mass of explosive per hole, mine personnel continued to load Cartridge H blastholes to the collar in order to maximise energy in an attempt to increase advance. Accordingly, 6 cartridges or 1.2 kg of explosive were loaded per blasthole.

6.3.2 Rand Uranium Blast Results – K9 Stope

Following the closure of the N6 panel on the UE1A reef, the pumpable emulsion system was moved to panel 2 West (2W) on the K9 reef, 128 level. The K9 reef is a thick conglomerate at the base of the Vlakfontein Member of the Kimberly Formation (Stratigraphy column of the Turffontein Subgroup (modified after JCI Internal Reports), 2012). While only a small number of blasts were undertaken on the panel, the results obtained from the K9 reef provide an interesting contrast to those experienced within the UE1A reef. The round design implemented on panel 2W on the K9 reef is illustrated in Figure 45 while Table 18 provides a comparison of energy within the initial round design for Cartridge H and Emulsion B.

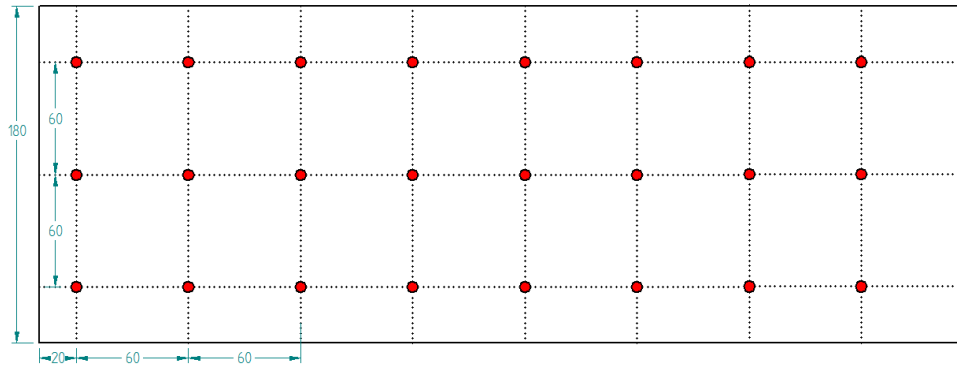


Figure 45, Round Design – Panel 2 West, K9 reef, Rand Uranium

Table 18, Round Design Parameters – Panel 2W, K9 reef, Rand Uranium

Rand Uranium	128 Level K9 Stope Design	
Round Design	Cartridge H	Emulsion B
RWS	109	92
RBS	159	131
Explosive Density (g/cc)	1.17	1.14
Coupling (%)	94%	100%
Mass per hole (g)	1000g	800g
Drilling Angle (degrees)	90˚	90˚
Drilled Length (m)	1.2	1.2
Nominal Hole Diameter (mm)	36	36
Hole Volume (L)	1.22	1.22
Burden (m)	0.6	0.6
Spacing (m) Top to Bottom	1.2	1.2
Design Stoping Width (m)	1.8	1.8
Column Length (%)	74%	57%
Unfilled Length (m)	0.31	0.51
Initiation System	Megadet SD	Megadet SD
Rock Density (g/cc)	2.7	2.7
Stemming Material	-	VALA (Bintech)
Drilling Pattern	Square	Square
Holes / line	3	3
Blastholes / metre	5.0	5.0
Powder Factor	2.3	1.9
Energy Factor	2.5	1.7

Table 19, Blast Results – K9 Panel

Explosive	Number of Blasts	Cumulative Drilled Length (m)	Cumulative Advance (m)	Average Drilled Length (m)	Average Advance (m)	Average Advance (%)
Cartridge H - Initial	5	5.7	5.4	1.14	1.08	95%
Emulsion B - Initial	2	2.4	2.4	1.20	1.20	100%
Emulsion B - Modified	3	3.5	3.5	1.17	1.17	100%
Cartridge H - Modified	5	5.9	5.9	1.18	1.18	100%

Blast results achieved on the 2W panel are listed in Appendix 10 and summarised in Table 19. At the outset of the blasting operations on the panel, the quality of drilling on the panel was good and an average burden size of 64cm was drilled on the panel compared to a design burden of 60cm. As the stoping width on the panel was 1.8 metres in height, three rows of

holes were drilled in the face in a box pattern with an average spacing of 1.05 metres between the top and bottom row of holes. Blast audits undertaken on the last five Cartridge H blasts prior to the change of explosives achieved an average of 95% advance for the panel. Following two blasts with Emulsion B on the 2W panel, a high level visit by mine management corrected what was perceived to be poor marking on the face. The corrected round design recommended a reduction in the burden to 50cm in length and a spacing of 1.2 metres between the top and bottom rows of holes. Three Emulsion B blasts were undertaken on the panel following the change in drilling practices. Both Emulsion B blasts undertaken prior to the change in design and all three blasts undertaken with Emulsion B following the change achieved 100% advance. As available Emulsion B blasting agent had been exhausted at this point in the trial, operators returned to the use of Cartridge H. Five blasts were monitored on the panel following the return to cartridged emulsion explosives. All five of these blasts achieved 100% advance.

6.3.3 Determination of Coupling Ratio

As discussed in section 2.3.3 of the literature review, the coupling ratio of an explosive is a significant consideration when comparing the performance and efficiency of explosives. In a blast, the energy available at the bottom of a blasthole for the breakout of the toe is proportional to the coupling ratio and the density of the explosive. Further, as the amplitude of the strain wave in the rock mass (generated by a fixed mass of explosive) is directly proportional to the coupling ratio, the coupling ratio plays a significant role in the shock wave transmitted to the rock mass. From this it can be seen that the higher the coupling ratio of an explosive, the greater the efficiency with which it will act on the rock mass.

In order for comparison of the RU and RMDC blast results, the coupling ratio for Cartridge L and Cartridge H needed to be determined. As the properties of the cartridged products are similar and both operations used 29 x 270mm emulsion cartridges in 36mm blastholes, coupling tests were undertaken on RU that were applicable to both operations. In order to determine the coupling ratio, the depth and diameter of blastholes was measured before individually identified blastholes were loaded by the team. Once the number of cartridges loaded had been counted and the length of the uncharged collar of the hole measured, the average compressed length of each cartridge within the blasthole could be calculated and the effective charge diameter calculated. This could then be used for the calculation of coupling

ratio within the blasthole. Measurements taken for the diameter of blastholes were consistent in size and no measureable difference in diameter could be determined through the use of a measuring tape or clinorule. In order to allow for a comparison of coupling ratio within varying hole sizes one set of coupling tests were also undertaken on 1 East panel on the K9 reef in which 32mm blastholes were drilled. Measurements used for the determination of the coupling ratio are listed in Appendix 11 and are summarised in Table 20.

Table 20, Coupling Ratio for 29 x 270mm Cartridge H in 36mm and 32mm Blastholes

Hole Diameter (mm)	36mm	32mm
Average Coupling (%)	84.8%	93.8%
Maximum Coupling (%)	92.9%	94.0%
Minimum coupling (%)	77.0%	93.2%
Standard Deviation (%)	3.8%	0.4%

From the coupling results it can be seen that the use of 29mm x 270mm Cartridge H explosives in 36mm blastholes resulted in an average coupling ratio of 84.8% with a standard deviation of 3.8%. In the single set of coupling tests undertaken in 32mm blastholes a significantly better coupling ratio of 93.8% was achieved with a standard deviation of only 0.36%. While limited results are available for the 32mm blasthole this supports the recommendation that greatly improved coupling ratio can be achieved if the initial cartridge diameter better matches the size of the blasthole.

6.4 Discussion of Blast Results

6.4.1 Explosive Performance and Blasting Efficiency

Table 21 and Table 22 compare the blast results obtained from underground trials on RMDC and RU respectively. It is evident from the results that there has been a general improvement in the average advance achieved per blast despite the decrease in energy factor in the round design. As the mass delivered to into each blasthole with the Portable pump was set and controlled by the BME operator, a decrease in powder factor of approximately 20% was

experienced in all blasts except the RMDC raise line, largely due to the prevention of overcharging of blastholes. In addition to the reduction in explosives mass, the lower RWS of Emulsion B when compared to Cartridges L and H facilitated a further reduction in energy factor for all round designs. These factors resulted in a decrease in the energy factor of RMDC blasts with a reduction of 3% for N3 raise and 22% for the Panel 3. However, due to the increase in advance achieved in these two faces, the total increase in blast efficiency for RMDC increased by 15% for the N3 raise and 27% for the S3 panel.

Table 21, Comparison of Blast Results, RMDC

Workplace	RMDC N3 Raise		RMDC S3 Panel	
Explosive	Emulsion B	Cartridge L	Emulsion B	Cartridge L
Average Advance (%)	93%	80%	100%	92%
Powder Factor	7.0	7.0	1.9	2.4
Energy Factor	6.4	6.6	1.8	2.2
Influence of Emulsion B on Blasting Efficiency				
Increase in Advance (%)	14%		8%	
Reduction in Powder Factor (%)	0%		20%	
Reduction in Energy Factor (%)	3%		22%	
Total Increase in Blast Efficiency (%)	15%		27%	

Due to overcharging practices on all faces on Rand Uranium, the Powder Factor experienced through the use of Emulsion B decreased by between 18% and 23% on all faces. In addition, as the RWS of Cartridge H used on RU was comparatively greater than the RWS of Cartridge L used on RMDC, the energy factor further reduced through the use of Emulsion B to between 30% and 33% for the various stoping panels. Due to this reduction in energy and the marginal increase in advance experienced in stoping operations, the overall increase in blast efficiency increased by of 37% for the K9 - 2W panel and 36% on the UE1A - 6N panel.

Table 22, Comparison of Blast Results, RU

Workplace	RU UE1A, 6N Panel		RU K9, 2W Panel Initial		RU K9, 2W Panel Modified	
Explosive	Emulsion B	Cartridge H	Emulsion B	Cartridge H	Emulsion B	Cartridge H
Average Advance (%)	78%	71%	100%	95%	100%	100%
Powder Factor	2.3	2.8	1.7	2.2	2.0	2.5
Energy Factor	2.1	3.0	1.6	2.4	1.9	2.8
Influence of Emulsion B on Blasting Efficiency						
Increase in Advance (%)	9%		5%		0%	
Reduction in Powder Factor (%)	18%		23%		20%	
Reduction in Energy Factor (%)	30%		33%		32%	
Total Increase in Blast Efficiency (%)	36%		37%		32%	

Figure 46 plots the energy factor achieved through the RU trial against the advance achieved in the first half of the K9 trial and second half of the UE1A trials. Despite the limited number of samples available a decrease in advance is evident with decreasing energy factor for both faces. Also evident on the graph is the lower energy factor required for advance through the use of Emulsion B than Cartridge H on the UE1A and K9 reefs.

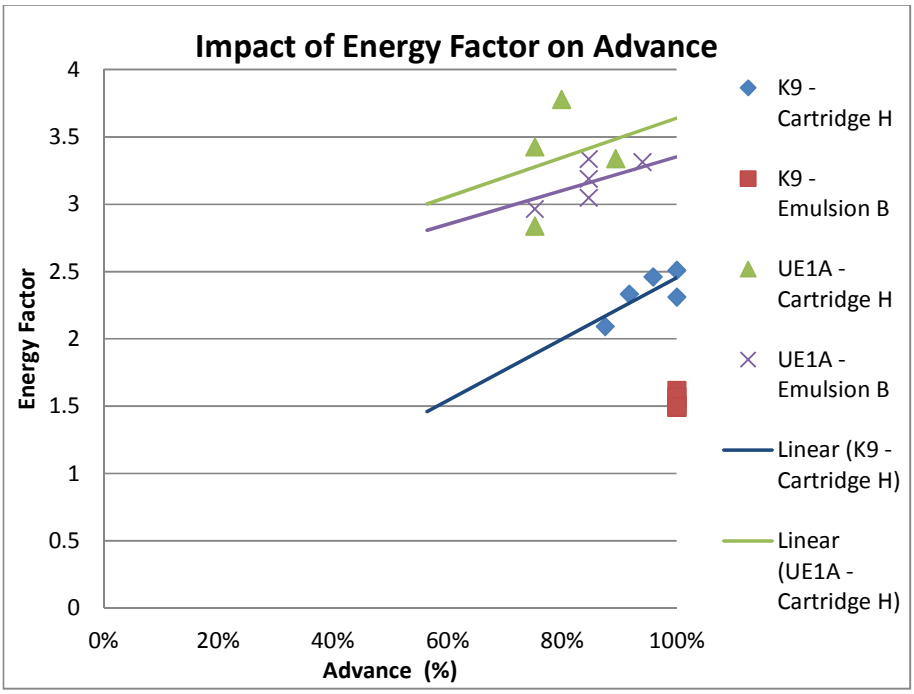


Figure 46, Comparison of energy factor and advance for Emulsion B and Cartridge H, RU

Despite the significant reduction in energy factor achieved through the use of Emulsion B, it is difficult to define a precise increase in blasting efficiency due to the number of variables present within field trials. Practices on the North 3 raise on RMDC indicated higher quality drilling than similar raises due to the increased level of supervision during the initial phase of the trial, while standard practices on Panel South 3 included the overcharging of blastholes. While the additional energy in the collar of the blast will have increased the Powder Factor of the blast it is unable to assist in advance thereby reducing apparent blasting efficiency. Varying rock mass conditions in the LG6 chromite reef also had a significant impact on the ability of stemming to hold the detonator within the blasthole and periods of high and low advance were dependent to a large degree on the condition of the rock mass prior to the introduction of the cartridge primer.

Rand Uranium presented similar challenges to the comparison of blast results. Poor and inconsistent drilling practices over the UE1A panel together with the overcharging of blastholes distorted the comparison of energies and blast results obtained on the panel. Teething problems encountered through the use of Emulsion B in short blastholes also presented a problem for the comparison of results in both the RMDC and RU trials. Initial attempts to prevent the detonator being pulled from the blasthole required the implementation of stemming within the collar of the blasthole. This was later followed by the introduction of a cartridged emulsion primer at the base of each blasthole. As it was demonstrated through the Losberg VOD test that the use of a cartridge primer had no beneficial influence on the VOD or reliability of Emulsion B, the improvement in advance indicated through the use of a cartridge primer can be attributed to the prevention of detonators being pulled from the blastholes. While these practices prevented the occurrence of misfires, they introduced a range of variables that impacted the efficiency of blasting practices thus preventing an equal comparison of the performance of Emulsion B and existing cartridged explosives.

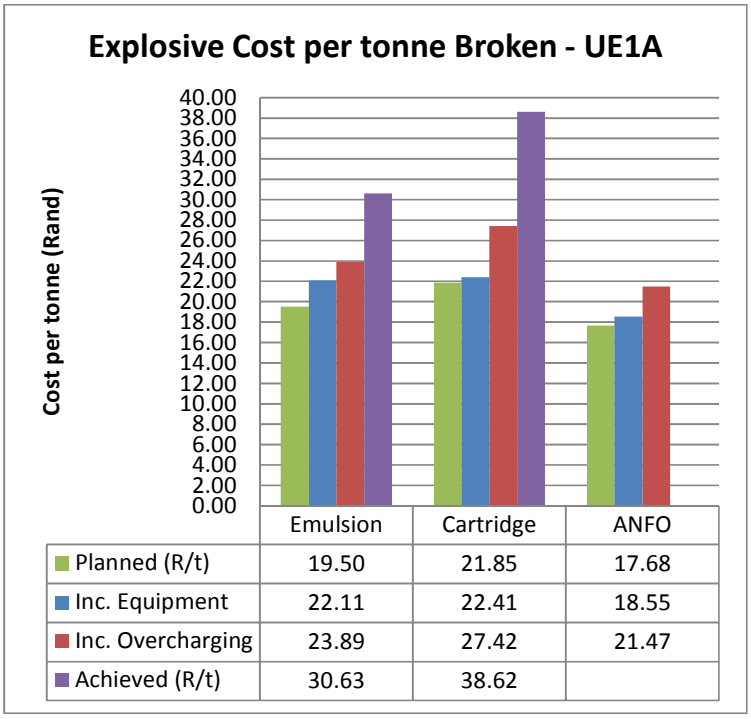
7 Financial Comparison of Explosives Systems

7.1 Comparison of Explosives Costs

In order to determine the direct cost of explosives for each blast and the total cost per tonne blasted for each of the blasting practices experienced through the trial period, an estimate of market related prices were determined for each of the products used in the RMDC and RU trials. While ANFO was not encountered on either of the operations on which trials took place, an estimated costing for ANFO is also included in order to allow for comparison with cartridged and pumpable emulsion explosives. Through the development of a detonator clip following the Rand Uranium trial it was possible to again eliminate the use of the cartridge emulsion primer from the base of the blasthole while maintaining optimal blast results. For this reason the following calculations are made with a single plastic detonator clip at the base of each blasthole and without the use of a cartridged emulsion primer. Through calculations for each blasting practice as presented in Appendix 12, Appendix 13, Appendix 14 and Appendix 15 a comparison of costs for each blasting practice is possible. Through these calculations the ideal direct cost of explosives, the costs for equipment and logistics, the cost of waste experienced in trials and the achieved advance are compounded allowing for the comparison of cost per tonne broken.

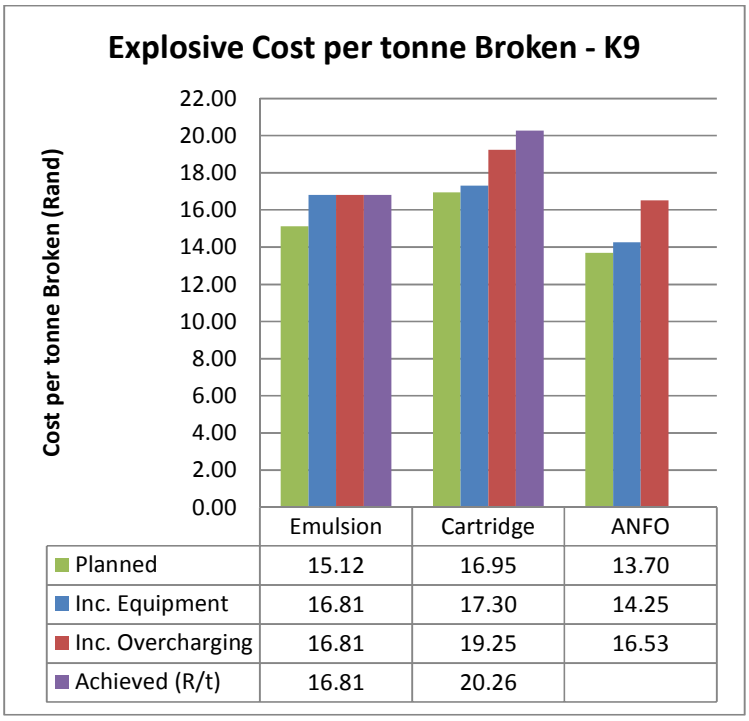
Table 23 and Table 24 depict the cost per tonne broken for the UE1A panel and K9 panels on RU while Table 25 and Table 26 depict the cost per tonne broken for the North 3 raise line and South 3 panel on RMDC respectively. From Table 23 it is evident that while the direct cost of explosives through the use of cartridges on the panel was planned at R21.85 per tonne, the achieved cost of explosives per tonne broken (including logistics cost of R0.56 per tonne) was approximately 75% over budget at R38.62 per tonne largely due to the practice of overcharging and poor advance achieved on the panel. Through the replacement of Cartridge H with Emulsion B, the planned cost of the explosives reduced to R19.50 per tonne while the cost per tonne achieved through the use of emulsion was R30.63 per tonne, 20% below the cost per tonne for the use of Cartridge H. Similarly, the planned cost of explosives per tonne broken reduced from R16.95 with the use of Cartridge H to a planned cost of R15.12 through the use of Emulsion B. As a result of the 5% increase in advance and the 20% reduction in powder factor through the use of Emulsion B the cost per tonne decreased by 17% from R20.26 per tonne for cartridges to R16.81 per tonne for Emulsion B.

