A PRELIMINARY EVALUATION OF A HYDRAULIC MECHANICAL SPLITTER AS A MEANS OF BREAKING ROCK IN A DEEP-LEVEL MINE

By

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Presented in fulfilment of the requirements for the degree

Master's in Engineering (Mining Engineering)

In the Faculty of Engineering, Built Environment and Information Technology

Department of Mining Engineering

University of Pretoria

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DECLARATION OF ORIGINALITY

I hereby declare that this project is my own unaided work and I have referenced all the sources I have used. It is being submitted in fulfilment of the requirements for the degree Master's in Engineering (Mining Engineering) at the University of Pretoria. It has not been submitted before for any degree or examination at any other university. This document represents my own opinion and interpretation of information received from research, interviews and practical field work. I thus accept the rules of assessment of the University and the consequences of transgressing them.

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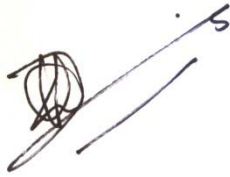
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LANGUAGE EDIT

I, <u>Linton Stanley Davies</u> hereby declare that I performed an English language edit on the final version of this project report.

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Date: 30 October 2018

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ABSTRACT

A PRELIMINARY EVALUATION OF A HYDRAULIC MECHANICAL
SPLITTER AS A MEANS OF BREAKING ROCK IN A DEEP-LEVEL MINE

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Conventional drill-and-blast practice in deep-level hard rock mining impacts negatively on the immediate environment and alternatives are frequently sought for efficient, continuous, automated and safe rock breaking. The current method for breaking rock, drilling and blasting, is a "cyclic" activity where the rock mass is drilled, blasted, cleaned and the area supported. The mining process must be completed within the blasting times. Continuous rock breaking presents the opportunity to eliminate the "cyclic or batch" mode and improve productivity. Such a system for non-explosives continuous rock breaking is the hydraulic rock splitter. The choice for the splitter is the equipment is relative simple, easy to use, readily available and affordable, and has been successfully used in the construction and civil industries.

The purpose of this study was to evaluate the functionality and applicability of the hydraulic mechanical splitter in deep-level hard rock mining. The specific instrument used in the study was the DARDA® hydraulic splitter. Rock breaking with the use of a hydraulic splitter has a place in niche applications in an underground mining operation. The static hand-held tool has distinct advantages in restricted areas. The unit is simple in design and is easily integrated into existing mining operations, and neither does it require a highly technical skilled workforce or expensive maintenance.

A literature study was undertaken, with the main focus on non-explosives rock breaking where a hole needs to be drilled into the rock mass. A device or application is inserted into the hole to fracture the rock mass. The specific DARDA® hydraulic splitter used during the trials required a hole diameter of 45mm to 48mm and a minimum hole length of 680mm.

Several trials were conducted on surface and underground. The most challenging process in mechanical rock splitting is to create a free face in the stope. In the trials four different "cut" layouts were evaluated to create a second free face. The trials highlighted the importance of quality drilling in terms of collaring the hole, hole length and directional accuracy.

The results showed the potential of the DARDA® hydraulic splitter. Drilling the least number of holes produced the least amount of rock. The greater the cross-sectional area of holes drilled, increased the amount of broken rock and resulted in easier splitting, due to the increased void. The mass of rock broken per cut varied between 30 kilograms to 65 kilograms with cross-sectional areas of 0.09m^2 and 0.144m^2 respectively.

The operational learning included the frequent lubrication of the feathers and the wedge. The unit needed to be supported during the splitting process, small rock fragments were caught between the moving parts. Fragments deep inside the "cut" area had to be removed manually and during the splitting process, starting at the hanging wall, obscured the holes close to the footwall due to rock fragments on the footwall.

To alleviate a number of operational issues experienced during the trials, include the automation of the lubrication of the feathers and wedge, supporting the cylinder unit during the splitting process and small stones wedged between the moving parts to be removed prior to inserting the splitter into the next pre-drilled hole. The cross-sectional area of the cut should be as large as possible for the rock fragments to easily fall to the foot wall and the splitting process should start from the bottom to the hanging wall to not obscure the drilled holes with rock fragments.

ACKNOWLEDGEMENTS

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LIST OF ABBREVIATIONS

ca	Centares
cm	Centimetre
CFI	Controlled Foam Injection
CSIR	Council for Scientific and Industrial Research
m^3	Cubic metres
^0C	Degree Celcius
Ø	Diameter
DMR	Department of Mineral Resources
kbar	Kilobar
Kg	Kilogram
KJ	Kilojoule
kN	Kilonewton
kW	Kilowatt
l/min	Litre per minute
m	metre
MJ	Megajoule
MJ/Kg	Megajoules per kilogram
mm	Millimetre
MPa	Megapascal
ms	Millisecond
RAP	Rapid-acting reverse-firing poppet valve
SAMERDI	South African Mining Extraction Research Development & Innovation
UCS	Uni-axial compressive strength
V	Volt

1   INTRODUCTION

1.1   Project background

For several decades research has been undertaken on developing viable alternative methods to the use of high-energy explosives for breaking rock and general mining (Haase et al., 1991; Murray et al., 1994). The motivating factors have predominantly been related to allowing continuous mining operations without interruption, reducing environmental impacts such as blast-induced ground vibrations, air blast, post-blast noxious fumes, flyrock and damage to the surroundings (Singh S.P., 1998). In underground mining operations safety considerations such as the triggering of ground falls, damage to the side and hanging walls and blast-induced seismic activities also feature prominently.

In South Africa particularly, underground mines have now reached unprecedented depths. There are numerous challenges associated with these depths, and the cost of mining has increased correspondingly. As a result, serious investigations have commenced in major mining houses looking at all aspects of modernisation, including mechanisation, automation and robotics, to meet the challenges of mining underground at great depths. Currently, drilling and blasting is the dominant method used to break rock in mining. Vogt (2016) elaborates on the cyclic nature of mining when blasting is applied in the mining cycle and a "batch" process is introduced. It includes drilling, blasting, cleaning and supporting. The mining process must be completed between blasting times. If the mining cycle activity time is reduced, time is wasted, and if it is longer the next blast is missed. In either scenario the timing cycles lead to system inefficiencies.

Drill-and-blast mining is difficult to automate, which is yet another reason for the search for a viable alternative. Continuous processes, as part of the Lean philosophy described by Womack and Jones (2003), have been identified as key to improve productivity in the mining industry.

Continuous rock breaking presents the opportunity to eliminate the "batch" mode and improve productivity. Various rock-breaking methods have been researched and trialled with various degrees of success. Reviews published in 2003 (Res et al., 2003) and 2010 (Ramezanzadeh et al., 2010), summarise progress to date on the subject.

The work conducted in this investigation was aimed at a preliminary qualitative evaluation on the use of a hydraulic splitter in deep-level underground mining and to recommend possible modifications to equipment and operating methods. There was no intention to evaluate the efficiency of the system as a method of continuous mining. Furthermore, as the purpose of the work was to conduct an initial investigation into the suitability of the equipment, no time and motion studies were carried out. Rather, the splitter is seen as a potential candidate for non-explosive rock breaking in niche applications where conventional blasting techniques are not possible or desirable. For this study, the choice of the mechanical splitter, as opposed to other non-explosive rock breaking systems, was motivated by the following: the device has been used successfully in the civil and construction industries; the systems are readily available and affordable; the application of the splitter is relatively similar to the application of drilling equipment, hence available infrastructure can be used and minimal training of personnel is needed; and the equipment is relatively simple to maintain. As described by Murray et al. (1994), two subgroups of mechanical splitters are identified, namely the wedge and feather splitter and the radial-axial splitter. The radial-axial splitter simply differs from the normal wedge and feather splitter, such as the one used in this study, by the feathers being moved outwards by a retracting rod as opposed to a protruding rod. The particular instrument used in this work was the wedge and feather splitter supplied by DARDA®. The choice of the DARDA® equipment was based solely on its commercial availability.

1.2   Problem statement

The problem statement is formulated as follows: investigate the feasibility and applicability of introducing the DARDA® hydraulic splitter into the hard rock mining environment for effective rock breaking as an alternative to breaking rock by conventional drilling and blasting methods.

1.3   Objectives

The following objectives were addressed during the research:

i.   Review technologies for non-explosive rock breaking with the emphasis on holes to be drilled.
ii.   Evaluate the DARDA® hydraulic splitter.
iii.   Evaluate proposed cut layouts.

iv.   Draw conclusions on the findings.

v.   Make recommendations on field trials and findings.

vi.   Propose areas for further work.

1.4 Methodology

A literature review was undertaken to identify methodologies for the breaking of rock without the use of explosives. The review was narrowed down to only technologies where a hole is drilled into the rock mass, a device or application is inserted into the hole and the rock mass is fractured. This would also lead to the understanding of the rate of fracturing and the successes achieved to date.

The purpose of the study was to evaluate the DARDA® hydraulic splitter. Initial thoughts were that the unit could be mounted onto an autonomous vehicle to be used underground. The DARDA® hydraulic splitter is commercially available, relative easy to operate, sufficiently robust for underground mining conditions and does not required a highly skilled labour force. Initial trials of the splitter were conducted on surface to demonstrate its applicability and ease of use. Once confidence had been gained on the applicability of the unit, trialling was carried out in the underground environment.

Several designs of hole layouts for splitting were based on conventional drilling and blasting knowledge and engineering judgement. In drilling and blasting terminology, the "cut" is a specific pattern of closely spaced blast holes charged with explosives. The main purpose of the cut is to create a second free face to improve the fragmentation and advance of the excavation. Various "cut" patterns were trialled in underground rock faces to evaluate the efficiency of each proposed hole layout regarding breakage and broken mass produced.

Conclusions were made based on the field trials and the recommendations regarding equipment and operational improvements.

Suggestions for future work were based on the results obtained and the practical application of the DARDA® hydraulic splitter.

1.5 Scope of the study

This study focused on the DARDA® hydraulic splitter for breaking rock in the deep narrow reef environment. Practical work was carried out at an outcrop on surface and in an

underground gold mine. The rock at the test site in the Alberton area, south-east of Johannesburg, was fairly weathered and the tests were not conclusive. The capabilities of the DARDA® hydraulic splitter were able to be shown. Holes were drilled in a pre-determined pattern and the rock was fractured to a desirable size range. Some lessons were learned during these trials on surface. The DARDA® hydraulic splitter was taken underground at Driefontein's no. 7 shaft at level 24, section 95. The mine has allocated this section for innovative mining methods and trialling new technologies and mining methods. All experiments using the DARDA® hydraulic splitter were conducted in this working area. A number of drill hole combinations and layouts were tried and tested. The purpose was to find the most suitable drilling pattern layout to optimise the number of holes drilled and to obtain the most effective breaking hole layout.

A few design patterns were also tried in a main drive, at Gold Field's South Shaft on level 95. Holes were drilled at 90° into the rock mass. Inserting the DARDA® hydraulic splitter experienced relative easy breaking for the first 25 cm as the rock mass was "damaged" by blasting activities. Beyond the blast damage zone the rock mass was much more competent and the breaking of the rock mass using the DARDA® hydraulic splitter became much more difficult and more challenging.

This study will assist the mining industry and associated companies who are conducting research and development of mechanical rock-breaking methods by providing guidelines to the use of the DARDA® hydraulic splitter in a production environment.

The scope of the study

1. To evaluate the DARDA® hydraulic splitter for underground rock breaking.
2. To evaluate the DARDA® hydraulic splitter in conjunction with proposed drill hole patterns.
3. There was no intention to do a time & motion study to evaluate efficiency or whether the DARDA® hydraulic splitter can be used for continuous mining.
4. To make recommendations for improvements to equipment and operational methodology.

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2   LITERATURE SURVEY

2.1   Overview of non-explosive rock-breaking systems with holes drilled

Rock mass excavation methods can be dated back to the Stone Age. The modern technique of drilling and blasting is the predominant method of rock breaking in the global mining industry. Many innovative ideas have been developed over the years to improve cost effectiveness, have control over the fragmentation size distribution and minimise the adverse effects of air blast, ground vibration, emission of noxious fumes and flyrock. New modern explosives, such as emulsions and water gels, together with precise initiation timing systems, have resulted in significant improvements in the predictability of the blast outcomes in terms of fragmentation, muckpile profile and environmental impacts.

In the context of this literature study, the definition of rock breaking is the process of separating some rock fragments from the rock mass (Res et al., 2003). Furthermore, one can say that rock breaking is the process involving energy and knowledge. The latter refers to the drilling of the holes and the layout of the hole pattern (Res et al., 2003).

There is a need for non-explosive rock-breaking methods for various reasons, including safe operations and continuity of working cycles. Explosive-free technologies and techniques were initially intended to replace secondary blasting, while the bigger picture also considered the potential of creating continuous excavation operations (Murray et al., 1994). Enormous efforts have been made in finding alternatives to breaking rock without the use of explosives. Various concepts have been tried, which include mechanical or hydraulic splitting of rock, plasma blasting, thermal heating, chemical expansion powders, and hydraulic and electro-hydraulic techniques (Res et al., 2003). Young and Graham (1999) identified mechanical, hydraulic and gas pressurisation techniques as the major attempts to develop non-explosive rock-breaking methods. Several demolition agents, adapted for mining, have been developed over the years. Conventional rock-breaking methods such as jack hammers, explosives, hydro-demolition and thermal techniques produce large levels of noise, vibration and dust, which considerably affect the environment and surrounding structures (Natanzi and Laefer, 2014).

The following rock-breaking methods require a hole to be drilled into the rock mass. To start the rock breaking, the chemical device or mechanical instrument is placed in the hole.

Controlled foam injection for rock breaking

Various forms of controlled rock-fracturing methods have been developed in the past, and these methods have been found to be efficient (Young and Graham, 1999). One of the recently developed methods is the high-pressure foam-fracturing method referred to as controlled foam injection (CFI). Pickering and Young (2017) describe foam injection as a safer, productive, environmentally friendly and well-developed method with multiple applications in rock and concrete breaking. The controlled foam injection method is applicable in hard rock excavation, breaking boulders as well as concrete structure demolition (Singh, 1998). Young (2002) classifies the controlled foam injection rock-breaking process as a continuous rock-breaking method. The method uses foam as a fracturing medium (Young and Graham, 1999) at a controlled pressure, which is obtained from a high-pressure gas using a propellant (Young and Graham, 1999). The propellant burns subsonically, which allows controlled pressure build-up (Watson and Young, 1994).

The foam medium was selected mainly due to its viscosity properties, which allow high pressure to be maintained during the fracture process (Young, 2002); furthermore, the foam is chemically inert and environmentally safe (Pickering and Young, 2017). The breakage process must be completed before the foam escapes during pressurisation and fracturing. The initial gas pressure must be in excess of 300 MPa (Young and Graham, 1999). However, excessively high-pressure foam may result in some flyrock and airblast. The advantage of using foam gases over water or steam is that gas expands more easily and quickly, and can maintain its ability to penetrate high-pressure gas into the fractures.

The foam is constituted of a combination of liquid chemicals and gas. For the successful application of the foam injection process, generating or creating foam with certain physical properties (Singh, 1998) is required. Foam is generated or created from various ratios of water chemicals and gas mixtures. The most common form of foam is a combination of water and air (Young and Graham, 1999). However, the water and air components are easily separable from each other. The separation process of water and air can be restricted or almost prevented by using surfactant additives. The stability of foam and the energy stored in the foam can be increased by adding gel to the mixture (Young, 2002; Pickering and Young, 2017). The viscosity and stored energy of the foam can be controlled or modified by the water, air and surfactant ratios (Young and Graham, 1999; Young, 2002; Pickering and Young, 2017). The controlled foam injection process can benefit from the advantages of both gas and liquid.

The CFI unit is applicable for both secondary rock breaking and solid rock excavation. The unit consists of a foam-injecting barrel which is inserted into a pre-drilled hole. The barrel must be completely sealed in the hole in order to obtain the desired results. Foam is a two-way mixture of gas and liquid which gives it a controllable level of viscosity (Young and Graham, 1999; Young, 2002; Pickering and Young, 2017). Foam has a very low escape rate from developing fractures compared to liquid and gas substances. Liquid in particular is not an ideal medium due to its relatively incompressible properties. Liquid is thus likely to lose pressure during fracture as its volume increases relative to the growth of the fractures (Young and Graham, 1999). Liquid has a high viscosity, and is relatively incompressible, which causes it to lose pressure rapidly (Young, 2002). On the other hand, gas alone has a high energy-storage capacity. The stored energy in gas propagates into fractures, resulting in an uncontrollable and violent fracturing process (Pickering and Young, 2017). Foam has the ability to maintain high pressure during fracturing due to the expansion of the gaseous phase of the fluid. Figure 1 shows how fractures develop at a slow rate by injecting foam into the rock, which allows a controllable fracturing process (Pickering and Young, 2017). These are the properties that give foam the ability to provide efficient and controlled fracturing without requiring high pressures.
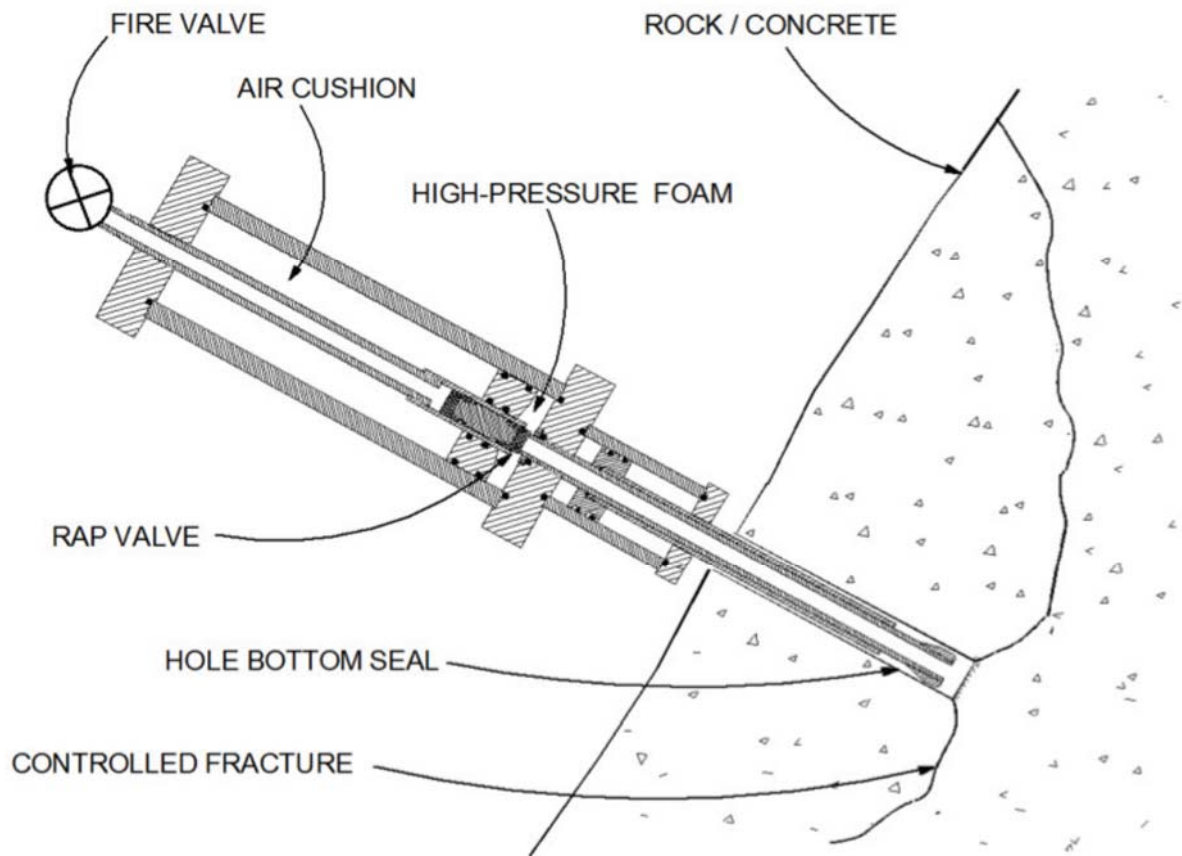
Figure 1: Controlled foam injection process for rock breaking (Pickering and Young, 2017)