Table 23, Explosive Cost per Tonne, RU – UE1A Panel & Gully



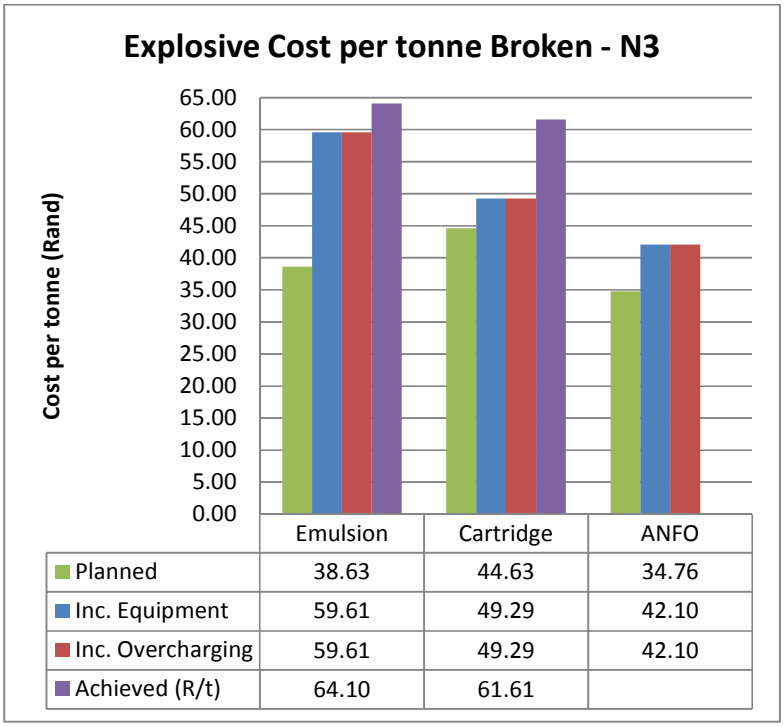
	Emulsion	Cartridge	ANFO
Planned (R/t)	19.50	21.85	17.68
Inc. Equipment	22.11	22.41	18.55
Inc. Overcharging	23.89	27.42	21.47
Achieved (R/t)	30.63	38.62	

Table 24, Explosive Cost per Tonne, RU – K9 Panel



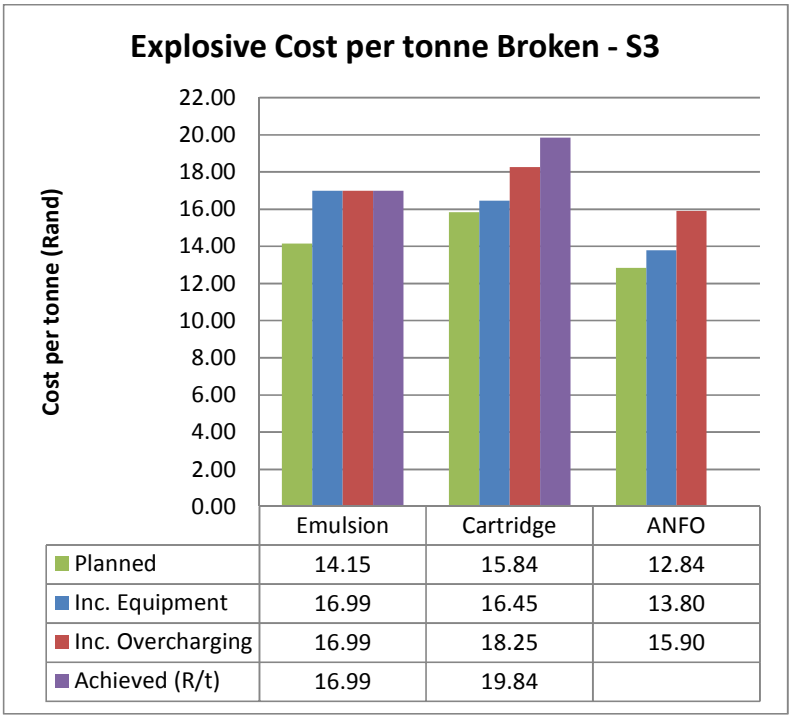
	Emulsion	Cartridge	ANFO
Planned	15.12	16.95	13.70
Inc. Equipment	16.81	17.30	14.25
Inc. Overcharging	16.81	19.25	16.53
Achieved (R/t)	16.81	20.26	

Table 25, Explosive Cost per tonne Broken, RMDC – N3 Raise



	Emulsion	Cartridge	ANFO
Planned	38.63	44.63	34.76
Inc. Equipment	59.61	49.29	42.10
Inc. Overcharging	59.61	49.29	42.10
Achieved (R/t)	64.10	61.61	

Table 26, Explosive Cost per Tonne, RMDC – S3 Panel



	Emulsion	Cartridge	ANFO
Planned	14.15	15.84	12.84
Inc. Equipment	16.99	16.45	13.80
Inc. Overcharging	16.99	18.25	15.90
Achieved (R/t)	16.99	19.84	

Table 25 depicts the cost per tonne broken for the North Raise Line on RMDC while Table 26 compares the cost per tonne for the South 3 panel. Though the reduction in cost per tonne achieved through the implementation of Emulsion B on the South 3 panel was similar to the reduction achieved on the panels on RU, 14%, the cost per tonne achieved through the use of Emulsion B in the North 3 raise on RMDC increased by 4% above the cost of Cartridge L. While blast results on the raise were good, the low mass of Emulsion B pumped through the Portable Pump on a monthly basis allowed for poor utilisation of the charging equipment. Due to the comparatively high monthly cost of equipment required for the implementation of pumpable emulsions and the poor utilisation of the system, the cost per tonne broken on the North 3 raise increased by R20.98 per tonne as a direct result of equipment costs. This is evident in the large increase in cost per tonne due to equipment costs in Table 25.

As discussed in Section 4.3, an understanding of the effect of equipment utilisation is important to the discussion of pumpable emulsion systems due to the elevated fixed costs associated with the operation and maintenance of charging units and logistics systems. Though the fixed costs associated with the use of the Portable pump are significantly lower than traditional emulsion charging equipment, as indicated in Figure 27, the utilisation of the pump is still an important consideration in the cost of the explosives system. Figure 47 depicts the sensitivity analysis for the use of Emulsion B and the Portable pump on the UE1A panel on Rand Uranium. As approximately 1.6 tonnes of Emulsion B were pumped through the Portable pump per month, the fixed cost of equipment when divided over the mass of explosive pumped (utilisation) totalled R3.12 per kilogram of emulsion used on the panel in addition to the basic cost of emulsion. Due to the volume of emulsion, the proportion of equipment costs of the total explosives system costs per month was small and therefore less significant than the mass of explosives utilised and the number of blastholes within the face.

In comparison to the UE1A panel, Figure 48 compares the sensitivity of blasting costs on the North 3 raise on RMDC to fluctuations in costs and blasting practices. As fixed equipment expenses totalled a large percentage of costs on the raise due to the low mass of Emulsion B planned for the month at only 440kg, fluctuations in equipment costs are of greater importance to blasting costs than both the volume and cost of Emulsion B and the direct cost of explosives as a result of the number of blastholes in the face.

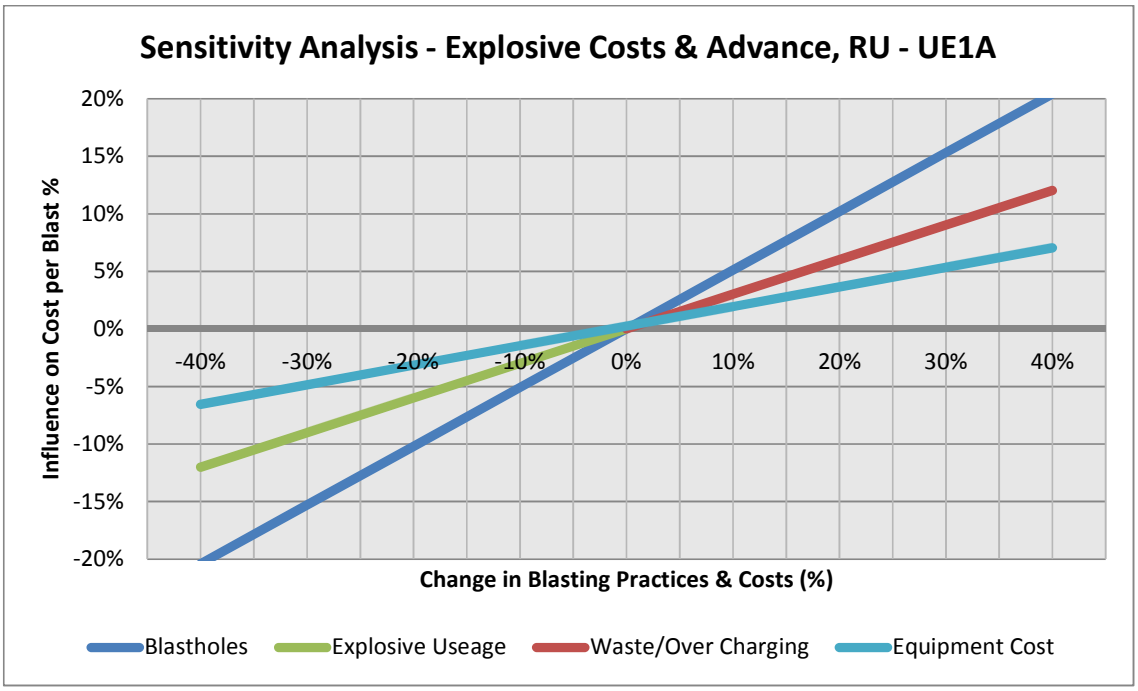


Figure 47, Sensitivity Analysis – Influence of blasting practices and variables on the cost per blast, RU - UE1A panel

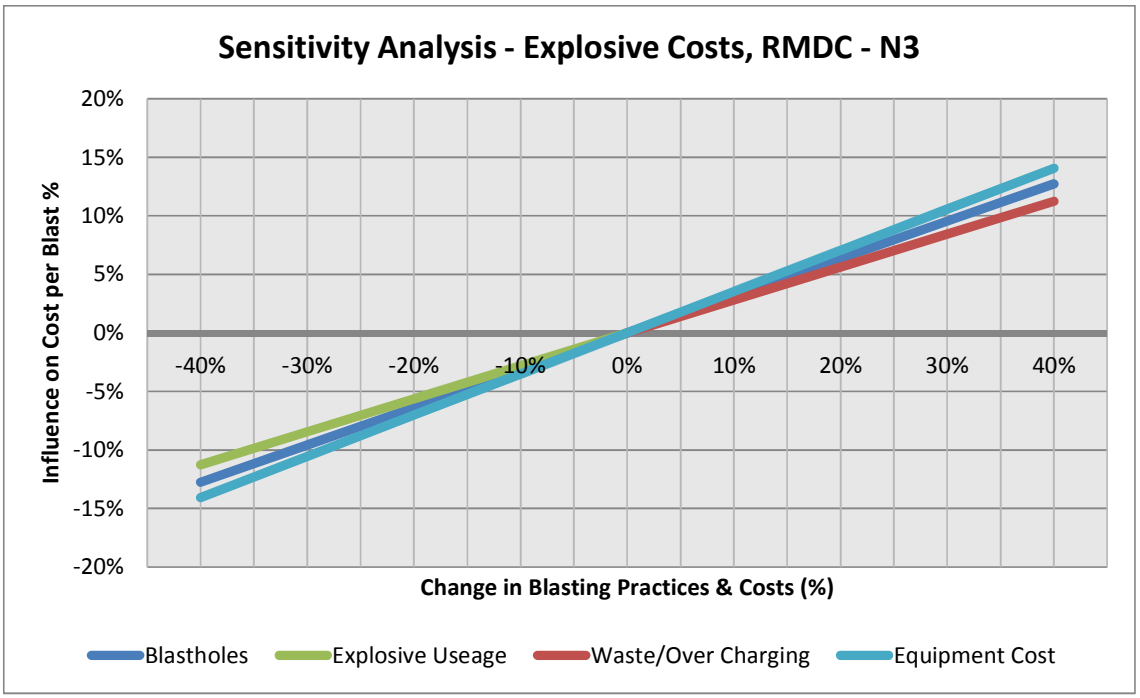


Figure 48, Sensitivity Analysis – Influence of blasting practices and variables on the cost per blast, RMDC – North 3 raise

7.2 Importance of Revenue to Blasting Costs and Practices

As discussed in the literature review, drilling and blasting operations are of vital importance to the profitability of mining operations as they are directly responsible for the liberation of payable ore from the stope face. Any loss in advance experienced through poor blasting practices or poor explosive selection is directly proportional to the loss in revenue that will be experienced by the mining operation. As it is possible to improve the consistency and performance of daily blasting operations through the selection of a better suited explosive, any loss in advance experienced through the use of an unsuitable or poor quality explosive could be compared to an increase in the cost of explosives used to achieve the blast results. Figure 49 compares the importance of revenue on the UE1A panel to the cost of explosives and equipment per blast as discussed in Figure 47. From the figure it is evident that a decrease in advance of only 1cm is equivalent to a 100% increase in the cost of explosives used to achieve the blast results. As a lost blast in turn represents approximately 100cm of lost advance, the failure of 1% of blasts due to the presence of water blastholes or the loss of 1% of blasts due to the presence of preventable oversize rock or damaged hanging is equivalent to a 100% increase in the cost of explosives for the entire mining operation.

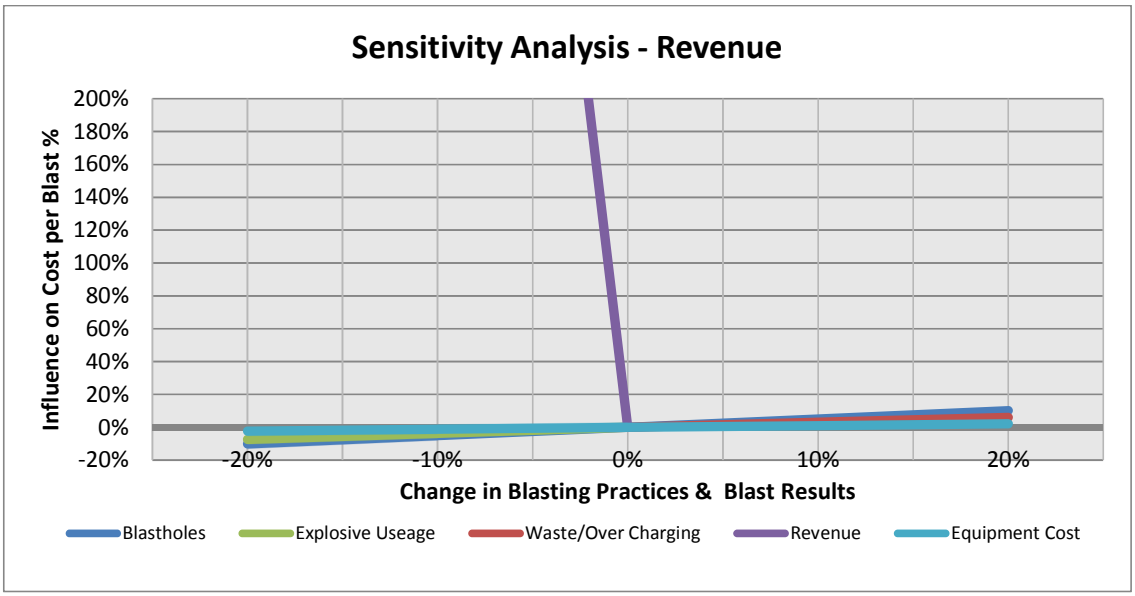


Figure 49, Sensitivity Analysis – Importance of revenue to blasting costs & practices

As discussed and illustrated in Table 27, revenue is an overpowering factor in the selection of an explosives system and any factor that may prevent the accomplishment of a daily quality blast of full advance needs to be carefully managed or eliminated. As discussed in the literature review, one area responsible for the loss of revenue is the operational downtime incurred through section 54 notices. Should the improper control of a class 1 explosive on the operation result in a section 54 notice being issued to the mine a significant loss in revenue will be incurred by the operation. As the average downtime experienced through a section 54 notice is 3.9 days and over this period 3 blasts are potentially lost per panel, the loss in revenue per notice totals 1.6% per annum. This equates to a loss in revenue of R 815'298 per section 54 per panel exceeding the total cost of explosives for the panel for the year.

Table 27, Importance of Revenue to Explosives Selection

Value per panel		
Panel length (m)	30	
Panel height (m)	1.2	
Drilled length (m)	1.13	
Volume per blast (m^3)	40.6	
Rock Density (t/m^3)	2.7	
Tonnes per blast	110	
Recovered grade (g/t)	6	
Gold recovered per blast (kg)	0.658	
Gold Price ($/Oz)	1260	
Exchange Rate R/$	10.2	
Value per blast R	271,766	
Loss of Revenue – Section 54		
Blasts per year (per panel)	192	
Revenue per year (per panel) (mill)	52.2	
Lost blasts per section 54 (per panel)	3	
Loss in Annual Revenue per Section 54 (per panel)	R 815,298	1.6%
Loss of Revenue – Blast Performance		
1% Failure of Blasts as % of explosives costs	R 2,718	106%
1cm Loss in advance as % of explosives costs	R 2,265	88%

7.3 Shaft Availability

In order to gain a better understanding of the reduction in shaft time available through the implementation of a class 5.1 bulk explosives system, time studies were undertaken for the handling of cartridged explosives through the Rand Uranium, Cook 2 shaft. Shaft head deliveries for cartridged explosives took place three mornings a week, while accessories were delivered on the remaining two mornings in the week. Time studies for the handling of explosives revealed that while 30 minutes of shaft time were planned for the daily transportation of explosives through the shaft, an average of 50 minutes or 5% of the available shaft time was utilised per delivery. This is illustrated in the utilisation of available shaft time for Cook 2 shaft in Figure 50.

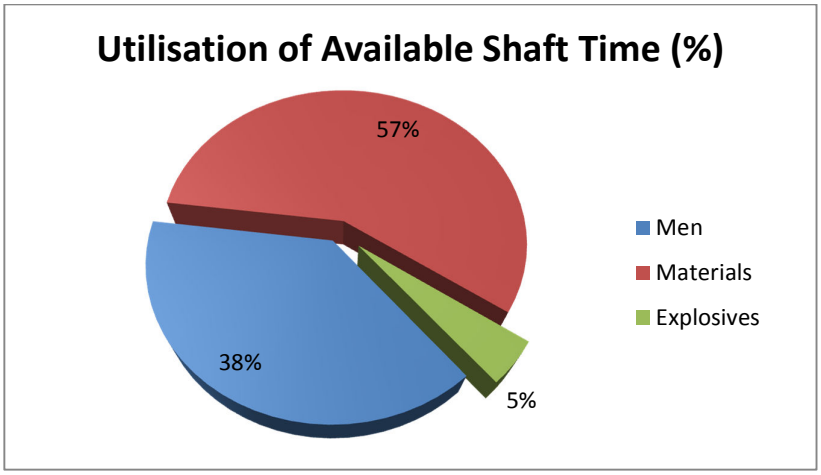


Figure 50, Utilisation of available shaft time, Cook 2

Table 28, Shaft Time Required for the Handling of Cartridged Explosives, Cook 2 Shaft

Day	Mass per Delivery (tonnes)	Explosive Cars Required per Delivery	Explosive Cars per Cage	Mass per Car (tonnes)	Loading Time (min/tonne)*	Shaft Time (min/tonne)
Day 1	3.1	5	2	0.62	21.6	11.3
Day 2	5	6	2	0.83	16.2	13
Day 3	4.15	5	2	0.83	17.3	11.6
Average	4.1	5.3	2	0.77	17.9	12.1

* Six personnel required for the off-loading of explosives from BME truck to explosives cars

Table 28 indicates the loading and shaft time required for the transportation of explosives through the Cook 2 shaft. Over the three days on which time studies were undertaken, an average mass of 4.1 tonnes of explosives was transported through the shaft per day. Following arrival of the explosives truck in the delivery bay, an average of 18 minutes were required per tonne for a six man team to off load the explosives into the designated explosives cars before shunting the cars to the shaft head for transportation underground. Following the closure of the shaft to men and materials an average of 12.1 minutes per tonne of shaft time was required for the transportation of explosives to the required levels.

One of the areas of greatest inefficiency identified in the transportation of explosives was the requirement to transport a small quantity of explosive to each level on a regular basis. If a larger quantity of explosive could be transported through the system less frequently, the length of shaft time required for the transport of explosives could be significantly reduced. Due to the strict controls on the transportation and storage of explosives on underground operations, an average of only 0.77 tonnes of explosives were loaded per explosives car before the car was sent to the designated working place. As only two explosives cars could be loaded per cage, this allowed for a payload significantly less than the maximum payload for the cage thus increasing the number of cycles required for the transportation of explosives underground.