Adequate foam pressure is released into the pre-drilled holes using a rapid reverse firing poppet valve as shown in Figure 2 (Young and Graham, 1999; Young, 2002). The reverse-acting poppet valve has a great influence on controlling the injection of high-pressure foam (Young and Graham, 1999; Young, 2002). The controlled foam injection method breaks rock in tension. During foam injection at the toe of the hole, the tensile strength of the rock mass is exploited by the high pressure of the foam (Pickering and Young, 2017). This allows the solid rock to break with relative ease in tension rather than against the compressive strength of the rock. The high-pressure foam is rapidly discharged at the bottom of the pre-drilled hole (Young and Graham, 1999). The controlled fracturing process of the rock begins at the toe of the hole where the foam is discharged. The high pressure of the discharged foam initiates radial fracturing and initiates the rock-breaking process. It must be ensured that sufficient foam is used during the process. Conversely, injecting excessive foam will result in a waste of energy.

Large volumes of rock and concrete can be broken and removed from the face at a very low outflow of energy (Young, 2002). The ability to maintain and control the injection of foam at

high pressure allows virtually no flyrock or airblast (Young, 2002) to be produced by this method. However, injection of excessive foam creates the possibilities of airblast and flyrock (Young and Graham, 1999). Pickering and Young (2016) explain that the CFI unit has shown that it can excavate between three to five cubic metres per hour in competent and solid rock. A 63 mm diameter CFI unit inserted into a single hole can break up to 0.5 m^3 (Pickering and Young, 2016). The CFI method has been proven and tested successfully in various types of rock with various properties. The foam is discharged into the drilled hole by means of a rapid-acting reverse-firing poppet valve (RAP) (Singh, 1998). The reverse-acting poppet is responsible for controlling the high-pressure foam injected in the drilled holes.
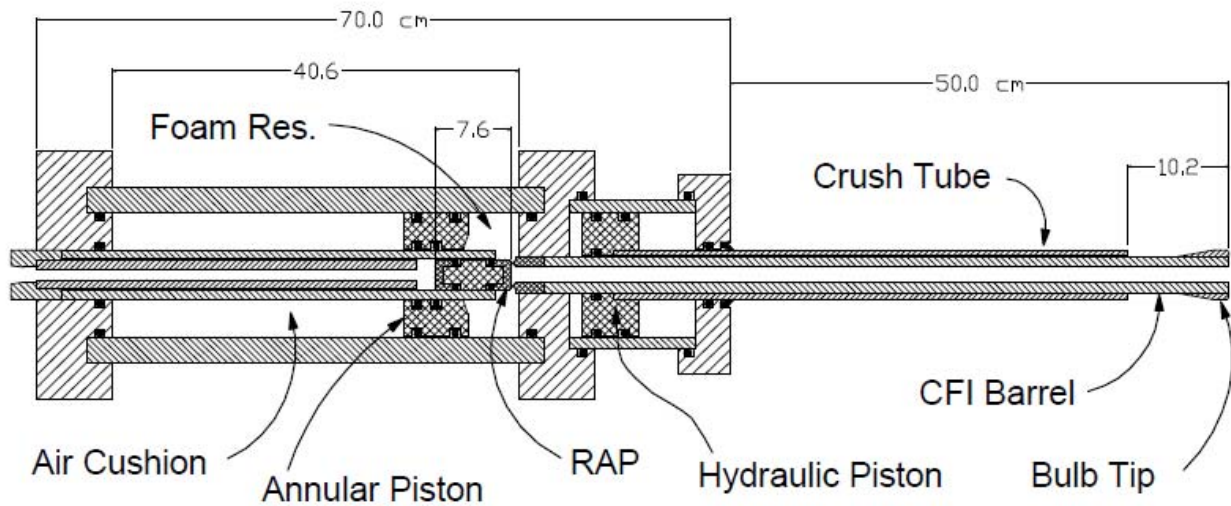
Figure 2: Foam injection tool and mechanism (Young and Graham, 1999)

Expansive chemicals for rock breaking

In recent years, expansive chemical agents have been introduced for the fragmentation and demolition of rock and concrete structures. Expansive chemical agents are commonly used in concrete structure demolition, and opencast and quarrying applications (Res et al., 2003; De Silva et al., 2016). Expansive chemical agents, also known as soundless cracking agents, expansive demolition agents or non-explosive demolition agents, provide an alternative rock and concrete fragmentation method (De Silva et al., 2016). Expansive chemical agents are non-toxic cementitious substances that expand through hydration at a particular temperature. The expansion of the agents when mixed with water occurs through the formation of ettringite ($3CaO.Al_2O_3.3CaSO_4.32H_2O$) and calcium oxide (CaO) hydration

(De Silva et al., 2016). These chemicals provide an explosive-free rock-breaking process, and are being introduced mainly in the mining, quarrying, demolition and construction industries (Gholinejad and Arshadnejad, 2012). The expansive chemical agents can be applied in both in situ rock breaking and secondary rock-breaking (Res et al., 2003). Chemical expanding agents provide a controlled fracturing process by exploiting the tensile strength of the rock. Large rocks can be fractured with no flyrock and no danger to the surrounding environment (Res et al., 2003). This method has a wide range of advantages compared to explosives as it eliminates most of the drawbacks associated with explosives blasting (De Silva et al., 2016).

Expansive chemical agents rely on their ability to expand in order to break and crush rock or concrete structures. Under confinement in a drilled hole, a volumetric expansion generates an expansive pressure that exerts a thrust force on the walls of the rock (De Silva et al., 2016). Expanding powders that consist of compound material mixed with water will, in a confined volumetric space, experience hydration and thus increase in volume by 20 to 30% (Res et al., 2003). During the expansion of the expansive chemicals in the confined space, a large pressure is slowly generated. The pressure generated during the expansion causes the initiation of the fracturing process. The pressure from the expansion causes rock stresses, resulting in the formation of radial cracks around the drilled hole when the pressure exceeds the tensile strength of the rock (De Silva et al., 2016). The method can be used in compromised conditions because it does not produce ground vibrations. The chemically expanding agents consist of a combination of substances, which include CaO (calcium oxide) and SiO_2 (silica). Other substances such as Al_2O_3 (aluminium oxide) and Fe_2O_3 (ferric oxide) are also present in very small quantities (Gholinejad and Arshadnejad, 2012). Table 1 shows a typical composition of expanding powders and the mass percentage of each substance according to Gholinejad and Arshadnejad (2012).

Table 1: The composition of expansive chemical agents (Gholinejad and Arshadnejad, 2012)

Composition of expansive chemicals (%)										
LO.I.	P_2O_5	MnO	TiO_2	K_2O	SO_3	MgO	Fe_2O_3	Al_2O_3	SiO_2	CaO
5.26	0.044	0.021	0.033	0.04	0.18	2.21	0.72	1.75	6.5	83.2

The composition of expansive chemical agents as shown by De Silva et al. (2016) are comparable to those listed by Gholinejad and Arshadnejad (2012) with minor variations. The variations are based on the manufacturers' specifications of the products. Table 2 shows the chemical mass composition of expansive chemical agents according to De Silva et al (2016).