Table 29, Reduction in Shaft Time Possible Through a Class 5.1 Blasting Intermediate

Explosives System	Mass per Car (tonnes)	Shaft Time (min/tonne)	% Reduction in Shaft Time for Explosives
Cartridged Explosives	0.77	12.1	-
Pumpable Emulsion System	2.2	4.2	65%

As blasting agents can be transported in larger quantities and stored for longer periods of time than cartridged explosives, transfer cassettes in excess of 2.2 tonnes can be used in place of existing explosives cars. As a result of this increase in mass, a reduction in shaft time in excess of 65% is possible due to the increased efficiency of each shaft cycle as illustrated in Table 29. As the average time for the transport of explosives through the shaft system is currently 50 minutes per delivery, a reduction in shaft time in excess of 32 minutes per day is

achievable. Due to their class 5.1 classification, pumpable emulsions can also be transported with materials throughout the day increasing the flexibility available within shaft and tramming cycles. While the demand for increased shaft availability will vary for individual mining operations, an increase in shaft availability will allow for increased time for the transport of materials and support reducing the risk of lost blasts and allowing for a possible increase in production capacity. Alternative savings include a reduction in electricity and maintenance requirements for the shaft and a possible reduction in overtime and off-peak maintenance expenses due to increased shaft availability within the week for maintenance.

- *Pumping of Emulsion Underground*

In recent years, bord and pillar mining operations at shallow depths have begun to install vertical pipe columns and infrastructure allowing emulsion to be pumped from storage tanks on surface to storage and distribution tanks underground. Given the success of these operations in eliminating the shaft time required for the transport of blasting intermediates, interest in piped emulsion logistics systems has increased from deep level operations. As bulk logistics systems continue to develop it is foreseeable that deep level operations will soon too be able to implement such systems thereby eliminating all together the shaft time required for the transportation of explosives.

8 Conclusion

Through the comparison of commercial explosives available for use in narrow reef mining operations, a number of improvements in safety and operational efficiency have been proposed through the implementation of pumpable emulsions. Arguably the greatest advantage of pumpable emulsions lies in their UN Class 5.1 classification. Through this classification the degree of legislation and control applicable to class 1 explosives is reduced allowing for considerable advantages throughout the transportation and storage of blasting intermediates. Additional advantages of the non-explosive classification of the system are evident in the prevention of the theft of explosives and the downstream use of commercial explosives in criminal activities. As a result of the reduction in legislation, blasting intermediates can be transported with other materials saving tramming and shaft time as well as allowing for longer storage periods underground. The bulk nature of pumpable emulsions too allows them to exhibit a number of advantages over pre-packaged explosives systems. Of greatest significance to use of bulk explosives is the full coupling of the explosive within the blasthole. This has been shown to increase the energy available at the toe of the blasthole as well as the efficiency through which shock energy is transmitted from the explosive into the surrounding rock mass.

While pumpable emulsions offer considerable advantages to underground operations it is important to understand their limitations with regard to use in the narrow reef environment. Likely the greatest challenge to the implementation of pumpable emulsions in conventional narrow reef operations is the dependence of blasting intermediates on pump technology for sensitisation during the loading cycle. Challenges encountered in the narrow reef environment including the poor level of skills within the narrow reef workforce, the large number of pumps required for rollout of the system, the confined nature of the operating environment and the relatively poor utilisation of each individual charging unit illustrate the fundamental differences that exist between the narrow reef environment and traditional mechanised emulsion technology. In order to allow for the successful implementation of the new emulsion technology in the narrow reef environment charging equipment needed to be safer, allow for an easier user interface, achieve constant density under varying operating conditions and be more reliable and cost effective than traditional charging units used in mechanised mining applications.

- *Controlled Emulsion Tests - Losberg*

In order to determine the influence of priming practices on the VOD of Emulsion B in underground field trials, a range of priming practices were evaluated through the use of steel pipes to simulate underground blastholes. Results obtained through priming tests revealed that the use of a 12 gram pentolite booster achieved the highest average VOD at 3320 m/s, while the use of a reverse primed 8D detonator resulted in the lowest coefficient of variance and the highest initial VOD (over-run). Despite the marginal improvement in VOD achieved through ideal priming practices, it is unlikely that a specific method would allow for a definable difference in blast results in field trials. During priming tests, the use of a single cartridged emulsion primer at the base of test pipes resulted in the failure of 2 of the 3 test pipes, with a low initial VOD experienced for the third pipe. This result was unexpected and further testing is necessary in order to eliminate the possibility of equipment error and to gain a better understanding of the effect that the action of tamping may have had on the VOD of the gas sensitised cartridged emulsion used in the priming test.

VOD results obtained with Emulsion B in comparatively confined (steel) and comparatively unconfined (PVC) pipes indicated unexpected results during the evaluation of steady state VOD in increasing pipe diameters. Results for the test revealed similar VOD trends for Emulsion B in small diameter steel and PVC pipes despite the intended increase in confinement through the use of steel pipes. Results obtained for Emulsion B within PVC pipes begun with a comparatively low average VOD of 2749 m/s for 27.8mm diameter pipes, increasing to 3275 m/s for 36mm pipes and 3810 m/s for 45mm pipe. Only a single reading was obtained for the 45mm PVC pipe. VOD results for Emulsion B in steel pipes begun at a similar average VOD of 2672 m/s in 27.8mm pipes increasing to 3270 m/s for 37.1mm pipes and 4005 m/s in 45.1mm pipes. The coefficient of variance experienced throughout the test was consistent with theory discussed in section 2.3.1 as the coefficient of variance decreased with increasing charge diameter.

Possible reasons for the similarity in results achieved in both steel and PVC pipes include the comparatively greater strength and confinement of PVC in small pipe diameters than in large diameters and the different VOD test equipment utilised in the VOD tests. As the AECE VOD-3 VOD Timer was used in the PVC pipe tests, the placement of the optical sensor lines across the centre of the pipe may have resulted in the measurement of the end of the run-up or

over-run zone that would have been over-driven through the practice of reverse priming. This may have indicated an inflated VOD for the PVC pipes. As the MREL Handitrap VOD recorder was used for the steel pipe tests in test 4 a better reading of VOD could be obtained for the pipes allowing for more accurate results. Comparative testing of the VOD-3 VOD Timer and the MREL Handitrap as well as further testing on the extremes of confinement is necessary in order to understand the similarity in VOD results for steel and PVC pipes.

Also evident in the results of the Losberg VOD test is the lower VOD of pumpable emulsions when compared to cartridged emulsions due to the increased water content of pumpable emulsions (for class 5.1 classification) and larger bubble size due to non-ideal on site sensitisation process. This result confirms that pumpable emulsions will have different performance characteristics and a lower VOD profile when compared to cartridged emulsions. These factors will in turn have implications on the breakage mechanisms of shock and gas within the blast as discussed within the literature review. The high average VOD for the single set of tests undertaken with Emulsion A indicate that while the VOD of a pumpable emulsion is lower than cartridged emulsions, it is possible to manipulate the VOD of the emulsion by altering factors such as the water content of the formulation. This will allow for the future development of both higher and lower VOD emulsions for use in underground blasting applications depending on the requirements of the specific blasting application.

- *Field Trials*

Blast results monitored in field trials on four faces across two mining operations experienced a noticeable increase in blasting efficiency through the implementation of a pumpable emulsion explosives system in place of cartridged emulsion explosives. While a reduction in powder factor (through the elimination of overcharging) was a significant contributor to increase in overall blasting efficiency, the advance achieved per blast likewise increased by as much as 14% compounding the improvement seen in blasting efficiency. Values for powder factor measured throughout the trial decreased by up to 23% throughout the trial while the achieved energy factor throughout the trial decreased between 15% and 37% for the various trial faces. These results support the proposed increase in blasting efficiency discussed in section 2.3.3. Vital to the success of the pumpable emulsion system was the full coupling achieved by the explosive column within the blasthole. Coupling tests undertaken with cartridged emulsions in the trial achieved an average coupling ratio of only 84.9% with a

standard deviation of 3.8% for the use of 29 x 270mm gas sensitised emulsion cartridges in 36mm blastholes. A single test undertaken with 29 x 270mm emulsion cartridges in 32mm blastholes indicated an average coupling ratio of 93.8%, with a standard deviation of 0.4%. This result clearly illustrates the increase in coupling and loading consistency as initial cartridge diameter approaches blasthole diameter.

Though it was evident through underground trials that the use of pumpable emulsions resulted in an increase in the advance and blasting efficiency, no specific increase could be calculated due to the number of variables present in blasting practices and rock properties on the trial faces. Poor drilling practices and the overcharging of blastholes across all stope faces elevated powder factors and energy factors across cartridged emulsion blasts, reducing their efficiency and preventing their direct comparison with the results achieved through the use of pumpable emulsions. The comparison of powder factors used on the K9 and UE1A reefs on Rand Uranium allows for the comparison of differences in drilling quality and rock properties on the energy required to achieve full advance on the same shaft. These results are illustrated in Figure 46 where Emulsion B achieved 100% advance in the K9 at an energy factor less than half that required to achieve full advance on the UE1A reef.

Teething problems experienced through the implementation of pumpable emulsions added to the complexity of the study as specific blasting practices needed to be introduced together with the pumpable emulsion system in order to ensure the reliable performance of the system. From the outset of the trail stemming was introduced in order to prevent detonators being ejected from blastholes prior to detonation, while the use of cartridged emulsion primers was later introduced in all stoping trials to anchor the detonator within the base of the blasthole. While it was demonstrated through the controlled Losberg tests that the use of the cartridged emulsion primer would have had little effect on the blast results, use of stemming with pumpable emulsions throughout the trial period will have increased the efficiency of blasting practices as discussed in section 2.3.4. The total increase in efficiency evident in blast results is therefore as a result of the cumulative effects stemming, the reduction of overcharging practices, improvements in drilling quality through increased supervision and the aforementioned performance characteristics and physical properties of Emulsion B. A better understanding of each of these variables needs to obtained before a definitive increase in blasting efficiency can be determined for the implementation of pumpable emulsions.

In order to allow for financial comparison of the pumpable emulsion system with existing explosives systems Section 7 of the report contains a cost comparison for cartridged and pumpable emulsion explosives for the four trial faces based on an estimation of market prices. Though the price per tonne of pumpable emulsions is considerably lower than that of cartridged emulsions, it is important to understand the financial implications of elevated equipment requirements necessary for the implementation of pumpable emulsions.

As the number of blastholes on the K9 panel necessitated the use of 1.9 tonnes of emulsion for the panel per month, a loading cost (excluding labour) of R2.60 was achieved per kilogram of pumpable emulsion utilised on the panel. In contrast, as the North 3 gully necessitated the use of only 28kg of pumpable emulsion per blast or 440 kg per month, an average loading cost of R 10.63 was achieved per kilogram of emulsion loaded in the round. This difference in loading cost is clearly evident in Table 23, Table 24, Table 25 and Table 26 when compared to the equivalent loading and logistics cost for cartridged emulsions on the same faces at R 0.54 and R 2.36 respectively. Due to the acceptable utilisation of the Portable Pump on the K9 panel the cost per tonne broken decreased by 21% through the use of Emulsion B over cartridged emulsions. The opposite was however evident for the implementation of pumpable emulsion system on the N3 raise. As the utilisation of the Portable Pump was poor, the cost per tonne broken increased by 4% through the implementation of Emulsion B over cartridged emulsions.

From this comparison it is evident that through the implementation of pumpable emulsion systems, narrow reef operations have the ability to increase both the efficiency and feasibility of daily blasting practices. Though the cost per tonne broken with pumpable emulsions may exceed the cost per tonne broken with cartridged emulsions on small faces, the financial benefit from the significantly greater mass of explosive utilised on larger production faces will increase the overall feasibility of the operation. Also evident from the financial comparison is the degree to which revenue losses through poor explosives performance, reliability and section 54's impact the feasibility of an operation far out weighing the cost of explosives. Though the financial implications of these factors are difficult to predict, their influence on the profitability of a mining operation cannot be overlooked.

It is evident through this work that pumpable emulsions are able to provide narrow reef operations with increased levels of flexibility, efficiency and control, unavailable or limited through the use of alternative commercially available explosives. This increase in

performance and efficiency throughout the mining operation renders pumpable emulsions as a financially desirable alternative to existing explosives systems within the narrow reef environment. This project, despite limitations, provides a foundation for the continued development of pumpable emulsion technology for conventional narrow reef mining operations. With further technological development, it is thought that pumpable emulsions exhibit the potential to become the preferred blasting system for use throughout the underground narrow reef mining industry.

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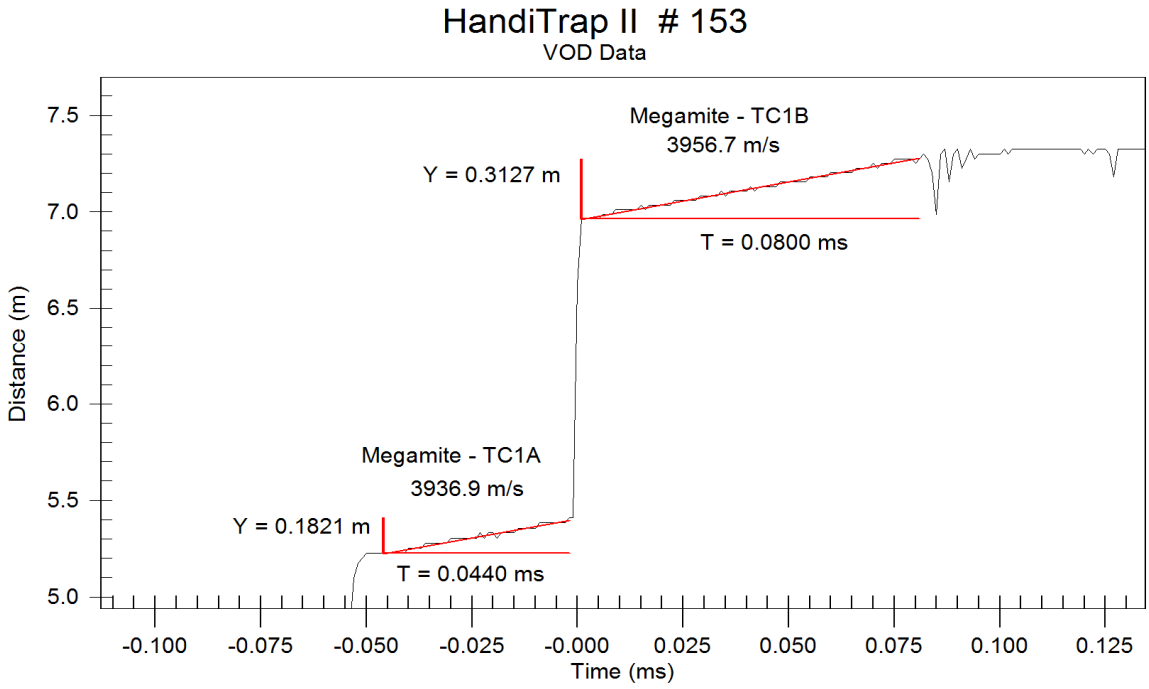
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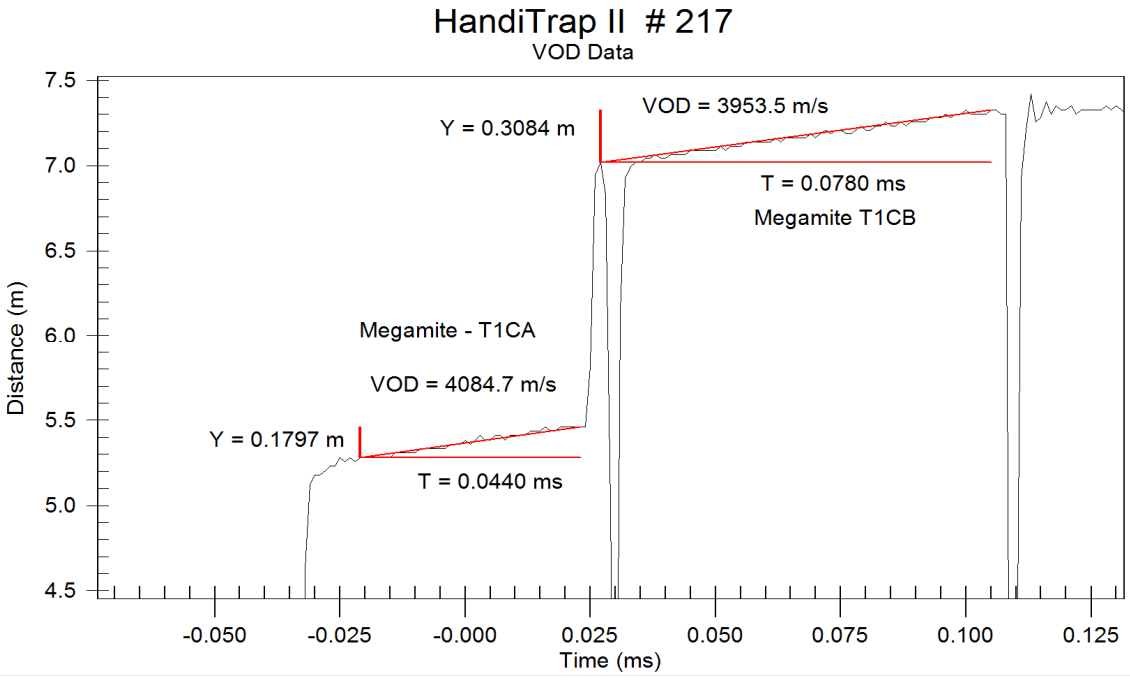
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10 Appendix

Appendix 1, VOD Test 4 – Results

T1CA & T1CB, Comparison of Handitrap 217 and Handitrap 153





Appendix 2, VOD Test Results – Series 1

VOD results as obtained on Emulsion A

Test Series: 1

Method: Measured time between two points

Site: Losberg

Test No.	Product	Method of Charging	Internal Diameter (mm)	Confinement	Pipe Length (m)	Pipe Volume (litres)	Measured Length (mm)	Time (us)	Calculated VOD (m/s)	Mass at Firing (g)	Calculated Firing Density (g/cc)	Primer
A	Emulsion A	Portable Pump	27.8	Steel, 3.05mm wall	0.6	0.36	300	-	-	440	1.02	8D, Reverse
B	Emulsion A	Portable Pump	27.8	Steel, 3.05mm wall	0.6	0.36	300	83.3	3601	413	0.96	8D, Reverse
C	Emulsion A	Portable Pump	27.8	Steel, 3.05mm wall	0.6	0.36	300	90.8	3304	390	0.90	8D, Reverse
D	Emulsion A	Portable Pump	27.8	Steel, 3.05mm wall	0.6	0.36	300	97.3	3083	384	0.88	8D, Reverse
E	Emulsion A	Portable Pump	27.8	Steel, 3.05mm wall	0.6	0.36	300	85	3529	370	0.85	8D, Reverse
F	Emulsion A	Portable Pump	27.8	Steel, 3.05mm wall	0.6	0.36	300	-	-	360	0.82	8D, Reverse

Appendix 3, VOD Test Results – Series 2

VOD results as obtained on Emulsion B

Test Series: 2

Method: Measured time between two points

Site: Losberg

Test No.	Product	Method of Charging	Internal Diameter (mm)	Confinement	Pipe Length (m)	Pipe Volume (litres)	Measuring Length (mm)	Time (us)	Calculated VOD (m/s)	Cup Density at Firing (g/cc)	Primer
1	Emulsion B	Portable Pump	27.8	PVC, 2.1mm wall	1.0	0.61	310	118.5	2616	1.1	8D, Reverse
2	Emulsion B	Portable Pump	27.8	PVC, 2.1mm wall	1.0	0.61	310	112.5	2756	1.08	8D, Reverse
3	Emulsion B	Portable Pump	27.8	PVC, 2.1mm wall	1.0	0.61	310	111.5	2780	1.05	8D, Reverse
4	Emulsion B	Portable Pump	27.8	PVC, 2.1mm wall	1.0	0.61	310	108	2870	1.03	8D, Reverse
5	Emulsion B	Portable Pump	27.8	PVC, 2.1mm wall	1.0	0.61	310	113.9	2722	0.99	8D, Reverse
6	Emulsion B	Portable Pump	36	PVC, 2mm wall	1.0	1.02	310	99.9	3103	1.14	8D, Reverse
7	Emulsion B	Portable Pump	36	PVC, 2mm wall	1.0	1.02	310	90.6	3422	1.11	8D, Reverse
8	Emulsion B	Portable Pump	36	PVC, 2mm wall	1.0	1.02	310	93.9	3301	1.04	8D, Reverse
9	Emulsion B	Portable Pump	36	PVC, 2mm wall	1.0	1.02	310	-	No Trigger	0.98	8D, Reverse

Appendix 4, VOD Test Results – Series 4

VOD results as obtained on Emulsion B
Test Series: 4
Method: MREL Handitrap - Continuous Measurement - Inside Pipe
Site: Losberg

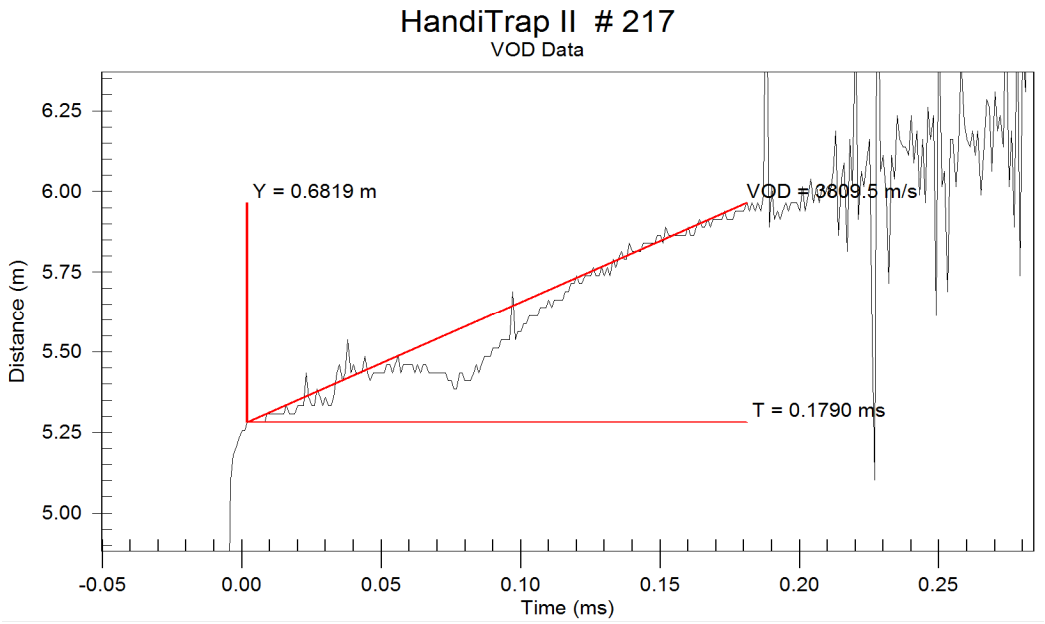
Test No.	Product	Method of Charging	Internal Diameter (mm)	Confinement	Pipe Length (m)	Pipe Volume (litres)	Cup Density on Firing (g/cc)	Calculated Mass per Pipe (kg)	Primer	Recorded VOD (m/s)	Initial VOD (m/s)	Handitrap ID	File No.	VOD Probe
T1CA	Cartridge M	-	29.0	Unconfined	0.54	0.36	1.16	0.41	8D, Forward	3937	-	153	35	Probecable-HT
T1CA	Cartridge M	-	29.0	Unconfined	0.54	0.36	1.16	0.41	8D, Forward	4084	-	217	35	Probecable-HT
T1CB	Cartridge M	-	29.0	Unconfined	0.54	0.36	1.16	0.41	8D, Forward	3957	-	153	36	Probecable-HT
T1CB	Cartridge M	-	29.0	Unconfined	0.54	0.36	1.16	0.41	8D, Forward	3954	-	217	36	Probecable-HT
T2 - I	Emulsion B	Portable Pump	45.0	PVC, 2.5mm wall	1	1.59	1.15	1.80	8D, Reverse	3810	-	217	40	Probecable-HT
T2 - O	Emulsion B	Portable Pump	45.0	PVC, 2.5mm wall	1	1.59	1.15	1.80	8D, Reverse	3202	-	153	37	Probecable-HT
1A	Emulsion B	Portable Pump	27.8	Steel, 3.05mm wall	1.5	0.91	1.10	1.00	8D, Forward	2570	3154	153	47	Probecable-HT
1B	Emulsion B	Portable Pump	27.8	Steel, 3.05mm wall	1.5	0.91	1.10	1.00	8D, Forward	2595	2595	217	49	Probecable-HT
1C	Emulsion B	Portable Pump	27.8	Steel, 3.05mm wall	1.5	0.91	1.08	0.98	8D, Forward	2852	3185	153	50	Probecable-HT
2A	Emulsion B	Portable Pump	37.1	Steel, 2.95mm wall	1.5	1.62	1.12	1.82	8D, Forward	3120	3295	153	43	Probecable-HT
2B	Emulsion B	Portable Pump	37.1	Steel, 2.95mm wall	1.5	1.62	1.12	1.82	8D, Forward	3466	3932	153	45	Probecable-HT
2C	Emulsion B	Portable Pump	37.1	Steel, 2.95mm wall	1.5	1.62	1.08	1.75	8D, Forward	3224	3530	217	53	Probecable-HT

VOD Test Results – Series 4 Continued

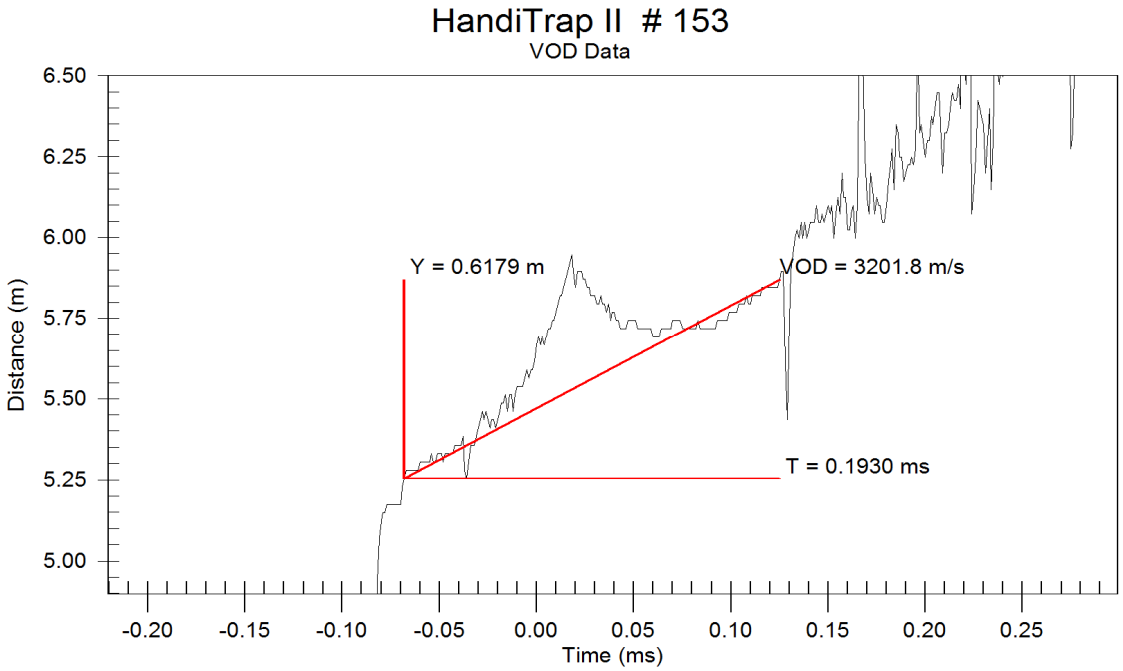
Test No.	Product	Method of Charging	Internal Diameter (mm)	Confinement	Pipe Length (m)	Pipe Volume (litres)	Cup Density on Firing (g/cc)	Calculated Mass per Pipe (kg)	Primer	Recorded VOD (m/s)	Initial VOD (m/s)	Handitrap ID	File No.	VOD Probe
3A	Emulsion B	Portable Pump	45.1	Steel, 1.65mm wall	1.2	1.92	1.07	2.05	8D, Forward	4063	-	153	61	Probecable-HT
3B	Emulsion B	Portable Pump	45.1	Steel, 1.65mm wall	1.2	1.92	1.07	2.05	8D, Forward	3947	-	217	59	Probecable-HT
3C	Emulsion B	Portable Pump	45.1	Steel, 1.65mm wall	1.2	1.92	1.06	2.03	8D, Forward	No Trigger	-	-	-	Probecable-HT
4A	Emulsion B	Portable Pump	37.1	Steel, 2.95mm wall	1.5	1.62	1.06	1.72	8D, Reverse	3339	3700	217	62	Probecable-HT
4B	Emulsion B	Portable Pump	37.1	Steel, 2.95mm wall	1.5	1.62	1.06	1.72	8D, Reverse	3158	3567	153	70	Probecable-HT
4C	Emulsion B	Portable Pump	37.1	Steel, 2.95mm wall	1.5	1.62	1.06	1.72	8D, Reverse	3217	3700	153	66	Probecable-HT
5A	Emulsion B	Portable Pump	37.1	Steel, 2.95mm wall	1.5	1.62	1.08	1.75	29 x 270 - 200g emulsion cartridge	3330	3000	153	72	Probecable-HT
5B	Emulsion B	Portable Pump	37.1	Steel, 2.95mm wall	1.5	1.62	1.08	1.75	29 x 270 - 200g emulsion cartridge	Failed	-	-	-	Probecable-HT
5C	Emulsion B	Portable Pump	37.1	Steel, 2.95mm wall	1.5	1.62	1.08	1.75	29 x 270 - 200g emulsion cartridge	Failed	-	-	-	Probecable-HT
6A	Emulsion B	Portable Pump	37.1	Steel, 2.95mm wall	1.5	1.62	1.08	1.75	12g, Forward	3245	3567	217	75	Probecable-HT
6B	Emulsion B	Portable Pump	37.1	Steel, 2.95mm wall	1.5	1.62	1.07	1.74	12g, Forward	3394	3615	153	77	Probecable-HT
6C	Emulsion B	Portable Pump	37.1	Steel, 2.95mm wall	1.5	1.62	1.07	1.74	12g, Forward	Pre-trigger	-	-	-	Probecable-HT

Appendix 5, VOD Data – VOD Test 4

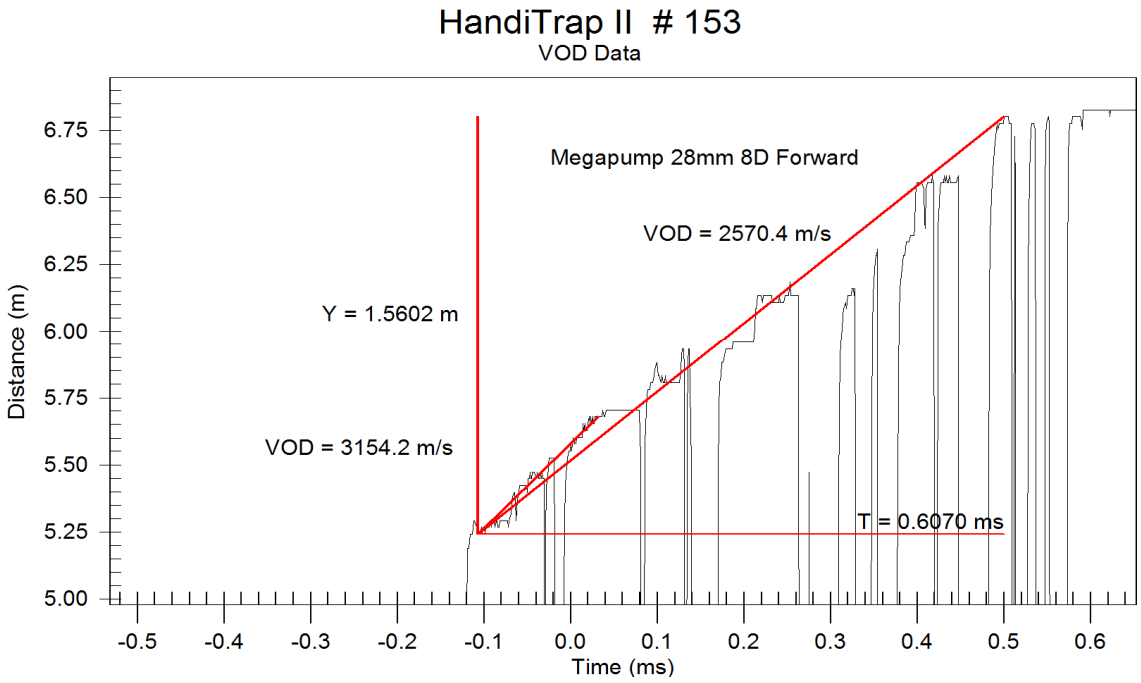
Pipe T2 - I, VOD measured inside 45mm PVC pipe - Reverse toe primed



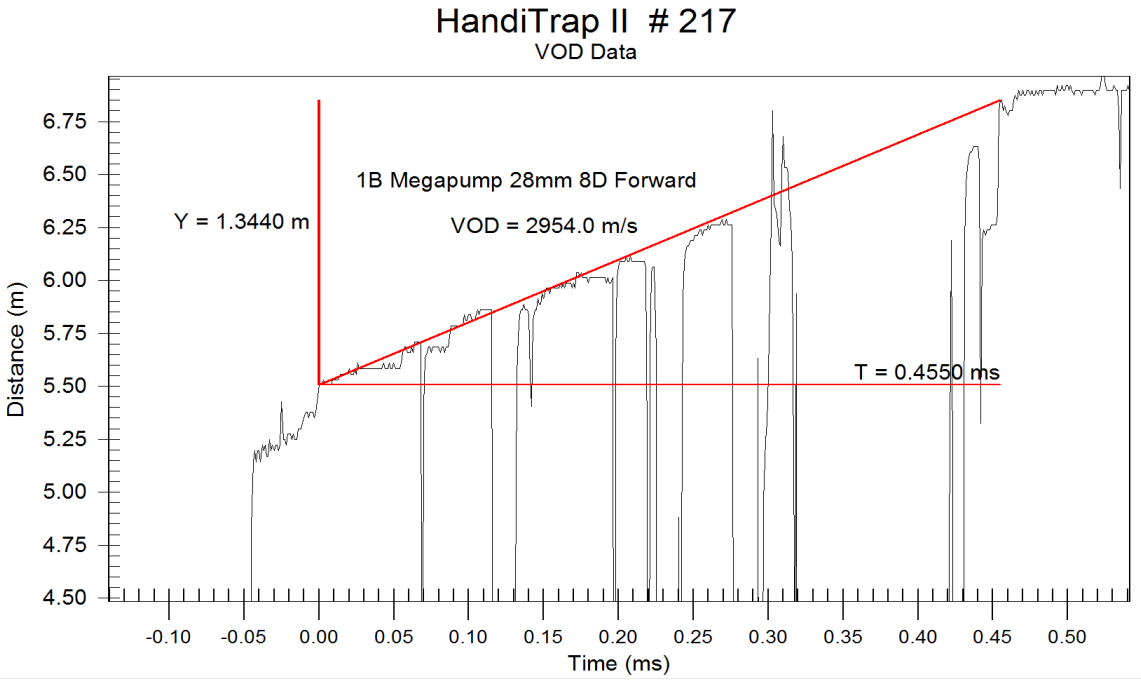
Pipe T2 – O, VOD measured outside 45mm PVC pipe - Reverse toe primed



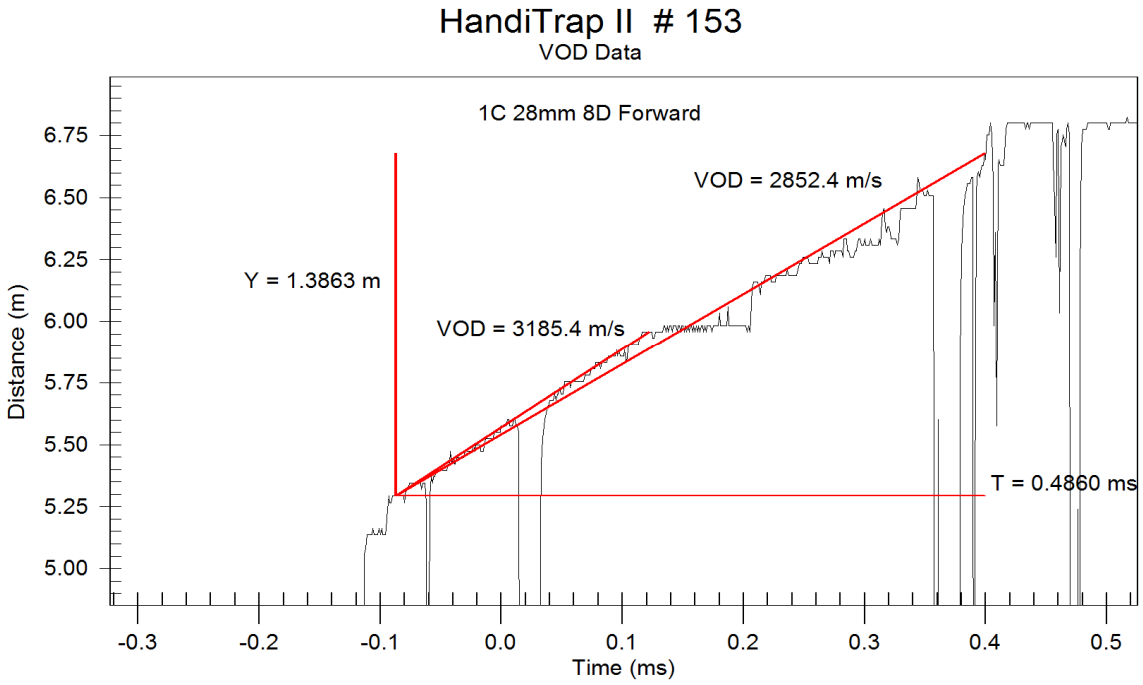
Pipe 1A, VOD measured inside 27.8mm steel pipe - Forward toe primed



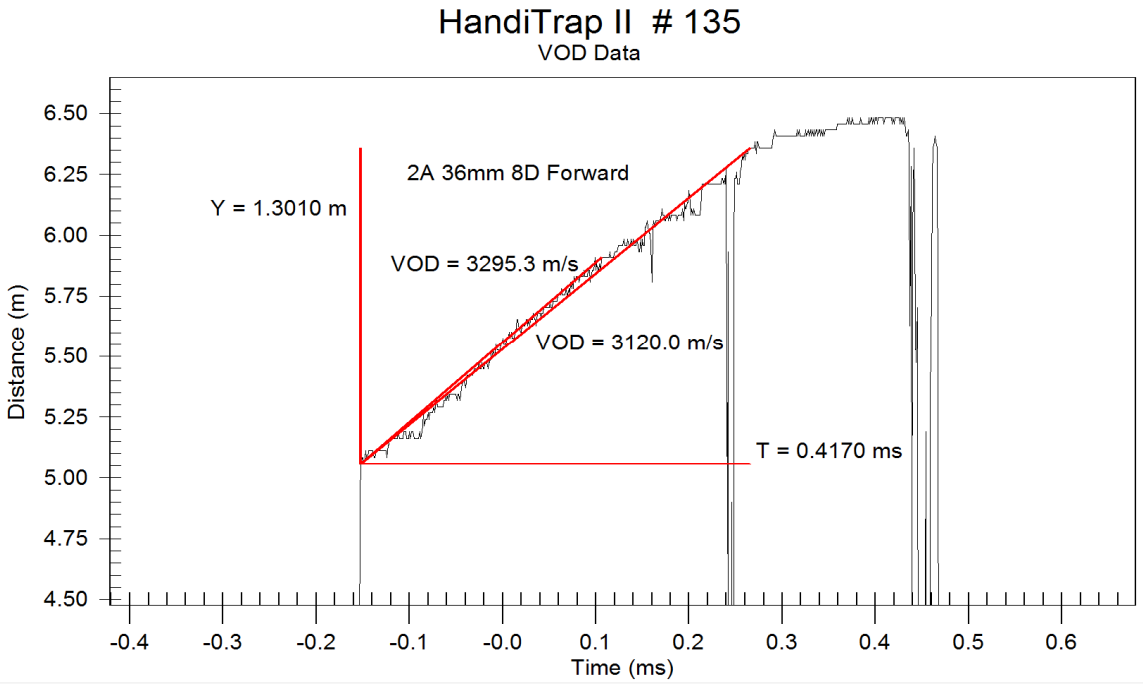
Pipe 1B, VOD measured inside 27.8mm steel pipe - Forward toe primed