Table 2: The composition of expansive chemical agents (De Silva et al., 2016)

Chemical component	Percentage by mass (%)
SiO_2	1.5-8.5
Al_2O_3	0.3-5.0
Fe_2O_3	0.2-3.0
CaO	81-96
MgO	0-1.6
SO_3	0.6-4.0

The chemically expanding powders or agents were discovered during the 1970s (De Silva et al., 2016) and 1980s (Natanzi and Laefer, 2014) when ettringite in cement was investigated. When sulphates and aluminates are added to the calcium oxide with pure water it causes the mixture to expand and build pressure (Natanzi and Laefer, 2014). The calcium oxide in the mixture becomes hydrated with an evolution in heat (Natanzi and Laefer, 2014; Arshadnejad et al., 2010). The chemical reaction of the expansion of the mixture is defined by the following equations (De Silva et al., 2016; Natanzi and Laefer, 2014; Gholinejad and Arshadnejad, 2012; Arshadnejad et al., 2010):

$$Equation\ 1: CaO + H_2O \rightarrow Ca(OH)_2 + 15.2\uparrow(kCal/mol)$$

$$Equation\ 2: 4CaO.3Al_2O_3.SO_3 + 8CaSO_4.2H_2O + 6Ca(OH)_2 + 74H_2O \rightarrow$$
$$3(3CaO.Al_2O_3.3CaSO_4 + 32H_2O)$$
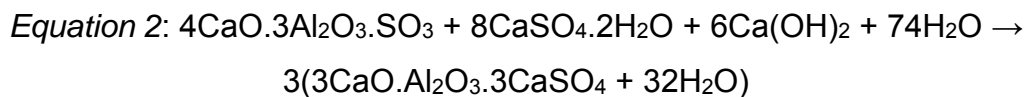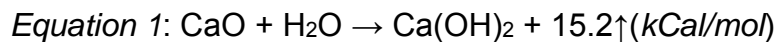
The expanding chemical agents are composed of anhydrite and calcium oxide as the main expansive additives. Various compositions of expanding agents exist, but all are constituted of a large proportion of CaO (De Silva et al., 2016). The capabilities or specifications of expansive compounds are generated by varying ratios of CaO, Al_2O_3 and SO_3 (De Silva et al., 2016). The development of pressure in expansive agents depends on the degree of hydration of the CaO substance. The dehydration of CaO produces $Ca(OH)_2$, ettringite crystals and $CaSO_4$. This leads to a gradual development of expansive pressure under

confinement. The expansion process is caused by an exothermic reaction of CaO during the process. A high concentration of CaO in the liquid phase of a wet mixture increases the rate at which ettringite forms and further contributes to the expansion volume.
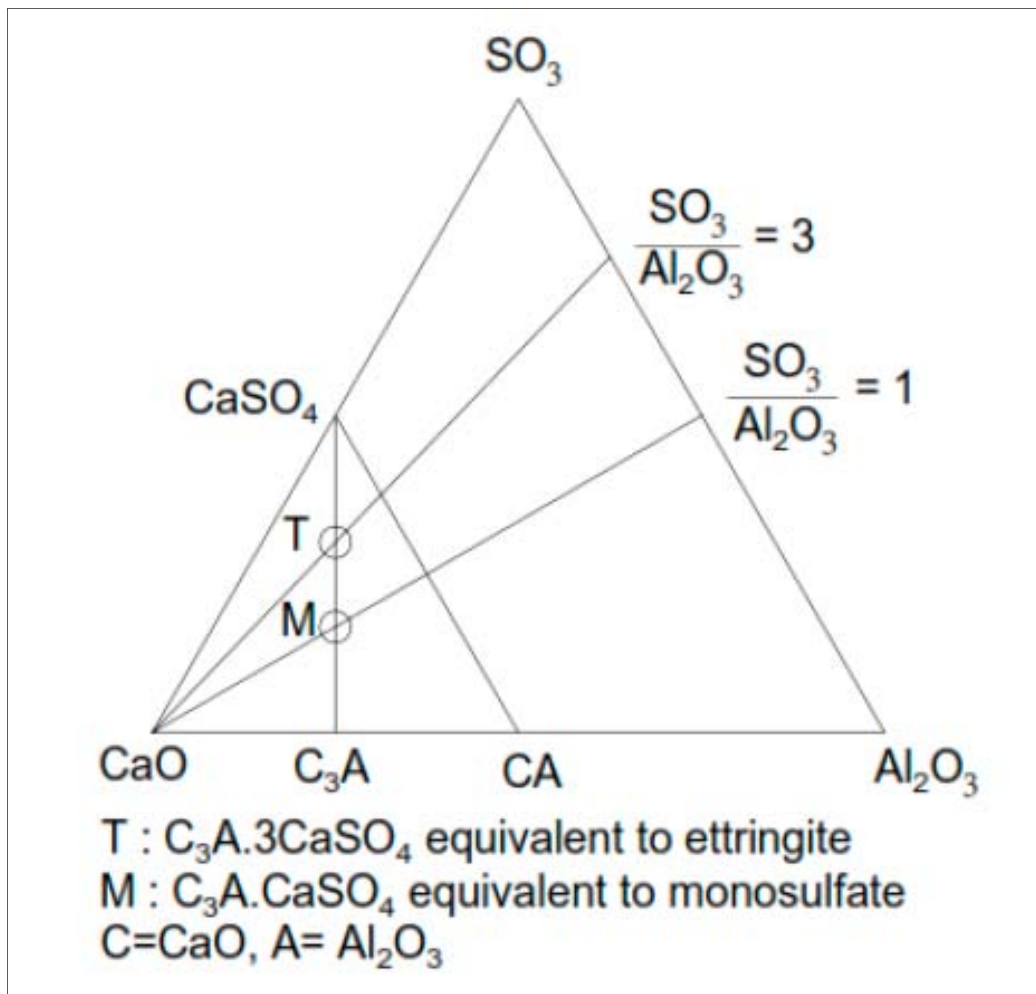
Figure 3: Chemical composition of expansive chemical agents (De Silva et al., 2016)

The material or mixture is poured into holes drilled in the rock mass that requires fracturing into small pieces. The subsequent exothermic chemical reaction results in an increase in the temperature of the compound up to 150 °C (De Silva et al., 2016). The mixture begins to expand volumetrically over a certain period of time. The expansive chemical agent begins to hydrate in the holes, generating heat and crystallising while hardening and expanding. The mixture solidifies and expansion occurs after some time. Expansion occurs under confinement, causing the expansive pressure to be generated. The walls of the rock undergo tensile force from the pressure generated by the expanding mixture (Natanzi and Laefer, 2014). The tensile force generated by the expansion initiates cracks on the walls of the rock due to stresses caused by the expansion pressure. This happens once the tensile

strength of expansion exceeds the tensile strength of the rock mass (Natanzi and Laefer, 2014; Hanif, 1997). Cracking is initiated at the collar of the drilled holes. This is where the highest stress will be experienced (Natanzi and Laefer, 2014; Hanif, 1997). Further expansion propagates into fracturing without producing noise, vibrations or flyrock (Natanzi and Laefer, 2014). Rock has a relatively low tensile strength, approximately 10% of the compressive strength, which is due to the micro-cracks within the rock mass (personal communication – Prof. F. Malan, University of Pretoria). The tensile strength of the rock varies according to the geology. The expansive thrust exerted on the walls of the holes causes a split along the line of the holes.

The vertically drilled holes are filled with a mixture of water and the chemically expanding agents. The mixture in the holes begins with time to expand in volume (Res et al., 2003). The expansion occurs at a very slow rate, and full expansion is reached after a certain period. During the expansion of the agents, expansive force from the pressure of the increase in volume is exerted on the surrounding walls of the holes. Continuously applied expansive force on the walls of the holes creates cracks and crack growth on the rock. The rock eventually fails in tension and results in a split break pattern. Xiamen Bestlink Factory Co. Ltd of China (2017) demonstrates the use of expansive agent in quarrying mining applications. See Figure 4.

Figure 4: Rock splitting by expansive powders in quarrying (Xiamen Bestlink Factory, 2017)

The expansion process differs according to the specifications of manufacturers. The duration of the fracturing process varies according to the properties of the rock mass. The crushing of the drilled holes usually takes approximately 6 – 24 hours (Res et al., 2003). The expansive strength along the hole depth is almost constant except at the collar of the hole. According to Gholinejad and Arshadnejad (2012), the expansion process generates an incremental static load around the walls of the holes, and fracture may occur after two to four hours (Gholinejad and Arshadnejad, 2012; Pal Roy, 2005). The overall expansion fracturing process is lengthy compared to other rock-breaking methods (Gholinejad and Arshadnejad, 2012). The expansive chemical agents crack the rock silently using their powder composition properties. The effectiveness of expansion of chemical agents is influenced by a number of factors, which include (Res et al., 2003):

- The ambient temperature of the surroundings
- Water content and the degree of hydration
- The diameter and depth of the pre-drilled holes
- The spacing distance between holes
- The presence of natural fracture planes in the vicinity of the holes
- Local rock-cracking properties.

The specifications of commercially available expansive chemicals are classified according to the water content, ambient temperature and drill diameter (De Silva et al., 2016). The rate of chemical reaction of the expansion process is greatly influenced by the ambient temperature. The specifications of products differ among various manufacturers. The expansion time for a crack to be initiated varies, for example approximately 8 hours at 38 $^{\circ}$C or 21 hours at 24 $^{\circ}$C (De Silva et al., 2016). For example, a Cevamit chemical agent consists of two variants. These variants were developed for applications during periods above 10 $^{\circ}$C or periods less than 10°C (Res et al., 2003). The manufacturers design their products to operate over a wide range of variable ambient temperatures, but normally between 0 and 40 $^{\circ}$C (Natanzi and Laefer, 2014). The ambient temperature dictates the time taken for the expansion to fracture the rock. For acceptable breakage of solid rock, the hole spacing should be considered. Chemical rock breaking works adequately in closely spaced holes. Tests conducted on a concrete block by the Dexpan company shows that the cracks start to form after about two hours. The formation of the cracks depends on the ambient temperature, humidity and rock mass or concrete properties. The cracks continue to grow,

widening and deepening towards adjacent holes. The cracks expand quicker when cracks from adjacent holes start to interact.

Figure 5: Cracks forming in a concrete block after about two hours (Dexpan, 2017)

The mechanical properties of the rock when subjected to expansive force depend on the extent and duration of the expansion period of the chemical reaction. The modulus of elasticity of the rock and diameter of the holes drilled and filled with the chemical agents determines the fragmentation outcomes (Gholinejad and Arshadnejad, 2012; Arshadnejad, 2010). The rock will fracture along the high-stress concentration path between two closely spaced holes (Gholinejad and Arshadnejad, 2012). Brittle materials will neither fracture nor undergo elastic deformation in hard rocks such as granite and quartzite (Gholinejad and Arshadnejad, 2012; Yagiz, 2009). Stress distribution in brittle materials remains elastic until fracturing of the rock is completed (Gholinejad and Arshadnejad, 2012). The fracturing mechanism consists of a fracture initiation phase and a fracture propagation phase when a rock is under expansive pressure. Chemical agents produce a controlled fracture process and exhibit high versatility with controlled crack formation. Optimum fragmentation results can be achieved by altering the arrangements, sizes and spacing of the drilled holes.

Chemical expansion is similar to hydraulic fracturing, which can produce fracture by splitting (De Silva et al., 2016).

According to De Silva et al (2016), the failure mode of rock under any mechanism is mainly governed by temperature and confinement pressure. Ductile failure occurs in non-porous rocks subjected to high confinement pressure. The expansive pressure developed by chemical agents inside drilled holes develops compressive stresses in the radial and tangential directions to the drilled holes. The stresses create fractures at a certain point located on the inside wall surface of the holes with minimal confinement. Fractures are initiated as an indication of a release of confinement from the rock. This results in a drop in the expansive pressure and thus a stable crack growth behaviour is experienced by the rock (De Silva et al., 2016). The fracture process will only propagate if the tensile strength applied by the expansive chemical agent is increased (De Silva et al., 2016). Stable crack propagation occurs at very low crack velocities during the expansion and fracturing process. Chemical expanding powders or agents create radial compressive and tangential stresses in the surrounding walls of the drilled holes. These stresses cause crack extension from pre-existing imperfections and micro-cracks in the rock. Crack initiation, nucleation and propagation occur from the tips of the pre-existing micro-cracks. Crack growth is accelerated by a tensile stress existing in a normal direction to the crack growth. The existence of negatively confined ratios within the rock may result in unstable crack propagation.
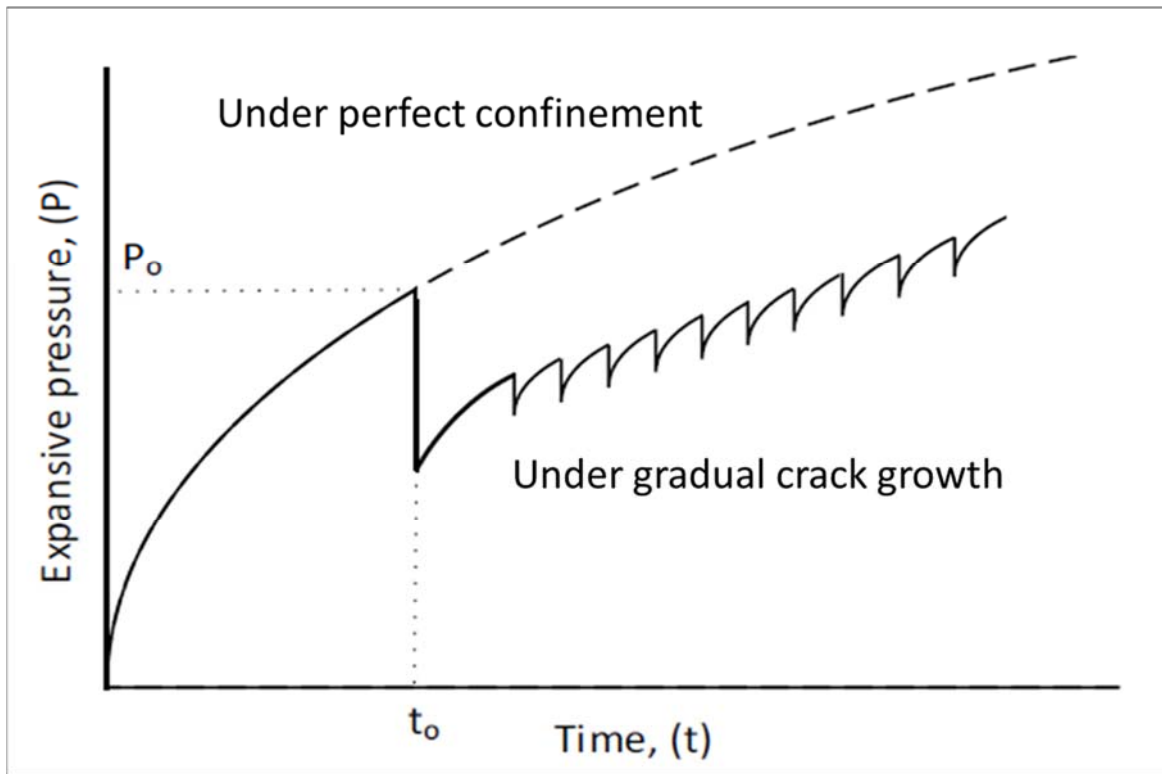
Figure 6: Stable crack growth behaviour (De Silva et al., 2016)

Expansive chemical agents are capable of fracturing any type of rock. Rocks are fractured regardless of their characteristics and properties such as strength and hardness (Res et al., 2003). The effectiveness of chemical expanding powders is influenced by the mixture placement in the holes and the diameter and depth of the holes. The spacing of holes can be adjusted to suit the varying physical and mechanical properties of the rocks (Res et al., 2003). Crackem, a manufacturer of expansive agents, suggests that hole diameters should range between 36 and 70 mm. Typical spacing between holes is 40 cm for 36 mm diameter holes. The spacing of the holes varies with the diameter of the drilled holes and affects the fragmentation results. Fragmentation may also be affected when the holes are incorrectly filled with the expansive chemicals. A slow rate of expansion ensures that the rock is fractured in a controlled manner (Res et al., 2003). Factors such as the diameter, depth and configuration of the holes have an influence on the breakage process and in ensuring that the process is controllable (Res et al., 2003). As the fracture process depends on the pressure exerted by the expansion in the hole, the pressure forms cracks in the surrounding rock mass which grow until the rock breaks (Gholinejad and Arshadnejad, 2012). Holes of equal length, diameter and centre-to-centre distance are drilled consecutively (Gholinejad and Arshadnejad, 2012). The expansive agents manufactured by Expando require a free breaking face to optimise results.
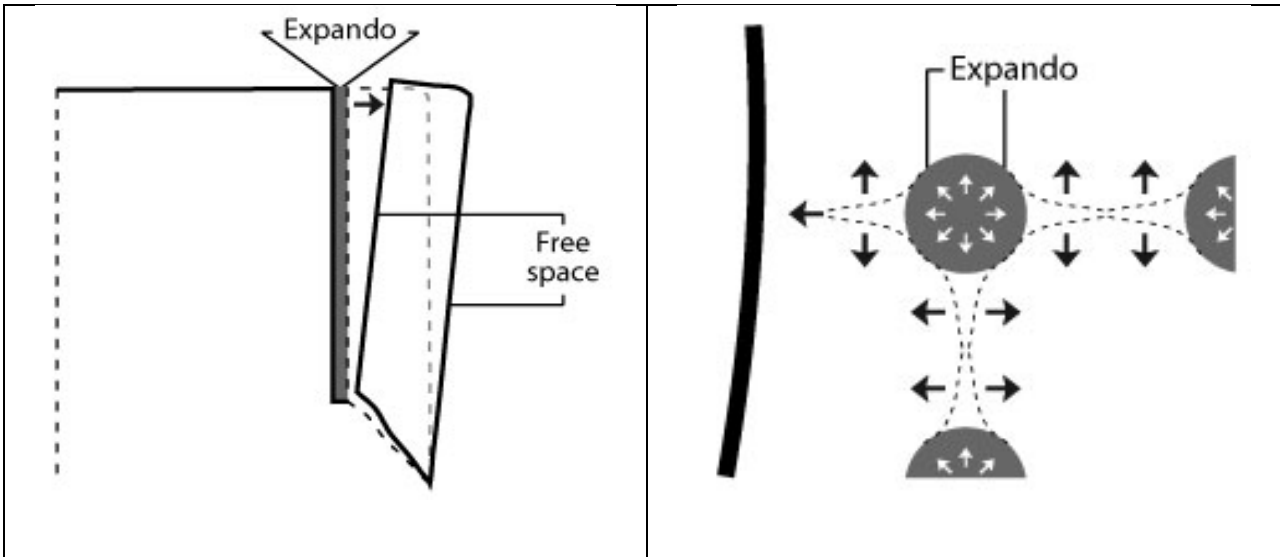
Figure 7: The influence of a free breaking face on the performance of expansive agents. Side view on left and plan view on right (Expando, 2017a)

In certain applications, a free breaking face is required to obtain optimum results for the method (see Figure 7). The pattern of holes drilled will have an effect on the duration of the fracture process. Quicker results can be obtained by placing the holes closer to one another, and the spacing of the holes can be extended when breaking speed is not essential. The spacing of the holes is directly proportional to their diameter. Smaller diameter holes result in sub-optimal performance, while larger holes result in blow-out shots during the expansion process (Expando, 2017b). The availability of a free breaking face has the potential to improve the performance of expansive agents. Chemical expansive agents have a strong adhesion and frictional resistance to the inner surface of the holes and do not require tamping.