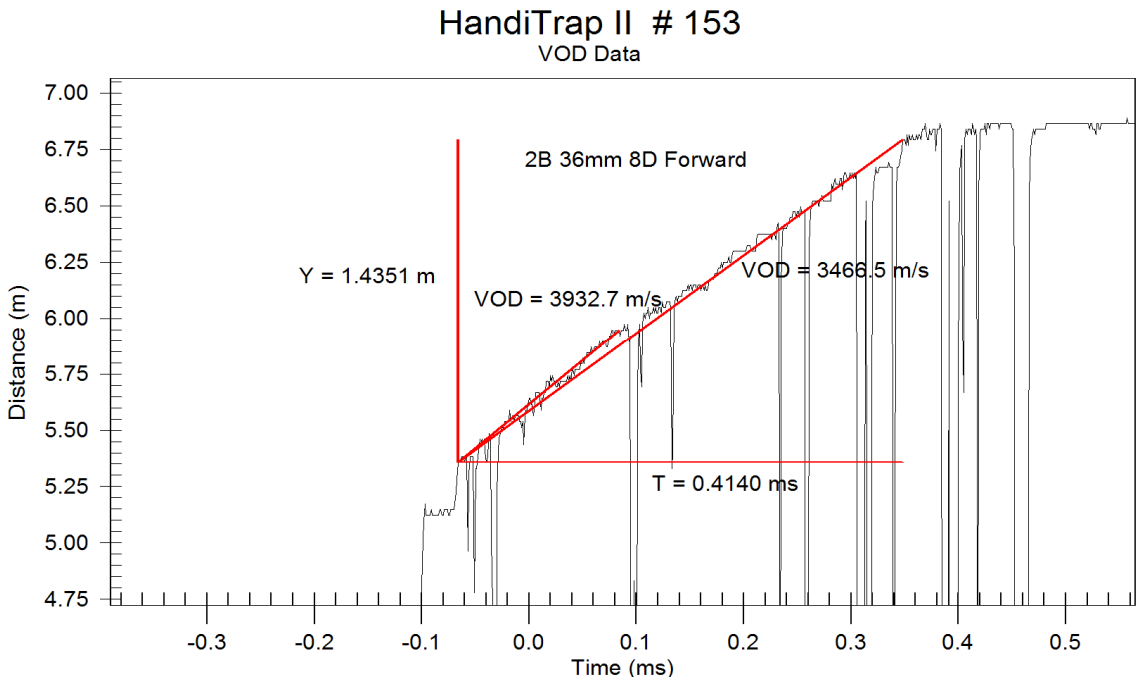
Pipe 1C, VOD measured inside 27.8mm steel pipe - Forward toe primed



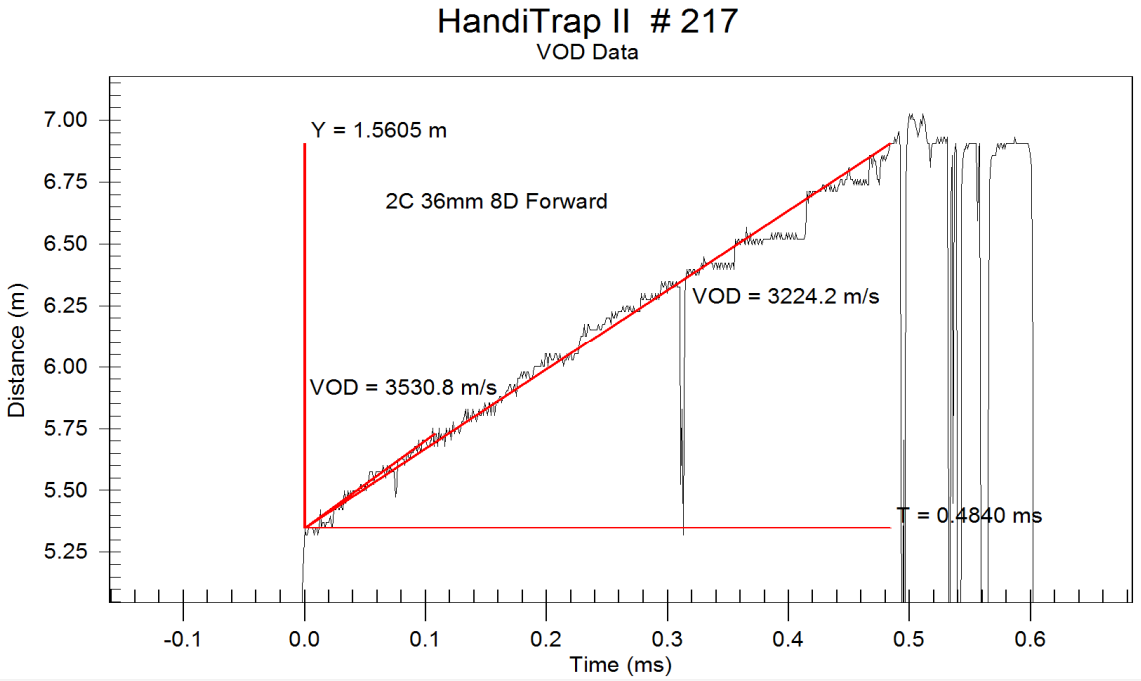
Pipe 2A, VOD measured inside 37.1mm steel pipe - Forward toe primed



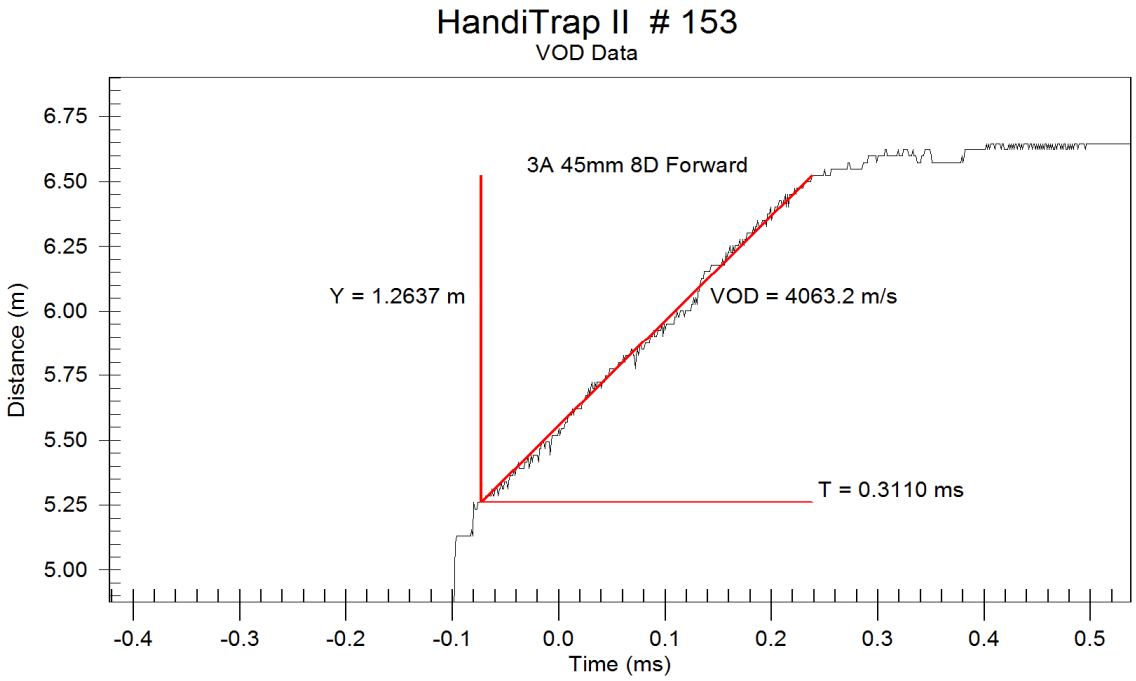
Pipe 2B, VOD measured inside 37.1mm steel pipe - Forward toe primed



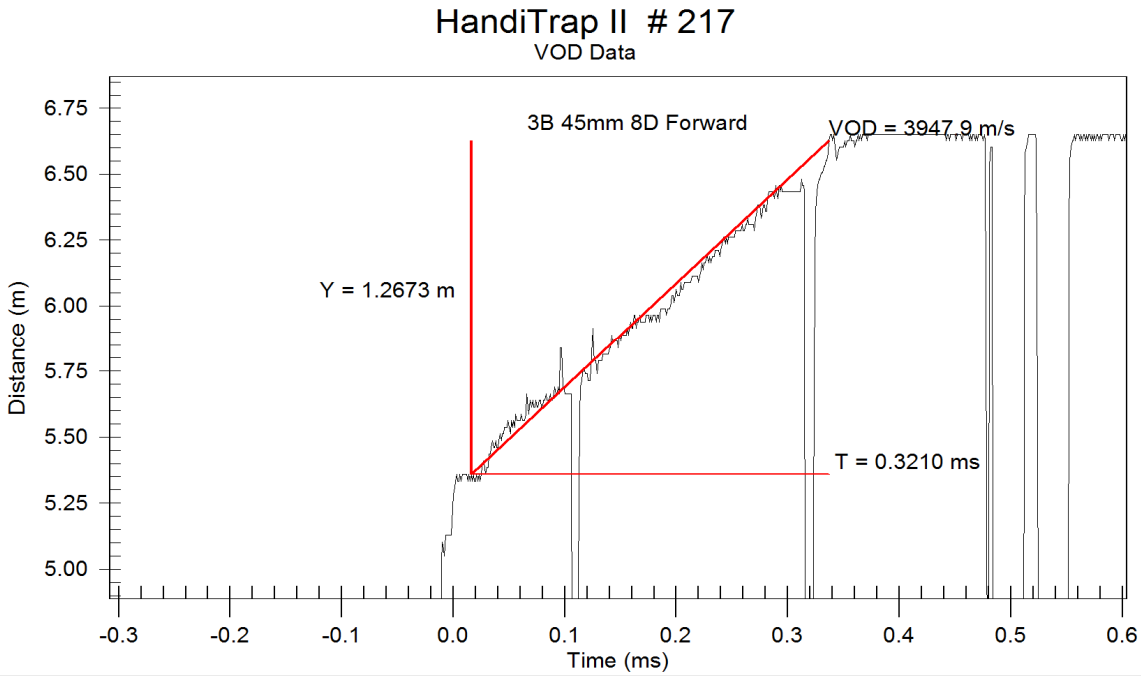
Pipe 2C, VOD measured inside 37.1mm steel pipe - Forward toe primed



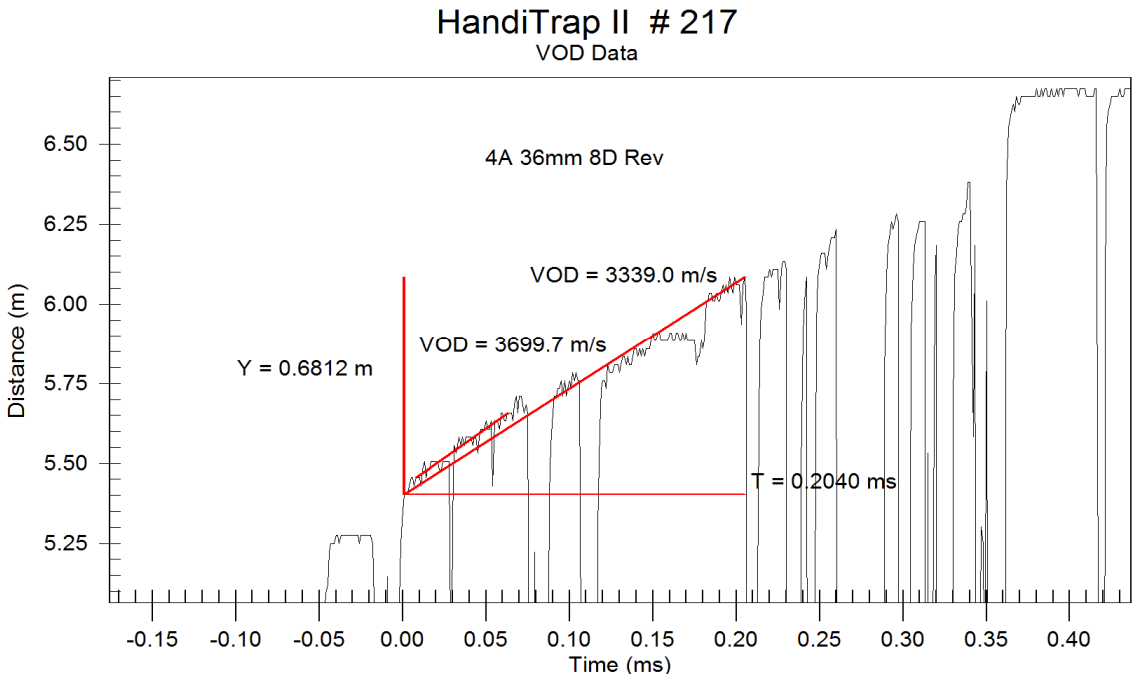
Pipe 3A, VOD measured inside 45.1mm steel pipe - Forward toe primed



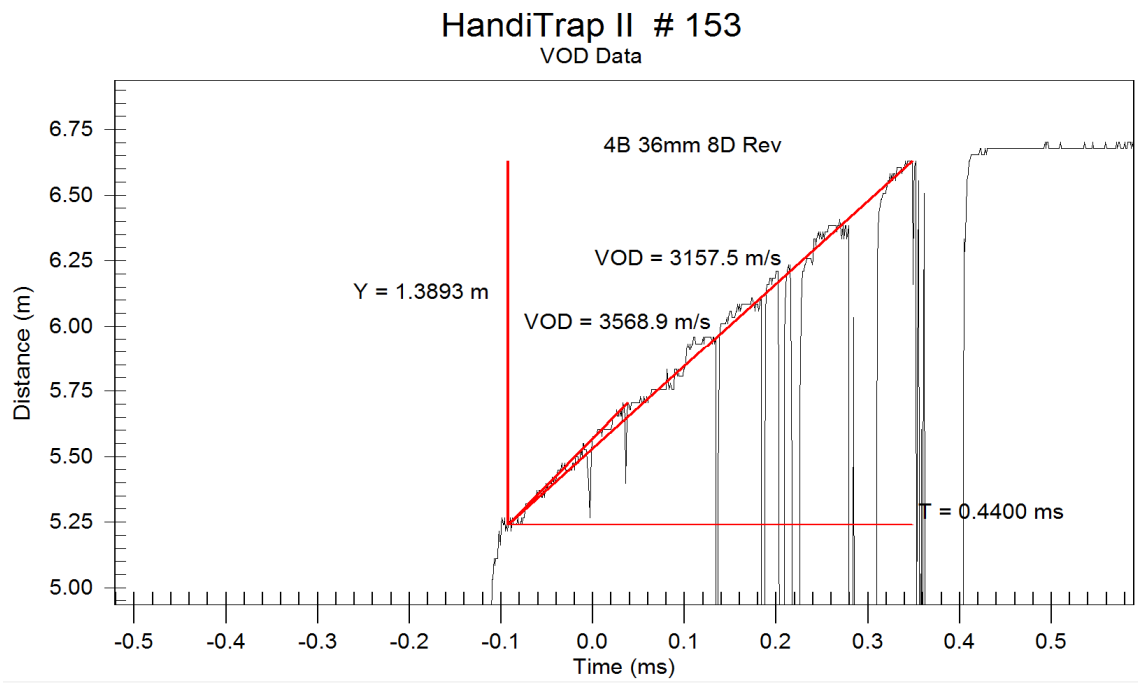
Pipe 3B, VOD measured inside 45.1mm steel pipe - Forward toe primed



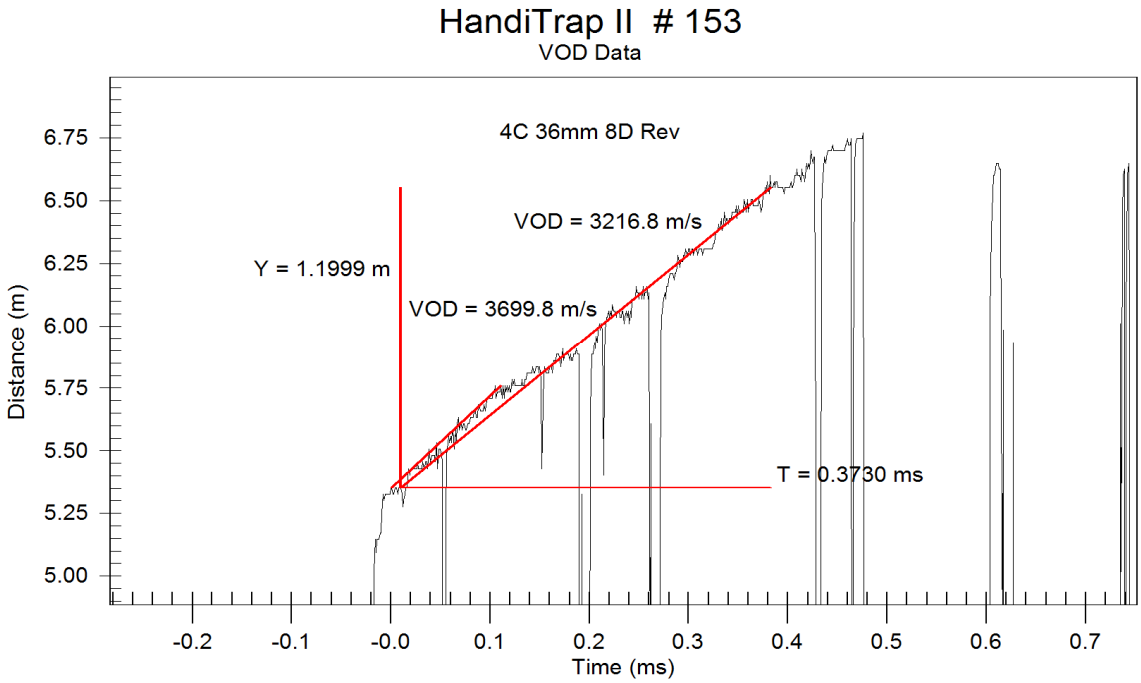
Pipe 4A, VOD measured inside 37.1mm steel pipe - Reverse toe primed



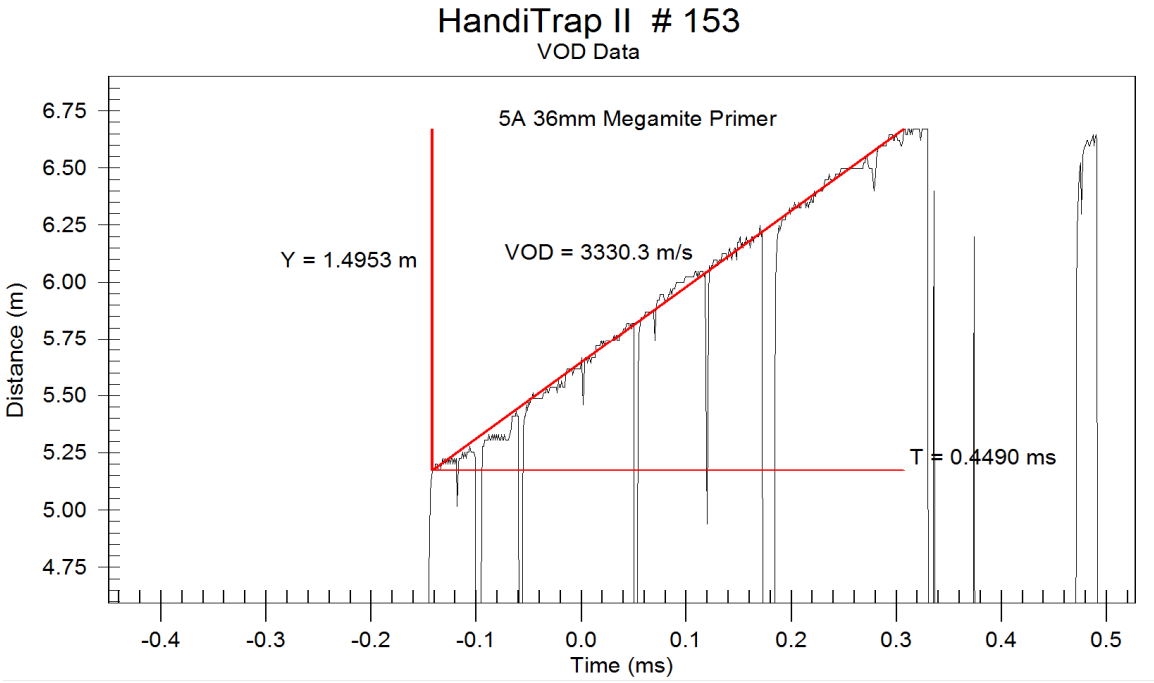
Pipe 4B, VOD measured inside 37.1mm steel pipe - Reverse toe primed



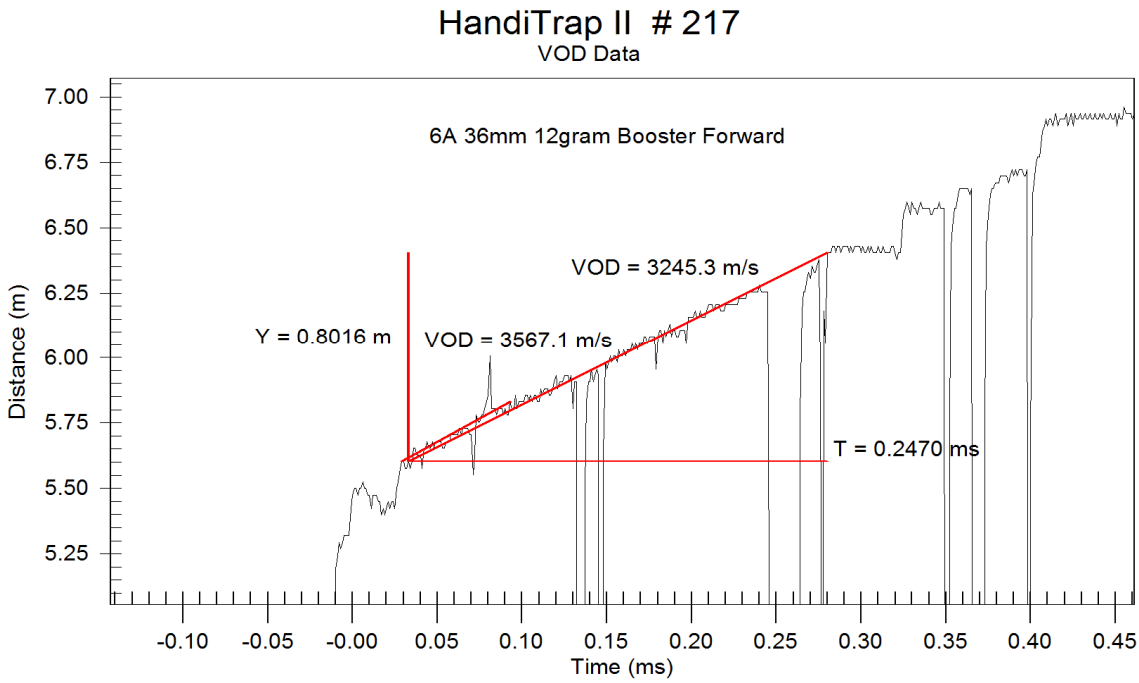
Pipe 4C, VOD measured inside 37.1mm steel pipe - Reverse toe primed



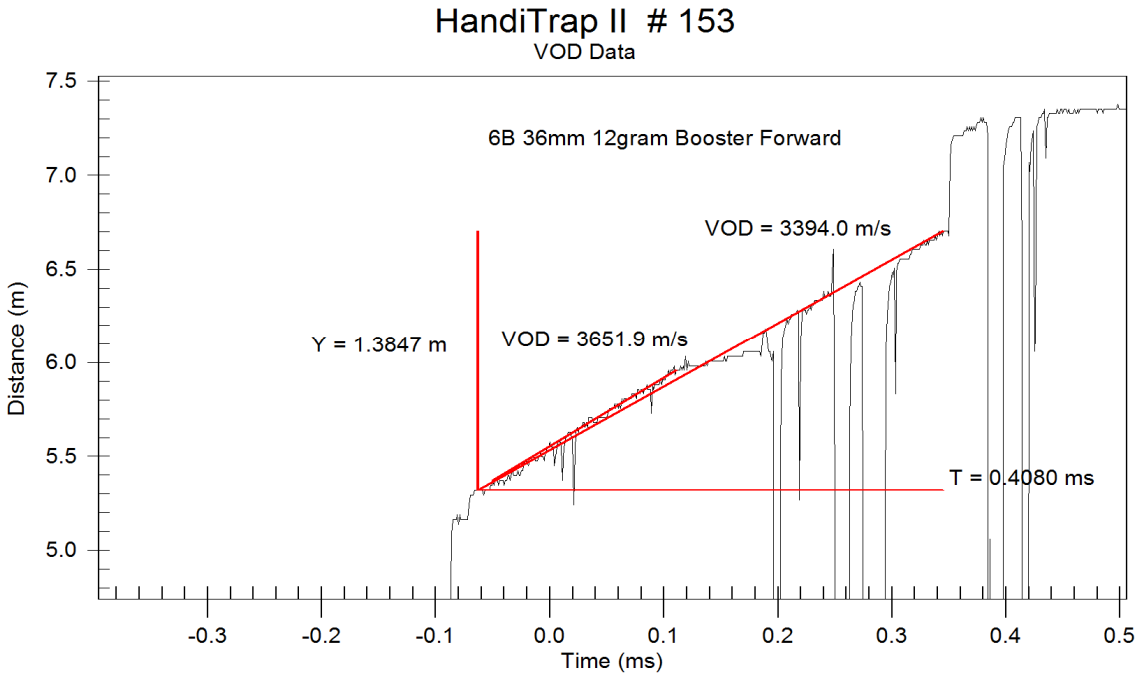
Pipe 5A, VOD measured inside 37.1mm steel pipe – Forward toe Cartridge M primer



Pipe 6A, VOD measured inside 37.1mm steel pipe – Forward toe 12 gram booster primed



Pipe 6B, VOD measured inside 37.1mm steel pipe – Forward toe 12 gram booster primed



Appendix 6, Blast Audit Template

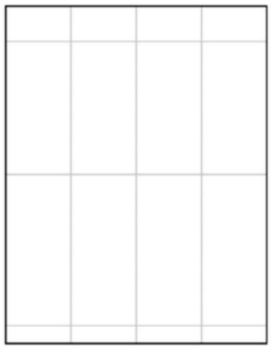
Mine: _____ Shiftboss: _____

Date: _____ Miner: _____

Workplace: _____ BME Rep.: _____

Blast Design - Gully

Gully Width:	m	Cut Size:	cm x cm
Gully Height:	m	Marked Holes:	
Burden (↔):		Drilled Holes:	
Spacing (↕):		Hole Depth (Measured):	m



Blast Results

Peg to Face Before:	
Peg to Face After:	
Face Advance:	
Socket Length:	
Socket Spacing:	

Hanging Wall Condition & Overbreak

Clear Parting Plane	☐	Limited Over-break	☐
Fractured / Blocky Hanging	☐	Moderate Over-break	☐
Highly Fractured Hanging	☐	Excessive Over-break	☐

Charging of Face

Gassing 1 + ____	Time	Sample 1	Sample 2
0 min (= 400g)	____:____		
15 min (< 345g)	____:____		
30 min (< 330g)	____:____		
45 min (< 320g)	____:____		

Density - Mass	
--- 1.5 - (400g)----	
--- 1.4 - (375g)----	
--- 1.3 - (350g)----	High
--- 1.2 - (320g)----	
--- 1.1 - (295g)----	Blasting Window
--- 1.0 - (270g)----	
--- 0.9 - (245g)----	Low

(Std. Cup 265ml, 5g)

Holes Desludged:	Y	N

Explosives Used: _____ Time of charging first hole: ____:____

Initiation & Primer: _____ Time of charging last hole: ____:____

Method of Charging: _____ Time of lighting face: ____:____

Tamping Used: _____ Stay-a-lites: X minutes

Emulsion Bags Used: _____ Pump No. _____

Charging Comments: _____

Photos: (Date Stamp ☐ **, Method of Charging ☐ , Explosives Used ☐ , Initiation & Primer ☐ , Timing Sequence ☐ & Free Breaking Face ☐)**

Blast Design - Panel Gully on: Left Right Down Dip: On Left On Right Level

Panel Length:	m	Burden (↔):		Total Holes (panel):		Hole Diameter - Initial:	mm	Discard:	mm
Panel Height:	m	Spacing (↕):		Depth of Holes:	m	Face Profile/shape:			

Direction & Dip of Holes: _____

General Comments: _____

Photos: **(**Date Stamp**, Fragmentation ☐ , Burden & Spacing ☐ , Direction & Dip of Holes ☐ , Consistency of Hole Length ☐ , Drilling Design ☐ , Free Breaking Point ☐ , Socket Direction & Spacing ☐ , Hanging Wall Condition ☐ ,Footwall Condition ☐ , Pre-conditioning ☐)**

Appendix 7, Rustenburg Minerals Blast Results – N3 Raise, Emulsion B

Blast No.	Raise Line	Explosive	Priming Practice	Time Started Charging	Time Ended Charging	Blasting Time	Charging Duration (min)	Emulsion Gassing Time (min)	Raise Width (m)	Raise Height (m)	Nominal Blasthole Size (mm)	Blastholes per Face	Mass per Hole (kg)	Mass per Face (kg)	Drilled Length (m)	Advance (m)	% Advance	Socket Length (m)	Design Powder Factor (kg/m^3)	Achieved Powder Factor (kg/m^3)	Design Energy Factor (kg/m^3)	Achieved Energy Factor (kg/m^3)	Charging Rate (Kg/min)
Round Design									1.4	1.8	36	22	1.2	26.4	1.5	1.5	100%	-	7.0		6.4		
1	N3	Emulsion B	8D	14:35	14:54	15:20	0:19	0:26	1.2	2.3	36	22	1.2	26.4	1.6	1.2	75%	0.4	6.0	8.0	5.5	7.3	1.39
2	N3	Emulsion B	8D	10:40	11:20	13:00	0:40	1:40	1.5	2.0	36	20	1.2	24.0	1.6	1.4	88%	0.2	5.0	5.7	4.6	5.3	0.60
3	N3	Emulsion B	8D	14:08	14:42	15:20	0:34	0:38	1.5	1.9	36	20	1.2	24.0	1.6	1.5	94%	0.1	5.3	5.6	4.8	5.2	0.71
4	*N3*	*Emulsion B*	*8D*	*13:15*	*13:37*	*15:00*	*0:22*	*1:23*	*1.4*	*2.0*	*36*	*24*	*0.8*	*19.2*	*0.8*	*0.8*	*100%*	*0*	*8.6*	*8.6*	*7.9*	*7.9*	*0.87*
5	N3	Emulsion B	8D	13:55	14:08	15:00	0:13	0:52	1.5	2.0	36	24	1.2	28.8	1.6	1.6	100%	0	6.0	6.0	5.5	5.5	2.22
6	N3	Emulsion B	8D	13:05	13:30	15:00	0:25	1:30	1.5	1.8	36	26	1.2	31.2	1.6	1.6	100%	0	7.2	7.2	6.6	6.6	1.25
7	N3	Emulsion B	8D	12:12	12:30	14:30	0:18	2:00	1.3	1.8	36	22	1.2	26.4	1.6	1.4	88%	0.2	7.1	8.1	6.5	7.4	1.47
8	N3	Emulsion B	8D	14:48	15:40	16:10	0:52	0:30	1.4	1.9	36	22	1.2	26.4	1.6	1.5	94%	0.1	6.2	6.6	5.7	6.1	0.51
9	N3	Emulsion B	8D	13:47	14:20	14:35	0:33	0:15	1.2	2.0	36	24	1.2	28.8	1.5	1	67%	0.5	8.0	12.0	7.4	11.0	0.87
10	N3	Emulsion B	8D	12:46	13:07	14:45	0:21	1:38	1.2	1.9	36	22	1.2	26.4	1.6	1.4	88%	0.2	7.2	8.3	6.7	7.6	1.76
11	N3	Emulsion B	8D	14:04	14:20	15:30	0:16	1:10	1.2	1.9	36	22	1.2	26.4	1.5	1.3	87%	0.2	7.7	8.9	7.1	8.2	1.65
12	N3	Emulsion B	8D	14:35	14:50	15:30	0:15	0:40	1.4	2.0	36	23	1.2	27.6	1.6	1.6	100%	0	6.2	6.2	5.7	5.7	1.84

Blast No.	Raise Line	Explosive	Priming Practice	Time Started Charging	Time Ended Charging	Blasting Time	Charging Duration (min)	Emulsion Gassing Time (min)	Raise Width (m)	Raise Height (m)	Nominal Blasthole Size (mm)	Blastholes per Face	Mass per Hole (kg)	Mass per Face (kg)	Drilled Length (m)	Advance (m)	% Advance	Socket Length (m)	Design Powder Factor (kg/m^3)	Achieved Powder Factor (kg/m^3)	Design Energy Factor (kg/m^3)	Achieved Energy Factor (kg/m^3)	Charging Rate (Kg/min)
13	N3	Emulsion B	8D	13:50	14:20	14:45	0:30	0:25	1.3	2.0	36	22	1.2	26.4	1.5	1.2	80%	0.3	6.8	8.5	6.2	7.8	0.88
14	N3	Emulsion B	8D	14:35	15:00	15:25	0:25	0:25	1.2	1.9	36	24	1.2	28.8	1.5	1.5	100%	0	8.4	8.4	7.7	7.7	1.15
15	N3	Emulsion B	8D	11:20	11:40	13:20	0:20	1:40	1.4	2.0	36	24	1.2	28.8	1.6	1.5	94%	0.1	6.4	6.9	5.9	6.3	1.44
16	N3	Emulsion B	8D	12:31	13:14	13:20	0:43	0:06	1.2	2.0	36	23	1.2	27.6	1.5	1.1	73%	0.4	7.7	10.5	7.1	9.6	0.64
17	N3	Emulsion B	8D	12:50	13:20	15:00	0:30	1:40	1.2	1.8	36	23	1.2	27.6	1.5	1.5	100%	0	8.5	8.5	7.8	7.8	0.92
18	N3	Emulsion B	8D	14:10	14:35	15:17	0:25	0:42	1.1	1.9	36	24	1.2	28.8	1.5	1.2	80%	0.3	9.2	11.5	8.5	10.6	1.15
19	N3	Emulsion B	8D	13:36	13:47	15:00	0:11	1:13	1.0	1.8	36	23	1.2	27.6	1.5	1.1	73%	0.4	10.2	13.9	9.4	12.8	2.51
20	*N3*	*Emulsion B*	*8D*	*14:32*	*14:50*	*15:50*	*0:18*	*1:00*	*1.2*	*2.1*	*36*	*23*	*0.8*	*18.4*	*1.1*	*1.1*	*100%*	*0*	*6.6*	*6.6*	*6.1*	*6.1*	*1.02*
21	*N3*	*Emulsion B*	*8D*	*14:02*	*14:15*	*15:00*	*0:13*	*0:45*	*1.0*	*1.9*	*36*	*24*	*0.8*	*19.2*	*1.2*	*1.2*	*100%*	*0*	*8.4*	*8.4*	*7.7*	*7.7*	*1.48*
22	N3	Emulsion B	8D	13:22	13:34	15:00	0:12	1:26	1.3	1.8	36	23	1.2	27.6	1.5	1.5	100%	0	7.9	7.9	7.2	7.2	2.30
23	*N3*	*Emulsion B*	*8D*	*14:09*	*14:18*	*15:00*	*0:09*	*0:42*	*1.2*	*1.8*	*36*	*24*	*0.8*	*19.2*	*1.2*	*1.2*	*100%*	*0*	*7.4*	*7.4*	*6.8*	*6.8*	*2.13*
24	N3	Emulsion B	8D	12:30	12:43	15:00	0:13	2:17	1.4	1.9	36	23	1.2	27.6	1.5	1.5	100%	0	6.9	6.9	6.4	6.4	2.12
25	*N3*	*Emulsion B*	*8D*	*13:55*	*14:14*	*15:00*	*0:19*	*0:46*	*1.4*	*1.8*	*36*	*22*	*0.8*	*17.6*	*1.2*	*1.1*	*92%*	*0.1*	*5.8*	*6.3*	*5.4*	*5.8*	*0.93*

Blast No.	Raise Line	Explosive	Priming Practice	Time Started Charging	Time Ended Charging	Blasting Time	Charging Duration (min)	Emulsion Gassing Time (min)	Raise Width (m)	Raise Height (m)	Nominal Blasthole Size (mm)	Blastholes per Face	Mass per Hole (kg)	Mass per Face (kg)	Drilled Length (m)	Advance (m)	% Advance	Socket Length (m)	Design Powder Factor (kg/m^3)	Achieved Powder Factor (kg/m^3)	Design Energy Factor (kg/m^3)	Achieved Energy Factor (kg/m^3)	Charging Rate (Kg/min)
26	N3	Emulsion B	8D	12:05	12:22	13:30	0:17	1:08	1.1	1.7	36	15	1.2	18.0	1.5	1.3	87%	0.2	6.4	7.4	5.9	6.8	1.06
27	N3	Emulsion B	8D & Stay-a-lite	13:30	13:40	15:00	0:10	1:20	1.3	1.9	36	22	0.8	17.6	1.3	1.3	100%	0	5.5	5.5	5.0	5.0	1.76
28	N3	Emulsion B	8D & Stay-a-lite	14:30	14:48	15:15	0:18	0:27	1.4	1.7	36	22	1.2	26.4	1.5	1.5	100%	0	7.4	7.4	6.8	6.8	1.47
29	N3	Emulsion B	8D & Stay-a-lite	13:40	14:11	15:00	0:31	0:49	1.3	1.8	36	22	1.2	26.4	1.5	1.4	93%	0.1	7.5	8.1	6.9	7.4	0.85
30	N3	Emulsion B	8D & Stay-a-lite	12:40	12:55	15:00	0:15	2:05	1.2	2.0	36	24	1.2	28.8	1.5	1.5	100%	0	8.0	8.0	7.4	7.4	1.92
31	*N3*	*Emulsion B*	*8D & Stay-a-lite*	*13:48*	*14:00*	*15:00*	*0:12*	*1:00*	*1.4*	*1.8*	*36*	*22*	*0.8*	*17.6*	*1.2*	*1.2*	*100%*	*0*	*5.8*	*5.8*	*5.4*	*5.4*	*1.47*
32	N3	Emulsion B	8D & Stay-a-lite	12:35	12:44	15:00	0:09	2:16	1.2	1.8	36	22	1.2	26.4	1.4	1.4	100%	0	8.7	8.7	8.0	8.0	2.93
33	N3	Emulsion B	8D & Stay-a-lite	13:18	13:36	15:00	0:18	1:24	1.3	1.9	36	23	1.2	27.6	1.5	1.5	100%	0	7.4	7.4	6.9	6.9	1.53
34	*N3*	*Emulsion B*	*8D & Stay-a-lite*	*11:00*	*11:20*	*15:00*	*0:20*	*3:40*	*1.2*	*1.8*	*36*	*22*	*0.8*	*17.6*	*1.2*	*1.2*	*100%*	*0*	*6.8*	*6.8*	*6.2*	*6.2*	*0.88*
35	N3	Emulsion B	8D & Stay-a-lite	13:48	14:17	15:00	0:29	0:43	1.2	2.0	36	24	1.2	28.8	1.5	1.5	100%	0	8.0	8.0	7.4	7.4	1.52
36	N3	Emulsion B	8D & Stay-a-lite	11:48	12:13	15:00	0:25	2:47	1.2	1.8	36	22	1.2	26.4	1.5	1.5	100%	0	8.1	8.1	7.5	7.5	1.06
37	N3	Emulsion B	8D & Stay-a-lite	15:15	15:31	16:00	0:16	0:29	1.3	1.9	36	22	1.2	26.4	1.5	1.4	93%	0.1	7.1	7.6	6.6	7.0	1.65
38	N3	Emulsion B	8D & Stay-a-lite	14:25	14:40	15:30	0:15	0:50	1.2	1.8	36	22	1.2	26.4	1.4	1.4	100%	0	8.7	8.7	8.0	8.0	1.76

Blast No.	Raise Line	Explosive	Priming Practice	Time Started Charging	Time Ended Charging	Blasting Time	Charging Duration (min)	Emulsion Gassing Time (min)	Raise Width (m)	Raise Height (m)	Nominal Blasthole Size (mm)	Blastholes per Face	Mass per Hole (kg)	Mass per Face (kg)	Drilled Length (m)	Advance (m)	% Advance	Socket Length (m)	Design Powder Factor (kg/m^3)	Achieved Powder Factor (kg/m^3)	Design Energy Factor (kg/m^3)	Achieved Energy Factor (kg/m^3)	Charging Rate (Kg/min)
39	N3	Emulsion B	8D & Stay-a-lite	15:35	15:50	16:20	0:15	0:30	1.4	1.8	36	22	1.2	26.4	1.5	1.4	93%	0.1	7.0	7.5	6.4	6.9	1.76
40	N3	Emulsion B	8D & Stay-a-lite	13:30	13:46	15:00	0:16	1:14	1.3	1.7	36	22	1.2	26.4	1.5	1.4	93%	0.1	8.0	8.5	7.3	7.9	1.65
41	N3	Emulsion B	8D & Stay-a-lite	13:35	13:57	14:30	0:22	0:33	1.2	1.8	36	23	1.2	27.6	1.5	1.5	100%	0	8.5	8.5	7.8	7.8	1.25
Average				13:28	13:49	14:57	0:21	1:08	1.28	1.89	36	22.5	1.12	25.2	1.45	1.35	93%	0.1	7.3	7.9	6.7	7.3	1.42

Blasts 4, 20, 21, 23, 25, 31 & 34, marked in italics were undertaken with a holes length of only 0.8 to 1.2 metres in length.