Thermal rock-breaking methods

Rock is broken by weakening and shattering the rock mass by applying high temperatures (Res et al., 2003). Rocmec Mining Company of Canada has patented a thermal fragmentation method, which was initially used in open-pit operations for increasing blast hole diameters. The method has been used for over 30 years in Russia in open-pit mining, but the focus has shifted towards narrow-vein applications in recent years (Poirier et al., 2003). Thermal burners or kerfs are used to transfer heat and kinetic energy of temperatures up to 1 230 °C of thermal stream or flux (Res et al., 2003). The effectiveness of thermal rock

breaking is influenced by the type of rock, the efficiency of the thermal burners, the type of fuel used and the operating procedure implemented (Res et al., 2003). The method uses heat energy to erode the surface layer of the rock mass. The cohesive structure of the surface layer of the rock is weakened by the heat from the burners due to changes in internal stresses, rock melting and evaporation. The erosion of the rock occurs as a result of changes in the thermal and mechanical properties of the rock (Res et al., 2003). The method described produces rock fragments of up to 13 mm (Brisebois and Brisebois, 2010). This will, in turn, reduce the crushing and milling costs in the downstream process (Brisebois and Brisebois, 2010).

Thermal burners, lances or kerfs are used to heat the surface of drilled holes. Thermal burners are repeatedly used to enlarge the diameters of the holes by continuously eroding the surface of the rock. This method has various applications which include quarrying and crushing of oversized rocks (Res et al., 2003). Thermal reaction during the point of contact between the burner and the surface of the rock in the drilled holes spalls the rock surface, enlarging the hole diameter (Poirier et al., 2003; Brisebois and Brisebois, 2010). The surface of the holes is subjected to heat energy from the burners (Fecteau, 2010). The rock mass absorbs heat from the thermal burner, which induces the thermal fragmentation process (Fecteau, 2010).

Figure 8: The surface of a thermally fragmented and enlarged drilled hole (Brisebois and Brisebois, 2010)

Thermal rock breaking is accompanied by loud noises produced by a sudden and quick outflow of the thermal flux or stream during the process (Res et al., 2003). The thermal stream caused by heating the rock results in an endothermic reaction, which occurs as a result of an input of heat energy. The cohesiveness of the rock is weakened by the heat energy during the process, which results in rock fragmentation. Rocks such as dolomite, marble, limestone and shale, in which the endothermic reaction takes place during heating, cannot be broken using thermal methods (Res et al., 2003). Thermal rock breaking depends on the ability to erode the surface of the drilled holes, which is influenced by (Res et al., 2003):

- The thermal expansion of the rock mass
- Polymorphous phase transition
- Micro-explosions within the rock mass
- Dissociation.

Rocks are characterised by a relatively low linear and volumetric expansion, and low thermal conductivity efficiency (Res et al., 2003). The thermal rock-breaking method is also influenced by a thermal expansion coefficiency which varies with temperature as a factor of

the rock type (Res et al., 2003). The volumetric expansion coefficiency of thermal rock breaking is different for rocks that display isotropic properties. The applicability of the thermal rock-breaking methods depends greatly on the physical and mechanical properties of the rock mass, in particular the thermal and strength properties (Res et al., 2003). Heat is transferred from the thermal burners to the rock in the drilled holes. The heat transfer process occurs by either electron conduction or elastic vibrations (Res et al., 2003). Thermal conductivity is dependent on the composition of the rock, grain size, porosity, size and shape of pores, and the volumetric ratio of solid, liquid and gas (Res et al., 2003).

Thermal burners are fuelled with either diesel, oil or petrol, but mostly diesel fuel (Res et al., 2003; Poirier et al., 2003). The diesel-fuelled burners are also powered by compressed air (Poirier et al., 2003; Brisebois and Brisebois, 2010) and create a thermal cushion of hot air in the drilled hole (Brisebois and Brisebois, 2010). They vary in size and the temperature of the stream or flux also varies. The applicability of thermal fragmentation is also affected by the amount of silica contained in the rock (Res et al., 2003). The quick outflow of the thermal stream is between 1.4×10^5 to 1.5×10^5 cm per second, and this leads to the generation of high noise levels (Res et al., 2003). Thermal stress is created within the rock mass due to the heat from the burners. The temperature difference between the heat cushion and the rock mass causes the rock to shatter during contact (Brisebois and Brisebois, 2010). The rock is scaled down on the surface walls of the pilot hole and loosened by the compressed air (Brisebois and Brisebois, 2010). Figure 8 shows an example of a drilled hole which was enlarged by heating the perimeter wall. Brisebois (2007) explains how the method can be applied in narrow-vein deposits:

- Holes are drilled in the rock mass.
- The thermal burner is inserted in the holes.
- The burners is ignited to enlarge the diameter of the holes.
- Fragment sizes vary from grains to approximately 40 mm, and
    o Some form of decrepitating is caused by the unequal expansion of rock crystals which overcomes molecular cohesion.

Brisebois and Brisebois (2010) explain the thermal fragmentation rock-breaking method aimed at reducing mining width in narrow-reef mining. Waste rock can be avoided by targeting the reef horizon. This study was conducted by Rocmec Mining to explore the potential implementation of thermal fragmentation in narrow-vein deposits with the aim mainly of reducing dilution. The thermal fragmentation method makes use of a strong burner

which is inserted into a drilled hole. The drilled hole is similar to pilot holes, usually 152 mm in diameter. The burner inserted in the hole starts breaking the surrounding rock wall, thereby increasing the diameter of the hole. The diameter of the hole increases to between 30 cm and 80 cm (Brisebois and Brisebois, 2010). The thermal fragmentation mining method significantly reduces fragmentation by 400% to 500% (Brisebois and Brisebois, 2010) as shown in Figure 9. The burner is powered by diesel fuel and compressed air, which together produce a thermal cushion of hot air. The rock wall is broken by the temperature variance between the cushion and rock mass. This assists with the crushing of the ore before it enters the milling circuit.

Figure 9: Fragmentation results after spalling surface of the rock (Brisebois and Brisebois, 2010)
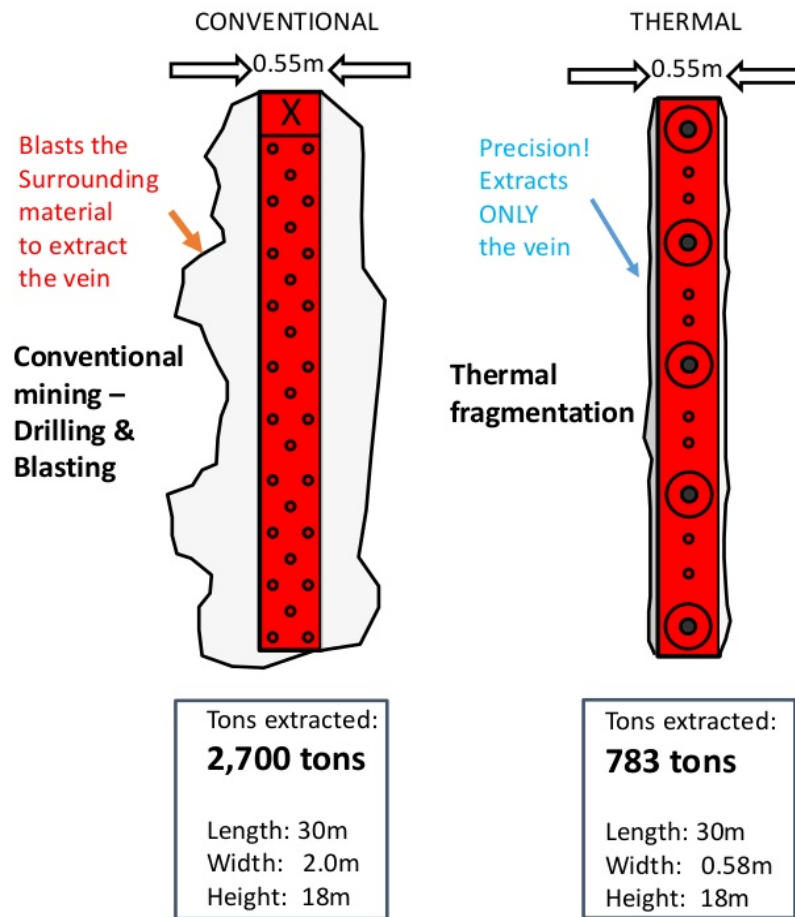
Figure 10: Reduced dilution through thermal fragmentation (Rocmec Mining – Nippon Dragon Resources)

The thermal fragmentation mining method is most suited for narrow-vein types of deposit. The method is similar to the sub-level drifting methods. Two sub-level tunnels are created, connected by a pilot hole. The pilot hole is then enlarged using the thermal fragmentation method as show in Figure 10 on the right. The distance between the two drifts is typically between 15 m and 20 m. The fragmented rock falls through the pilot hole into the bottom tunnel. The thermal fragmentation method is mainly used in reef. This reduces dilution by up to 500% compared to conventional drilling and blasting methods (Brisebois and Brisebois, 2010). The thermal fragmentation method can also be applied to create raise lines. A small pilot is pre-drilled to the required distance. A thermal fragmentation rod is then inserted to enlarge the pilot hole to the desired diameter. Two people are required working as a team. Selective mining is possible with the thermal fragmentation mining method, which will increase the overall grade in mines (see Figure 11).
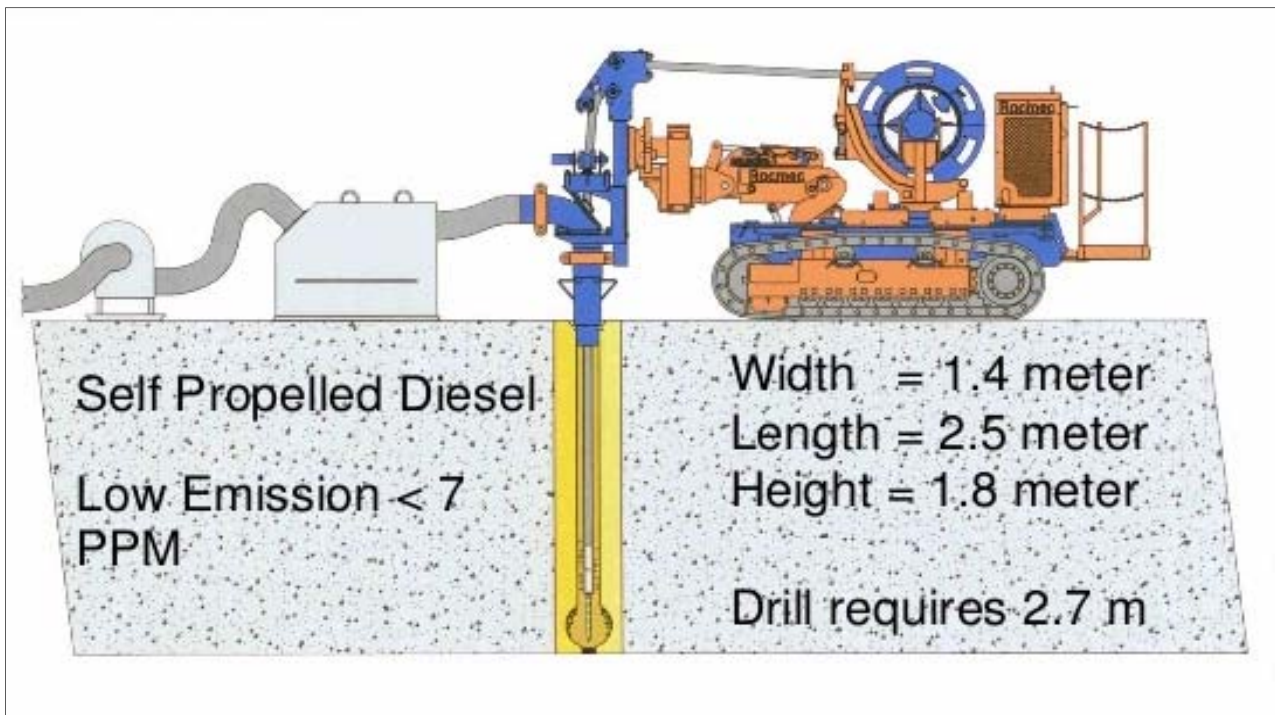
Figure 11: Large-scale thermal fragmentation method (Rocmec Mining – Nippon Dragon Resources)

The thermal fragmentation mining method replaces the bord and pillar, long-hole and shrinkage mining methods and is applicable to the following operations:

- Drop raise and/or blind raise
- Primary and secondary ventilation raises
- Small drainage holes
- Vibration blasting control
- Perimeter blasting and stress reduction
- Drift cuts
- Ore recovery in drifting
- Ore passes.

Hydraulic impact rock-breaking technology

Hydraulic fracturing refers to the injection of high-pressure steady or pulsejet water into drilled holes (Singh, 1998; Genet et al., 2009). Water at high pressure provides significant energy for shattering and cutting rock and concrete materials. The high pressure is generated by pumps. Fracture in rocks is caused by mechanical indentation or cyclic loading of impact forces (Hagan, 1992). Ultra-high-pressure water has previously been used for a wide variety of applications in cutting materials such as steel, concrete and rock (Gauert et

al., 2013). The method of hydraulic fracturing by impact technology makes use of an impact from a high-speed piston that pressurises liquid water. Various approaches have been attempted for determining the most effective rock-breaking method. Attempts have included blasting a charge while maintaining water pressure in the holes and driving a piston to impact the fluid, forcing a substance as a coherent mass column into a drilled hole (Singh, 1998). The hydraulic fracturing method is more effective in splitting unconfined boulders rather than breaking rock under confined conditions. The pressure from the high-speed water causes fractures parallel to the axis of the drilled holes, and this is not sufficient to excavate a face (Singh, 1998). The propagated cracks will intersect any free surface, in close proximity, to the drilled hole.

The high-pressure water in the hole reflects pressure waves which cause a build-up of high stresses in the holes. The huge stress in the rock is highest at the toe as shown in Figure 12 (Genet et al., 2009). Cracks are eventually initiated in the hole as a result of the pressure waves from the hydraulic pressure of the water. The cracks are propagated from the toe of the hole, rising all the way to the collar region. The pressurised water penetrates into the cracks, providing the driving force for crack propagation (Genet et al., 2009). The surrounding rock eventually fails, resulting in fracturing. The mechanism of hydraulic pressurisation is based on a pressure vessel. The pressure vessel is charged to 400 MPa and stores significant energy in the form of compressed water (Singh, 1998 from a study by Kolle, 1988). The study by Kolle (1988) investigated the potential of rock excavation by high-pressure water introduced into a pre-drilled hole. A hydraulic excavation tool was developed with a vessel charged at 400 MPa from a high-pressure pump to induce fracturing. Kolle (1988) explains that high-pressure water stores significant energy in its compressed volume. The total energy stored in a 2.2-litre vessel at a pressure of 380 MPa is equivalent to 49 KJ, with a discharge volume of 250 ml of water (Kolle, 1988; also noted by Singh, 1998). The findings of the study show that water at the correct pressure is capable of fracturing rock when injected into a pre-drilled hole at high velocity.
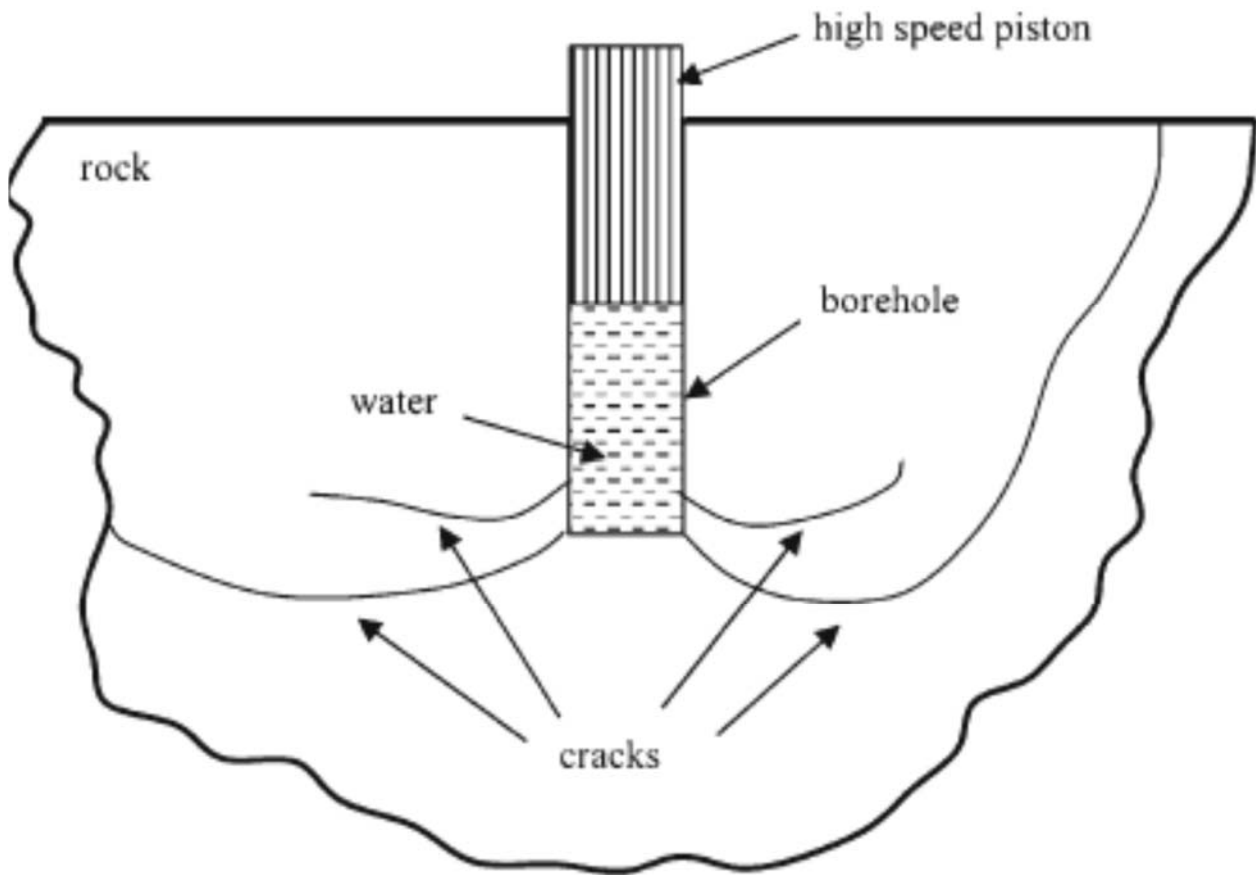
Figure 12: Illustration of the hydraulic impact rock-breaking method (Genet et al., 2009)

A hole is drilled into the rock and water is introduced under pressure into the hole either statically or dynamically (Lavon et al., 1978). Statically refers to when static water in a drilled hole is pressurised by a high-speed piston, and dynamically refers to when a high-velocity jet of water or other liquid is produced by a nozzle. Working pressure requirements in underground mining applications may restrict the use of the method (Lavon et al., 1978). Liquid is relatively incompressible and hence generates high pressures. The pressure creates stresses within the rock that result in rock fracturing. The velocity necessary to break rock by pressurising liquid in a hole is lower than the velocity in jet impact.