Incorrect or substandard blasting practices:

Blast 1 – Inadequate time for sensitisation of Emulsion B prior to blasting, 26 minutes (no stay-a-lite used)

Blast 9 – Inadequate time for sensitisation of Emulsion B prior to blasting, 15 minutes (no stay-a-lite used)

Blast 13 – Inadequate time for sensitisation of Emulsion B prior to blasting, 25 minutes (no stay-a-lite used)

Blast 16 – Inadequate time for sensitisation of Emulsion B prior to blasting, 6 minutes (no stay-a-lite used)

Blast 19 – Abnormally low air pressure resulting in poor mixing and slow gassing rate.

Blast 26 – Sub-standard blasting practices, only 15 holes drilled in face.

Appendix 8, Rustenburg Minerals Blast Results – Panel South 3, Cartridge L & Emulsion B

Blast Number	Panel	Explosive	RWS	Priming Practice	Panel Length (m)	Panel Height (m)	Nominal Blasthole Size (mm)	Rows of Holes	Drilling Angle (Degrees)	Burden (m)	Spacing (m)	Blastholes per Face	Mass per Blasthole (kg)	*Mass per Blast (Kg)*	Drilled Length (m)	Advance (m)	% Advance	Socket Length (m)	Design Powder Factor (kg/m^3)	Design Energy Factor (kg/m^3)	Achieved Powder Factor (kg/m^3)	Achieved Energy Factor (kg/m^3)	Charging Time (min)	Loading Rate Kg/min
Cartridge L, RMDC S3																								
1	P3S	Cartridge L	0.94	8D	11.0	1.2	36	2	90° ± 10°	41	70	54	1.0	54	1.2	0.84	70%	0.36	3.41	3.20	4.87	4.58	-	-
2	P3S	Cartridge L	0.94	8D	11.0	1.2	36	2	90° ± 10°	40	60	55	1.0	55	1.2	0.82	68%	0.38	3.47	3.26	5.08	4.78	-	-
3	P3S	Cartridge L	0.94	8D	11.0	1.2	36	2	90° ± 10°	39	60	57	1.0	57	1.2	1.1	88%	0.2	3.60	3.38	4.11	3.87	-	-
4	P3S	Cartridge L	0.94	8D	11.0	1.2	36	2	90° ± 10°	38	60	58	1.0	58	1.2	1.2	100%	0	3.66	3.44	3.66	3.44	-	-
5	P3S	Cartridge L	0.94	8D	5.5	1.3	36	2	90° ± 10°	41	60	27	1.0	27	1.2	1.2	100%	0	3.15	2.96	3.15	2.96	-	-
6	P3S	Cartridge L	0.94	8D	11.4	1.5	36	2	90° ± 10°	42	60	54	1.0	54	1.2	1.2	100%	0	2.63	2.47	2.63	2.47	-	-
7	P3S	Cartridge L	0.94	8D	11.7	1.4	36	2	90° ± 10°	41	60	57	1.0	57	1.2	1.2	100%	0	2.90	2.73	2.90	2.73	-	-
8	P3S	Cartridge L	0.94	8D	11.7	1.4	36	2	90° ± 10°	43	60	55	1.0	55	1.2	1.2	100%	0	2.80	2.63	2.80	2.63	-	-
9	P3S	Cartridge L	0.94	8D	11.7	1.4	36	2	90° ± 10°	41	60	57	0.4	23	0.5	0.5	100%	0	2.78	2.62	2.78	2.62	-	-
10	P3S	Cartridge L	0.94	8D	11.7	1.4	36	2	90° ± 10°	43	60	55	0.6	33	0.6	0.6	100%	0	3.36	3.16	3.36	3.16	-	-
Average					10.8	1.3	36	2	90° ± 10°	41	61	53	0.9	47	1.1	1.0	93%	0.1	3.2	3.0	3.5	3.3	-	-

Blast Number	Panel	Explosive	RWS	Priming Practice	Panel Length (m)	Panel Height (m)	Nominal Blasthole Size (mm)	Rows of Holes	Drilling Angle (Degrees)	Burden (m)	Spacing (m)	Blastholes per Face	Mass per Blasthole (kg)	*Mass per Blast (Kg)*	Drilled Length (m)	Advance (m)	% Advance	Socket Length (m)	Design Powder Factor (kg/m^3)	Design Energy Factor (kg/m^3)	Achieved Powder Factor (kg/m^3)	Achieved Energy Factor (kg/m^3)	Charging Time (min)	Loading Rate Kg/min
Emulsion B – 8D, No primer, RMDC S3																								
1	P3S	Emulsion B	0.92	8D	8.4	1	36	2	90° ± 10°	40	60	42	0.8	34	1.2	1.2	100%	0	3.33	3.07	3.33	3.07	57	0.59
2	P3S	Emulsion B	0.92	8D	8.4	1.1	36	2	90° ± 10°	47	60	36	0.8	29	1.2	0.6	50%	0.6	2.60	2.39	5.19	4.78	30	0.96
3	P3S	Emulsion B	0.92	8D	10.7	1.3	36	2	90° ± 10°	44	60	49	0.6	29	0.9	0.9	100%	0	2.35	2.16	2.35	2.16	25	1.18
4	P3S	Emulsion B	0.92	8D	10.7	1.3	36	2	90° ± 10°	44	60	49	0.6	29	0.9	0.9	100%	0	2.35	2.16	2.35	2.16	25	1.18
5	P3S	Emulsion B	0.92	8D	12.4	1.2	36	2	90° ± 10°	49	50	51	0.8	41	0.6	0.6	100%	0	4.57	4.20	4.57	4.20	30	1.36
6	P3S	Emulsion B	0.92	8D	12.4	1.2	36	2	90° ± 10°	46	60	54	0.8	43	1.2	1.2	100%	0	2.42	2.23	2.42	2.23	20	2.16
7	P3S	Emulsion B	0.92	8D	12.4	1.2	36	2	90° ± 10°	39	60	63	0.8	50	1.2	1.2	100%	0	2.82	2.60	2.82	2.60	47	1.07
8	P3S	Emulsion B	0.92	8D	12.4	1.2	36	2	90° ± 10°	37	60	67	0.8	54	1.2	1.1	92%	0.1	3.00	2.76	3.27	3.01	40	1.34
9	P3S	Emulsion B	0.92	8D	12.4	1.2	36	2	90° ± 10°	36	60	68	0.8	54	1.2	1.2	100%	0	3.05	2.80	3.05	2.80	32	1.70
10	P3S	Emulsion B	0.92	8D	11.5	1.2	36	2	90° ± 10°	35	60	65	0.8	52	1.2	1.1	92%	0.1	3.14	2.89	3.43	3.15	30	1.73
11	P3S	Emulsion B	0.92	8D	11.5	1.2	36	2	90° ± 10°	46	50	50	0.8	40	1.2	0.9	75%	0.3	2.42	2.22	3.22	2.96	43	0.93
12	P3S	Emulsion B	0.92	8D	11.5	1.2	36	2	90° ± 10°	45	50	51	0.8	41	1.2	0.9	75%	0.3	2.46	2.27	3.29	3.02	45	0.91
13	P3S	Emulsion B	0.92	8D	11.5	1.5	36	2	90° ± 10°	48	60	48	0.8	38	1.5	1.26	84%	0.24	1.48	1.37	1.77	1.63	35	1.10

Blast Number	Panel	Explosive	RWS	Priming Practice	Panel Length (m)	Panel Height (m)	Nominal Blasthole Size (mm)	Rows of Holes	Drilling Angle (Degrees)	Burden (m)	Spacing (m)	Blastholes per Face	Mass per Blasthole (kg)	*Mass per Blast (Kg)*	Drilled Length (m)	Advance (m)	% Advance	Socket Length (m)	Design Powder Factor (kg/m^3)	Design Energy Factor (kg/m^3)	Achieved Powder Factor (kg/m^3)	Achieved Energy Factor (kg/m^3)	Charging Time (min)	Loading Rate Kg/min
14	P3S	Emulsion B	0.92	8D	11.0	1.2	36	2	90° ± 10°	36	60	61	0.8	49	1.2	0.8	67%	0.4	3.08	2.83	4.62	4.25	27	1.81
Average					11.2	1.2	36	2	90° ± 10°	42	58	54	0.8	42	1.1	1.0	88%	0.1	2.8	2.6	3.3	3.0	35	1.3
Emulsion B – 8D, Cartridge L Primer, RMDC S3																								
15	P3S	Emulsion B	0.92	Cartridge L	11.4	1.3	36	2	90° ± 10°	39	60	58	0.8	46	1.2	1.2	100%	0	2.61	2.40	2.61	2.40	40	1.16
16	P3S	Emulsion B	0.92	Cartridge L	11.4	1.3	36	2	90° ± 10°	40	60	57	0.8	46	1.2	1.2	100%	0	2.56	2.36	2.56	2.36	15	3.04
17	P3S	Emulsion B	0.92	Cartridge L	11.4	1.3	36	2	90° ± 10°	39	60	59	0.4	24	0.6	0.6	100%	0	2.65	2.44	2.65	2.44	15	1.57
18	P3S	Emulsion B	0.92	Cartridge L	11.7	1.4	36	2	90° ± 10°	40	60	59	0.4	24	0.6	0.6	100%	0	2.40	2.21	2.40	2.21	20	1.18
19	P3S	Emulsion B	0.92	Cartridge L	11.7	1.3	36	2	90° ± 10°	40	60	59	0.8	47	1.2	1.2	100%	0	2.59	2.38	2.59	2.38	22	2.15
20	P3S	Emulsion B	0.92	Cartridge L	12.0	1.5	36	2	90° ± 10°	43	65	56	0.8	45	1.2	1.2	100%	0	2.07	1.91	2.07	1.91	28	1.60
21	P3S	Emulsion B	0.92	Cartridge L	12.0	1.5	36	2	90° ± 10°	44	70	54	0.8	43	1.2	1.2	100%	0	2.00	1.84	2.00	1.84	22	1.96
Average					11.7	1.4	36	2	90° ± 10°	41	62	57	0.7	39	1.0	1.0	100%	0.1	2.4	2.2	2.4	2.2	23	1.8

Appendix 9, Rand Uranium Blast Results – 85 Level, UE1A, 6 North, Cartridge H & Emulsion B

Blast Number	Panel	Explosive	RWS	Priming Practice	Panel Length (m)	Panel Height (m)	Nominal Blasthole Diameter (mm)	Rows of Holes	Drilling Angle (Degrees)	Burden (cm)	Spacing (cm)	Blastholes per Face	Mass per Blasthole (kg)	Mass per Blast (Kg)	Blasthole Length (m)	* Max Possible Advance (m)	Advance (m)	% Advance	Socket Length (m)	Design Powder Factor (kg/m^3)	Design Energy Factor (kg/m^3)	Achieved Powder Factor (kg/m^3)	Achieved Energy Factor (kg/m^3)	Charging Time (min)	Loading Rate Kg/min
Cartridge H, Averaged Face Profile, N3																									
1	N6	Cartridge H	1.09	8D	11	1.2	36	2	70° ± 20°	59	95	37	1.2	44.4	1.2	1.13	0.56	50%	0.6	2.98	3.25	5.96	6.50	-	-
2	N6	Cartridge H	1.09	8D	11	1.2	36	2	70° ± 20°	58	90	38	1.2	45.6	1.2	1.13	0.56	50%	0.6	3.06	3.34	6.13	6.68	-	-
3	N6	Cartridge H	1.09	8D	11	1.2	36	2	70° ± 20°	63	90	35	1.2	42.0	1.2	1.13	0.75	67%	0.4	2.82	3.07	4.23	4.61	-	-
4	N6	Cartridge H	1.09	8D	11	1.1	36	2	70° ± 20°	56	85	39	1.2	46.8	1.2	1.13	0.66	58%	0.5	3.43	3.74	5.88	6.41	-	-
Average					11	1.2	36	2	70° ± 20°	59	90	37	1.2	44.7	1.2	1.13	0.63	0.56	0.53	3.07	3.35	5.55	6.05	-	-
Cartridge H, N3																									
5	N6	Cartridge H	1.09	8D	11	1.2	36	2	70° ± 20°	51	70	43	1.2	51.6	1.2	1.13	0.80	71%	0.35	3.47	3.78	4.89	5.33	-	-
6	N6	Cartridge H	1.09	8D	11	1.2	36	2	70° ± 20°	56	90	39	1.2	46.8	1.2	1.13	0.75	67%	0.4	3.14	3.43	4.71	5.14	-	-
7	N6	Cartridge H	1.09	8D	11	1.3	36	2	70° ± 20°	63	70	35	1.2	42.0	1.2	1.13	0.75	67%	0.4	2.60	2.84	3.91	4.26	-	-
8	N6	Cartridge H	1.09	8D	11	1.2	36	2	70° ± 20°	58	80	38	1.2	45.6	1.2	1.13	0.89	79%	0.25	3.06	3.34	3.87	4.22	-	-
Average					11	1.2	36	2	70° ± 20°	57	77	38	1.2	46.5	1.2	1.13	0.8	0.71	0.35	3.07	3.34	4.35	4.74	-	-

* Maximum Possible Advance is the effective drilled length of the round calculated through drilled length and drilling angle

Blast Number	Panel	Explosive	RWS	Priming Practice	Panel Length (m)	Panel Height (m)	Nominal Blasthole Diameter (mm)	Rows of Holes	Drilling Angle (Degrees)	Burden (cm)	Spacing (cm)	Blastholes per Face	Mass per Blasthole (kg)	Mass per Blast (Kg)	Blasthole Length (m)	Max Possible Advance (m)	Advance (m)	% Advance	Socket Length (m)	Design Powder Factor (kg/m^3)	Design Energy Factor (kg/m^3)	Achieved Powder Factor (kg/m^3)	Achieved Energy Factor (kg/m^3)	Charging Time (min)	Loading Rate Kg/min
Emulsion B, Averaged Face Profile, N3																									
1	N6	Emulsion B	0.92	12g	11	1.3	36	2	70° ± 20°	61	60	36	1.2	43.2	1.5	1.41	0.66	47%	0.8	2.14	1.97	4.59	4.22	15	2.9
2	N6	Emulsion B	0.92	12g	11	1.2	36	2	70° ± 20°	46	65	48	0.8	38.4	1.2	1.13	0.66	58%	0.5	2.58	2.37	4.42	4.07	22	1.8
3	N6	Emulsion B	0.92	12g	11	1.1	36	2	70° ± 20°	50	80	44	1.0	44.0	1.2	1.13	0.56	50%	0.6	3.22	2.97	6.45	5.93	25	1.8
4	N6	Emulsion B	0.92	12g	11	1.1	36	2	70° ± 20°	48	75	44	1.0	44.0	1.2	1.13	0.56	50%	0.6	3.38	3.11	6.75	6.21	40	1.1
5	N6	Emulsion B	0.92	Cartridge (Not tied)	10	1.2	36	2	70° ± 20°	57	70	35	1.0	35.0	1.2	1.13	0.71	63%	0.45	2.59	2.38	4.14	3.81	20	1.8
Average					11	1.2	36	2	70° ± 20°	52	70	41	1	40.9	1.3	1.2	0.63	0.53	0.59	2.78	2.56	5.27	4.85	24	1.8
Emulsion B, N3																									
6	N6	Emulsion B	0.92	Cartridge H	10	1.1	36	2	70° ± 20°	44	70	45	1.0	45.0	1.2	1.13	0.85	75%	0.3	3.63	3.34	4.84	4.45	14	3.2
7	N6	Emulsion B	0.92	Cartridge H	10	1.1	36	2	70° ± 20°	47	90	43	1.0	43.0	1.2	1.13	0.85	75%	0.3	3.47	3.19	4.62	4.25	20	2.2
8	N6	Emulsion B	0.92	Cartridge H	10	1.1	36	2	70° ± 20°	50	70	38	1.0	38.0	1.2	1.13	0.75	67%	0.4	3.22	2.97	4.84	4.45	25	1.5
9	N6	Emulsion B	0.92	Cartridge H	8	1.1	36	2	70° ± 20°	44	80	36	1.0	36.0	1.2	1.13	1.03	92%	0.1	3.63	3.34	3.96	3.64	18	2.0
10	N6	Emulsion B	0.92	Cartridge H	9	1.1	36	2	70° ± 20°	49	75	37	1.0	37.0	1.2	1.13	0.85	75%	0.3	3.31	3.05	4.42	4.06	14	2.6
11	N6	Emulsion B	0.92	Cartridge H	8.5	1.1	36	2	70° ± 20°	45	70	38	1.0	38.0	1.2	1.13	0.94	83%	0.2	3.60	3.31	4.32	3.98	15	2.5
Average					9.2	1.1	36	2	70° ± 20°	46	76	40	1	39.5	1.2	1.13	0.88	0.78	0.27	3.48	3.2	4.5	4.14	18	2.3

Appendix 10, Rand Uranium Blast Results – 128 Level, K9, Panel 2 West, Cartridge H& Emulsion B

Blast Number	Panel	Explosive	RWS	Priming Practice	Panel Length (m)	Panel Height (m)	Nominal Hole Diameter (mm)	Rows	Drilling Angle (Degrees)	Burden (cm)	Spacing (cm) Top to Bottom	Blastholes per Face	Mass per Hole (kg)	Total Mass per Blast (kg)	Drilled Length (m)	Advance (m)	% Advance	Socket Length (m)	Powder Factor (kg/m^3)	Energy Factor (kg/m^3)	Achieved Powder Factor (kg/m^3)	Achieved Energy Factor (kg/m^3)
Cartridge H - Initial Round Design, 2W																						
1	2W	Cartridge H	1.09	8D	15	1.9	36	3	90° ± 10°	66	100	68	0.8	54	0.9	0.9	100%	0	2.12	2.31	2.12	2.31
2	2W	Cartridge H	1.09	8D	8	1.9	36	3	90° ± 10°	69	100	35	1.0	35	1.2	1.05	88%	0.2	1.92	2.09	2.19	2.39
3	2W	Cartridge H	1.09	8D	8	1.9	36	3	90° ± 10°	57	110	42	1.0	42	1.2	1.2	100%	0	2.30	2.51	2.30	2.51
4	2W	Cartridge H	1.09	8D	8	1.8	36	3	90° ± 10°	65	100	37	1.0	37	1.2	1.1	92%	0.1	2.14	2.33	2.34	2.55
5	2W	Cartridge H	1.09	8D	8	1.8	36	3	90° ± 10°	62	110	39	1.0	39	1.2	1.15	96%	0.1	2.26	2.46	2.36	2.57
Average					9	1.9	36	3	90° ± 10°	64	104	44	0.96	41	1.14	1.08	95%	0.1	2.15	2.34	2.26	2.47
Emulsion B - Initial Round Design, 2W																						
B1	2W	Emulsion B	0.92	Cartridge H	8	1.9	36	3	90° ± 10°	65	100	37	0.8	30	1.2	1.2	100%	0	1.62	1.49	1.62	1.49
B2	2W	Emulsion B	0.92	Cartridge H	8	1.8	36	3	90° ± 10°	63	110	38	0.8	30	1.2	1.2	100%	0	1.76	1.62	1.76	1.62
Average					8	1.9	36	3	90° ± 10°	64	105	38	0.8	30	1.2	1.2	100%	0	1.69	1.56	1.69	1.56

Blast Number	Panel	Explosive	RWS	Priming Practice	Panel Length (m)	Panel Height (m)	Nominal Hole Diameter (mm)	Rows	Drilling Angle (Degrees)	Burden (cm)	Spacing (cm) Top to Bottom	Blastholes per Face	Mass per Hole (kg)	Total Mass per Blast (Kg)	Drilled Length (m)	Advance (m)	% Advance	Socket Length (m)	Powder Factor (kg/m^3)	Energy Factor (kg/m^3)	Achieved Powder Factor (kg/m^3)	Achieved Energy Factor (kg/m^3)
Emulsion B - Modified Round Design, 2W																						
B3	2W	Emulsion B	0.92	Cartridge H	8	2.1	36	3	90° ± 10°	63	120	38	0.8	30	1.1	1.1	100%	0	1.65	1.51	1.65	1.51
B4	2W	Emulsion B	0.92	Cartridge H	8	2.1	36	3	90° ± 10°	56	140	43	0.8	34	1.2	1.2	100%	0	1.71	1.57	1.71	1.57
B5	2W	Emulsion B	0.92	Cartridge H	8	2.1	36	3	90° ± 10°	59	120	41	0.8	33	1.2	1.2	100%	0	1.63	1.50	1.63	1.50
Average					8	2.1	36	3	90° ± 10°	59	127	41	0.8	33	1.17	1.17	100%	0	1.66	1.53	1.66	1.53
Cartridge - Modified Round Design, 2W																						
6	2W	Cartridge H	1.09	8D	8	2.1	36	3	90° ± 10°	51	120	47	1.0	47	1.2	1.2	100%	0	2.33	2.54	2.33	2.54
7	2W	Cartridge H	1.09	8D	8	2.1	36	3	90° ± 10°	53	120	45	1.0	45	1.2	1.2	100%	0	2.23	2.43	2.23	2.43
8	2W	Cartridge H	1.09	8D	8	2.1	36	3	90° ± 10°	59	140	41	1.0	41	1.2	1.2	100%	0	2.03	2.22	2.03	2.22
9	2W	Cartridge H	1.09	8D	8	2.1	36	3	90° ± 10°	55	130	44	1.0	44	1.1	1.1	100%	0	2.38	2.60	2.38	2.60
10	2W	Cartridge H	1.09	8D	8	2.1	36	3	90° ± 10°	57	120	42	1.0	42	1.2	1.2	100%	0	2.08	2.27	2.08	2.27
Average					8	2.1	36	3	90° ± 10°	55	126	44	1	44	1.18	1.18	100%	0	2.21	2.41	2.21	2.41

Appendix 11, Data Recorded for the Calculation of Coupling Ratio with Cartridge H

Panel	Reef	Blasthole Number	Blasthole Length (cm)	Cartridges per Hole	Collar Length (cm)	Charge Length (cm)	Length / Cartridge (mm)	Mass per Cartridge (g) 29 x 270	Manufactured Density (g/cc)	Hole Diameter (mm)	Effective Charge Diameter (mm)	Coupling Ratio	Standard deviation (%)
6N	UE1A	1	119	5	18	101	20.2	200	1.17	36	32.8	83.1%	1.66%
6N	UE1A	2	113	5	13	100	20.0	200	1.17	36	33.0	84.0%	0.83%
6N	UE1A	3	113	5	18	95	19.0	200	1.17	36	33.8	88.4%	3.59%
6N	UE1A	4	115	5	18	97	19.4	200	1.17	36	33.5	86.6%	1.77%
6N	UE1A	5	121	5	12	109	21.8	200	1.17	36	31.6	77.0%	7.76%
6N	UE1A	6	114	5	17	97	19.4	200	1.17	36	33.5	86.6%	1.77%
Average							*20.0*	*200*	*1.17*	*36*	*33.0*	*84.3%*	*2.89%*
Dev.	K9	1	230	12	13	217	18.1	200	1.17	36	34.7	92.9%	8.07%
Dev.	K9	2	226	11	10	216	19.6	200	1.17	36	33.3	85.5%	0.72%
Dev.	K9	3	234	10	16	218	21.8	200	1.17	36	31.6	77.0%	7.76%
Dev.	K9	4	227	11	8	219	19.9	200	1.17	36	33.1	84.4%	0.45%
Dev.	K9	5	224	11	0	224	20.4	200	1.17	36	32.7	82.5%	2.33%
Dev.	K9	6	236	11	7	229	20.8	200	1.17	36	32.3	80.7%	4.13%
Average							*20.1*	*200*	*1.17*	*36*	*32.9*	*83.8%*	*3.91%*
2W	K9	1	93	4	12	81	20.3	200	1.17	36	32.8	82.9%	1.87%
2W	K9	2	80	4	6.5	73.5	18.4	200	1.17	36	34.4	91.4%	6.59%
2W	K9	3	72	3	8	64	21.3	200	1.17	36	31.9	78.7%	6.08%
2W	K9	4	82	4	8	74	18.5	200	1.17	36	34.3	90.8%	5.98%
2W	K9	5	91	4	10	81	20.3	200	1.17	36	32.8	82.9%	1.87%
2W	K9	6	81	4	7	74	18.5	200	1.17	36	34.3	90.8%	5.98%
Average							*20.3*	*200*	*1.17*	*36*	*33.4*	*86.3%*	*4.73%*
1E	K9	1	123	5	9	114	22.8	200	1.17	32	30.9	93.2%	0.58%
1E	K9	2	122	5	9	113	22.6	200	1.17	32	31.0	94.0%	0.25%
1E	K9	3	123	5	10	113	22.6	200	1.17	32	31.0	94.0%	0.25%
1E	K9	4	118	5	5	113	22.6	200	1.17	32	31.0	94.0%	0.25%
1E	K9	5	121	5	8	113	22.6	200	1.17	32	31.0	94.0%	0.25%
1E	K9	6	122	5	8	114	22.8	200	1.17	32	30.9	93.2%	0.58%
Average							*22.7*	*200*	*1.17*	*32*	*31.0*	*93.8%*	*0.36%*

Appendix 12, Explosive Cost Breakdown, Rand Uranium, UE1A

Total Stoping Crews	60
Number of Sections	15
Stoping Crews per section	4
Pumps per crew	1.6

Equipment Cost per Section	Units	Price/pump	Depreciation (months)	Cost per crew/month
Pumps	6.4			
Emulsion Bags	24			
Sensitiser Bags	4			
Refilling Stations	1			
Railcars	1			
Transfer Tanks	1			
				4,915

Equipment Cost per kg Emulsion	3.12

Equipment Cost per month	Units	Price/pump	Depreciation (months)	Cost per crew/month
Explosive Boxes	16	2,500	24	417
Explosive Cars	2	30,000	48	313
Railcars	2	30,000	48	313
				1,042

Equipment Cost per kg Cartridges	0.66

Equipment Cost per month	Units	Price/pump	Depreciation (months)	Cost per crew/month
Anfo Loaders	4	1,800	3	600
Explosive Boxes	16	2,500	24	417
Explosive Cars	2	30,000	48	313
Railcars	2	30,000	48	313
				1,642

Equipment Cost per kg ANFO	1.04

Workplace	**UE1A**	**UE1A**	**UE1A**
Explosive	**Emulsion B**	**Cartridge H**	**ANFO**
Stope Panel			
Panel Length (m)	30	30	30
Panel Height (m)	1.2	1.2	1.2
Drilling Angle (degrees)	70	70	70
Blasthole Length (m)	1.2	1.2	1.2
Effective Drilled Length (m)	1.13	1.13	1.13
Volume per blast (m^3)	40.6	40.6	40.6
Rock density (t/m^3)	2.7	2.7	2.7
Tonnes Broken per blast (t)	110	110	110
Burden (m)	0.6	0.6	0.6
Spacing (m)	0.6	0.6	0.6
Blastholes per row	2	2	2
Blastholes per panel	100	100	100

Explosive per hole (kg)	0.8	0.8	0.8
Explosive per panel (kg)	80	80	80
Powder Factor (kg/m^3)	2.0	2.0	2.0

Gully

Gully height (m)	1.8	1.8	1.8
Gully width (m)	1.4	1.4	1.4
Volume per blast (m^3)	3.0	3.0	3.0
Tonnes Broken per blast (t)	8.2	8.2	8.2
Blastholes per gully	23	23	23
Explosive per hole (kg)	0.8	0.8	0.8
Explosive per gully (kg)	18.4	18.4	18.4
Powder Factor (kg/m^3)	6.1	6.1	6.1

Total

Total Blastholes	123	123	123
Explosive per blast (kg)	98.4	98.4	98.4
Ave. Powder Factor (kg/m^3)	2.3	2.3	2.3

Blast per month	16	16	16
Tonnes Broken/panel/month (t)	1884	1884	1884
Explosive/panel/month (t)	1.57	1.57	1.57

Cost of Explosives (Rand)

Explosive Cost per tonne	8,500	12,000	7,000
Cost per Detonator Clip			
Cost per Detonator	11.00	11.00	11.00
Cost per Starter	22	22	22
Cost of Starter wire per Blast (15m)	18	18	18

Explosive Cost per Blast	836.40	1,180.80	688.80
Cost of Detonator Clips		-	-
Cost of Detonators per blast	1,353	1,353	1,353
Electric Starter	22	22	22
Starter wire per blast (15m)	18	18	18
Planned Explosive Cost per Blast	**2,297**	**2,574**	**2,082**
Planned Explosive Cost/tonne Broken	**19.50**	**21.85**	**17.68**
% Comparison with Cartridge	89%	100%	81%
Equipment Cost per kg Emulsion	3.12	0.66	1.04
Cost of Equipment per blast	307	65	103
System Cost per Blast	**2,604**	**2,639**	**2,184**
Planned Cost/tonne broken inc. Equipment	**22.11**	**22.41**	**18.55**
% Comparison with Cartridge	99%	100%	83%
% Overcharging/Waste of Explosive	25%	50%	50%
Explosive Cost inc. Overcharging & Waste	**2,813**	**3,229**	**2,529**
Cost/tonne Broken inc. Over charging & waste	**23.89**	**27.42**	**21.47**
% Comparison with Cartridge	87%	100%	78%

Appendix 13, Explosive Cost Breakdown, Rand Uranium, K9

Total Stoping Crews	60
Number of Sections	15
Stoping Crews per section	4
Pumps per crew	1.6

Equipment Cost per Section	Units	Price/pump	Depreciation (months)	Cost per crew/month
Pumps	6.4			
Emulsion Bags	28			
Sensitiser Bags	4			
Refilling Stations	1			
Railcars	1			
Transfer Tanks	1			
				4,989

Equipment Cost per kg Emulsion	2.60

Equipment Cost per month	Units	Price/pump	Depreciation (months)	Cost per crew/month
Explosive Boxes	16	2,500	24	417
Explosive Cars	2	30,000	48	313
Railcars	2	30,000	48	313
				1,042

Equipment Cost per kg Cartridges	0.54

Equipment Cost per month	Units	Price/pump	Depreciation (months)	Cost per crew/month
Anfo Loaders	4	1,800	3	600
Explosive Boxes	16	2,500	24	417
Explosive Cars	2	30,000	48	313
Railcars	2	30,000	48	313
				1,642

Equipment Cost per kg ANFO	0.86

Workplace	**K9**	**K9**	**K9**
Explosive	**Emulsion B**	**Cartridge H**	**ANFO**
Stope Panel			
Panel Length (m)	30	30	30
Panel Height (m)	1.9	1.9	1.9
Drilling Angle (degrees)	90	90	90
Blasthole Length (m)	1.2	1.2	1.2
Effective Drilled Length (m)	1.2	1.2	1.2
Volume per blast (m^3)	68.4	68.4	68.4
Rock density (t/m^3)	2.7	2.7	2.7
Tonnes Broken per blast (t)	185	185	185
Burden (m)	0.6	0.6	0.6
Spacing (m)	0.6	0.6	0.6

Blastholes per row	3	3	3
Blastholes per panel	150	150	150
Explosive per hole (kg)	0.8	0.8	0.8
Explosive per panel (kg)	120	120	120
Powder Factor (kg/m^3)	1.8	1.8	1.8

Total

Total Blastholes	150	150	150
Explosive per blast (kg)	120	120	120
Ave. Powder Factor (kg/m^3)	1.8	1.8	1.8

Blast per month	16	16	16
Tonnes Broken/panel/month (t)	2955	2955	2955
Explosive/panel/month (t)	1.92	1.92	1.92

Cost of Explosives (Rand)

Explosive Cost per tonne	8,500	12,000	7,000
Cost per Detonator Clip			
Cost per Detonator	11.00	11.00	11.00
Cost per Starter	22	22	22
Cost of Starter wire per Blast (15m)	18	18	18

Explosive Cost per Blast	1,020	1,440	840
Cost of Detonator Clips		-	-
Cost of Detonators per blast	1,650	1,650	1,650
Electric Starter	22	22	22
Starter wire per blast (15m)	18	18	18
Planned Explosive Cost per Blast	**2,793**	**3,130**	**2,530**
Planned Explosive Cost/tonne Broken	**15.12**	**16.95**	**13.70**
% Comparison with Cartridge	89%	100%	81%
Equipment Cost per kg Emulsion	2.60	0.54	0.86
Cost of Equipment per blast	312	65	103
System Cost per Blast	**3,104**	**3,195**	**2,633**
Planned Cost/tonne broken inc. Equipment	**16.81**	**17.30**	**14.25**
% Comparison with Cartridge	99%	100%	83%
% Overcharging/Waste of Explosive	25%	50%	50%
Explosive Cost inc. Overcharging & Waste	**3,104**	**3,555**	**3,053**
Cost/tonne Broken inc. Over charging	**16.81**	**19.25**	**16.53**
% Comparison with Cartridge	87%	100%	86%

Appendix 14, Explosive Cost Breakdown, RMDC, Raise North 3

Total Stoping Crews	60
Number of Sections	15
Stoping Crews per section	4
Pumps per crew	1.6

Equipment Cost per Section	Units	Price/pump	Depreciation (months)	Cost per crew/month
Pumps	6.4			
Emulsion Bags	12			
Sensitiser Bags	4			
Refilling Stations	1			
Railcars	1			
Transfer Tanks	1			
				4,695

Equipment Cost per kg Emulsion	10.63

Equipment Cost per month	Units	Price/pump	Depreciation (months)	Cost per crew/month
Explosive Boxes	16	2,500	24	417
Explosive Cars	2	30,000	48	313
Railcars	2	30,000	48	313
				1,042

Equipment Cost per kg Cartridges	2.36

Equipment Cost per month	Units	Price/pump	Depreciation (months)	Cost per crew/month
Anfo Loaders	4	1,800	3	600
Explosive Boxes	16	2,500	24	417
Explosive Cars	2	30,000	48	313
Railcars	2	30,000	48	313
				1,642

Equipment Cost per kg ANFO	3.72

Workplace	**Raise N3**	**Raise N3**	**Raise N3**
Explosive	**Emulsion B**	**Cartridge H**	**ANFO**
Gully			
Gully height (m)	1.8	1.8	1.8
Gully width (m)	1.4	1.4	1.4
Drilled Length (m)	1.5	1.5	1.5
Volume per blast (m^3)	3.8	3.8	3.8
Rock Density (t/m^3)	3.7	3.7	3.7
Tonnes Broken per blast (t)	14	14	14
Blastholes per gully	23	23	23
Explosive per hole (kg)	1.2	1.2	1.2
Explosive per gully (kg)	27.6	27.6	27.6
Powder Factor (kg/m^3)	7.3	7.3	7.3
Total			

Total Blastholes	23	23	23
Explosive per blast (kg)	27.6	27.6	27.6
Ave. Powder Factor (kg/m^3)	7.3	7.3	7.3

Blast per month	16	16	16
Tonnes Broken/panel/month (t)	224	224	224
Explosive/panel/month (t)	0.44	0.44	0.44

Cost of Explosives (Rand)

Explosive Cost per tonne	8,500	12,000	7,000
Cost per Detonator Clip			
Cost per Detonator	11.00	11.00	11.00
Cost per Starter	22	22	22
Cost of Starter wire per Blast (15m)	18	18	18

Explosive Cost per Blast	234.60	331.20	193.20
Cost of Detonator Clips		-	-
Cost of Detonators per blast	253	253	253
Electric Starter	22	22	22
Starter wire per blast (15m)	18	18	18
Planned Explosive Cost per Blast	**540**	**624**	**486**
Planned Explosive Cost/tonne Broken	**38.63**	**44.63**	**34.76**
% Comparison with Cartridge	87%	100%	78%
Equipment Cost per kg Emulsion	10.63	2.36	3.72
Cost of Equipment per blast	293	65	103
System Cost per Blast	**834**	**689**	**589**
Planned Cost/tonne broken inc. Equipment	**59.61**	**49.29**	**42.10**
% Comparison with Cartridge	121%	100%	85%
% Overcharging/Waste of Explosive	0%	0%	0%
Explosive Cost inc. Overcharging & Waste	**834**	**689**	**589**
Cost/tonne Broken inc. Over charging & waste	**59.61**	**49.29**	**42.10**
% Comparison with Cartridge	121%	100%	85%

Appendix 15, Explosive Cost Breakdown, RMDC, Stoping Panel South 3

Total Stoping Crews	60
Number of Sections	15
Stoping Crews per section	4
Pumps per crew	1.6

Equipment Cost per Section	Units	Price/pump	Depreciation (months)	Cost per crew/month
Pumps	6.4			
Emulsion Bags	20			
Sensitiser Bags	4			
Refilling Stations	1			
Railcars	1			
Transfer Tanks	1			
				4,842

Equipment Cost per kg Emulsion	4.73

Equipment Cost per month	Units	Price/pump	Depreciation (months)	Cost per crew/month
Explosive Boxes	16	2,500	24	417
Explosive Cars	2	30,000	48	313
Railcars	2	30,000	48	313
				1,042

Equipment Cost per kg Cartridges	1.02

Equipment Cost per month	Units	Price/pump	Depreciation (months)	Cost per crew/month
Anfo Loaders	4	1,800	3	600
Explosive Boxes	16	2,500	24	417
Explosive Cars	2	30,000	48	313
Railcars	2	30,000	48	313
				1,642

Equipment Cost per kg ANFO	1.60

Workplace	**Stope S3**	**Stope S3**	**Stope S3**
Explosive	**Emulsion B**	**Cartridge H**	**ANFO**
Stope Panel			
Panel Length (m)	20	20	20
Panel Height (m)	1.2	1.2	1.2
Drilling Angle (degrees)	90	90	90
Blasthole Length (m)	1.2	1.2	1.2
Effective Drilled Length (m)	1.2	1.2	1.2
Volume per blast (m^3)	28.8	28.8	28.8
Rock density (t/m^3)	3.7	3.7	3.7
Tonnes Broken per blast (t)	107	107	107
Burden (m)	0.5	0.5	0.5
Spacing (m)	0.6	0.6	0.6
Blastholes per row	2	2	2
Blastholes per panel	80	80	80

Explosive per hole (kg)	0.8	0.8	0.8
Explosive per panel (kg)	64	64	64
Powder Factor (kg/m^3)	2.2	2.2	2.2

Total

Total Blastholes	80	80	80
Explosive per blast (kg)	64	64	64
Ave. Powder Factor (kg/m^3)	2.2	2.3	2.2

Blast per month	16	16	16
Tonnes Broken/panel/month (t)	1705	1705	1705
Explosive/panel/month (t)	1.02	1.02	1.02

Cost of Explosives (Rand)

Explosive Cost per tonne	8,500	12,000	7,000
Cost per Detonator Clip			
Cost per Detonator	11.00	11.00	11.00
Cost per Starter	22	22	22
Cost of Starter wire per Blast (15m)	18	18	18

Explosive Cost per Blast	544	768	448
Cost of Detonator Clips		-	-
Cost of Detonators per blast	880	880	880
Electric Starter	22	22	22
Starter wire per blast (15m)	18	18	18
Planned Explosive Cost per Blast	**1,508**	**1,688**	**1,368**
Planned Explosive Cost/tonne Broken	**14.15**	**15.84**	**12.84**
% Comparison with Cartridge	89%	100%	81%
Equipment Cost per kg Emulsion	4.73	1.02	1.60
Cost of Equipment per blast	303	65	103
System Cost per Blast	**1,811**	**1,753**	**1,471**
Planned Cost/tonne broken inc. Equipment	**16.99**	**16.45**	**13.80**
% Comparison with Cartridge	103%	100%	84%
% Overcharging/Waste of Explosive	0%	25%	50%
Explosive Cost inc. Overcharging & Waste	**1,811**	**1,945**	**1,695**
Cost/tonne Broken inc. Over charging & waste	**16.99**	**18.25**	**15.90**
% Comparison with Cartridge	93%	100%	87%