The internal stresses are caused by the high-pressure water, which result in fractures. The stresses within the rock mass weaken bonds between components in the rock, and free surfaces are created for the fracture process to occur (Hagan, 1992). Various factors are considered in the functionality of the hydraulic impact. These include the depth of the hole, the diameter of the hole and the properties of the surrounding rock. The efficiency of the hydraulic impact depends on the maximum principal stresses generated in the rock during

the impact process. Genet et al. (2009) investigated the parameters that affect the impact process during rock fracturing and came to the following conclusions:

- *Rock density:* This has a negligible effect on the stress created in the rock.
- *Depth of the hole:* This has no direct effect on the stress generated in the rock.
- *Diameter of the hole:* An increase in the diameter of the hole requires an increase in the diameter of the piston, which results in an increase in the initial kinetic energy of the piston with fixed initial velocity.
- *Arc radius of the bottom of the hole:* The arc radius at the toe of the hole has a substantial effect on the generation of stress in the rock mass.
- *Depth of water in the hole:* An increase in the depth of water in the drilled hole increases the stress generated in the hole. The deeper the water in the hole, the more energy is dispersed into the rock. The largest stress is therefore generated at the toe of the hole as defined by the distance travelled by the shock wave. The depth of the water increases the travel distance of the shock wave at the toe of the hole.
- *Length of the piston:* An increase in the length of the piston results in an increase in the stress generated. Longer pistons generate higher initial energy due to the fixed initial velocity.
- *Initial velocity of the piston and the initial kinetic energy of the piston:* The initial kinetic energy of the piston changes with changing piston length, piston diameter and initial piston velocity. An increase in the initial kinetic energy results in an increase in stress generated in the rock mass.

Rock breaking by mechanical splitters

There have always been mechanical rock-breaking methods, but they have not been commonly used in the mining industry until recently. The method of breaking rock mechanically is often compared to the traditional well-established drill-and-blast method (Rastomi, 2011). The mechanical rock breaking method has several advantages over drill-and-blast methods. These include an increased safety, higher production rates, reduction in labour, a higher degree of automation and minimal damages on the perimeter (Rastomi, 2011). Mechanical rock breaking methods are more common in underground mining than in surface mining.

As the method of rock breaking by splitting targets the tensile strength of the rock, the tensile stress in the rock is induced by a wedge set that is inserted in the rockmass to produce rock

fracturing. This requires an understanding of the stress distribution around the pre-drilled hole in which the splitter is exerting force. Hydraulic pressure is applied to the wedge, causing the feathers to push against the walls of the holes in the rock. Forces of up to 413 tons or 4 048 kN can be produced, for example by the DARDA® (2017) hydraulic splitters. The tensile strength of rock and concrete materials is a fundamental and important property due to the fact that rocks and concrete structures are brittle in nature, and are much weaker in tension as compared to compression (Luong, 1990; Sarfarazi et al., 2016). The strength of rock depends on its intact strength and the presence of micro-cracks, flaws and discontinuities. The strength properties of rock and concrete can be assessed from their deformation modes of fracture mechanics (Luong, 1990). A stand-alone rock mass sample, as compared to intact rock, has a reduced tensile and shear strength. Rock fails in different modes, such as tension, compression and shear, depending on the configurations of the load applied. Rock splitting is a function of the strength of the rock mass. The parameters that influence the efficiency and effectiveness of rock splitting include, but are also not limited to the following:

- The diameter of the drilled holes (mm)
- The spacing between the holes (m)
- The number of holes required

The hole must be drilled accurately so that it is straight and of a specific diameter for improved outcomes (ACI 555R-01, 2001). As indicated, a free face is needed on at least one side of the rock mass. This is to allow movement of the rock fragments. Having two free surfaces would be more effective and efficient in the splitting and fracture process (ACI 555R-01, 2001). A split between holes drilled along the same axis occurs when tensile loading is applied on the walls of the holes. Sarfarazi et al. (2016) conducted experiments that demonstrate how concrete material splits along the centre of a hole subjected to a tensile force. Splitting tests that were conducted show that a splitting failure occurs along the axis of the holes.

Rock breaking requires the breaking of the bonds between the constituent grains and minerals within the rock (Rastomi, 2011). The energy exerted by the rock-breaking method must be greater than the bond strength of the rock (Rastomi, 2011). The bond is broken by inducing stress between the particles making up the rock mass (Rastomi, 2011). Rocks and concrete structures are resistant to impact forces due to their higher compressive strength. Hydraulic splitters are designed to work through exploitation of the inside of the rock mass

where the tensile strength is lower. The tensile strength of rock is dependent on the direction and size of flaws in the rock. The stress required to break the rock in tension is 6% – 15% of the stress required to break the same rock in compression (Harper, 2008). Attempting to break the face of the rock in compression requires very high energy and bigger tools and machines. An inefficient transfer of energy from the rock-breaking machine to the rock results in energy being absorbed by the machine (Harper, 2008), causing excessive vibrations in and damage to the machine. Rastomi (2011) mentions that one of the most efficient strategies of breaking the bond is to apply a force using mechanical equipment that can produce concentrated forces. There is an inverse relationship between the specific energy of breaking rock and the sizes of the particles produced (Harper, 2008).

The failure process in brittle rocks under compression is characterised by micro-mechanical processes such as nucleation, growth and the coalescence of very small cracks within the rock (Lisjak and Grasselli, 2014). The micro-cracks lead to strain localisation in the form of microscopic fracturing (Benson et al., 2008; Lisjak and Grasselli, 2014). Rocks exhibit a strong pressure which is greatly dependent on mechanical behaviour. The failure behaviour of rocks is characterised by a non-linear envelope and a transition from a brittle state to ductile state (Kaiser and Kim, 2008; Peterson and Wong, 2004; Lisjak and Grasselli, 2014). The failure process in rock mass is further characterised by the presence of geological discontinuities and weak zones (Lisjak and Grasselli, 2014). Geological discontinuities and weak zones reduce the strength of rock as they affect the response of intact rock. This induces a non-linear relationship in the stress-strain mode failure (Lisjak and Grasselli, 2014). The fracturing process of rocks is greatly affected by the properties of the rock, including the bond strength and pre-existing discontinuities.

Variations of failure modes, from axial splitting to shear band formation, increase the confining pressure in the pre-existing micro-cracks. The method of breaking rock by axial splitting was initially developed for secondary breaking because boulders are easier to break in tension. The mechanical or hydraulic splitters are placed or inserted in drilled holes along a breakage plane (American Concrete Institute 555R-01, 2001). A split-fracture in the rock occurs as a result of a splitting action developed by a wedge between the two side-by-side feathers. Hydraulic pressure is applied to the piston of the wedge and the feathers exert force on the walls of the holes. The piston pushes the wedge in a downward direction while causing the counter-wedges to push sideways. Forces of up to 3 100 kN can be produced by splitters depending on the specifications of the unit (American Concrete Institute 555R-

01, 2001). The rock splitting operation does not generate dust, noise, shocks or vibrations. The operators are clear of the operation, and there are no safety hazards associated with the splitting operation. The rock splitting operation is envisaged as a continuous excavation method with no interference in nearby activities (Singh, 2008). The potential of splitters as a primary rock-breaking technique has been studied (Hadjigeorgion, 1994).

Several researchers have shown that various materials have either a ductile or a brittle failure mode. Brittle material such as rock and concrete fails by fracture rather than yielding, as opposed to ductile materials. The brittle fracturing process occurs as a result of tensional force loading. Brittle material, such as rock and concrete, contains flaws, cavities, inclusions and other inhomogeneities (Horii and Nemat-Nasser, 1985; Li et al., 2017). In brittle materials any discontinuities may produce tensions cracks under compression (Horii and Nemat-Nasser, 1985). Li et al. (2017) also note that when the confining pressure is very close to zero, an axial failure mode is experienced. The failure mode changes into a shear failure mode when the confining pressure is increased. The nucleation, growth and interactions of micro-cracks within the rock or concrete is the dominant and controlling factor of the fracture mechanism (Horii and Nemat-Nasser, 1985).

There are two distinct mechanical splitting methods (Murray et al., 1994):

1. Radial-axial splitters
2. Plug and feather (radial) splitters

The splitting actions of these splitters are described in more detail in the following section.

Radial-axial splitters

The radial-axial splitter is a hydraulic tool which is inserted into the pre-drilled hole. Paraszczak et al. (1994) came to the conclusion during a literature study that the tool is of a relatively simple design and does not require any complex supporting infrastructure (see Figure 13). Radial-axial splitters incorporate a special hydraulic cylinder employing two independently actuated hydraulic pistons. The pistons "A" and "B" operate the in-hole components, namely the thrust rod and the wedge and feathers respectively. The thrust rod (rigidly anchored to piston "A") is used to exert an axial load against the toe of the drilled hole. The wedge is attached to piston "B", and the feathers are attached to the splitter casing. The wedge and feathers transmit a radial force to the drilled hole walls. After the

tool has been inserted into the hole, port III is opened for the hydraulic fluid to enter, forcing piston "B" laterally outwards and wedging the feathers against the drill hole walls (F_r). The hydraulic fluid then enters through port I, pushing piston "A" forwards and exerting a force (F_a) against the toe of the hole through the thrust rod. The combined radial (F_r) and axial forces (F_a) will initiate a crack starting at the contact point between the surrounding rock and the feathers, propagating towards the nearest free face. A cone-shaped crater is formed, as seen in Figure 13. After the fracture has occurred, both pistons are returned to the initial positions by allowing hydraulic fluid to enter through port II.

According to Paraszczak et al. (1995), two problems were encountered with the thrust rod. Due to the high pressures exerted on the rod, buckling and plastic deformation occurred on the end of the rod. Both these issues were resolved by an improved design which makes buckling not possible. The rod is guided along most of its length. It is manufactured using a high-strength material of fixed diameter.
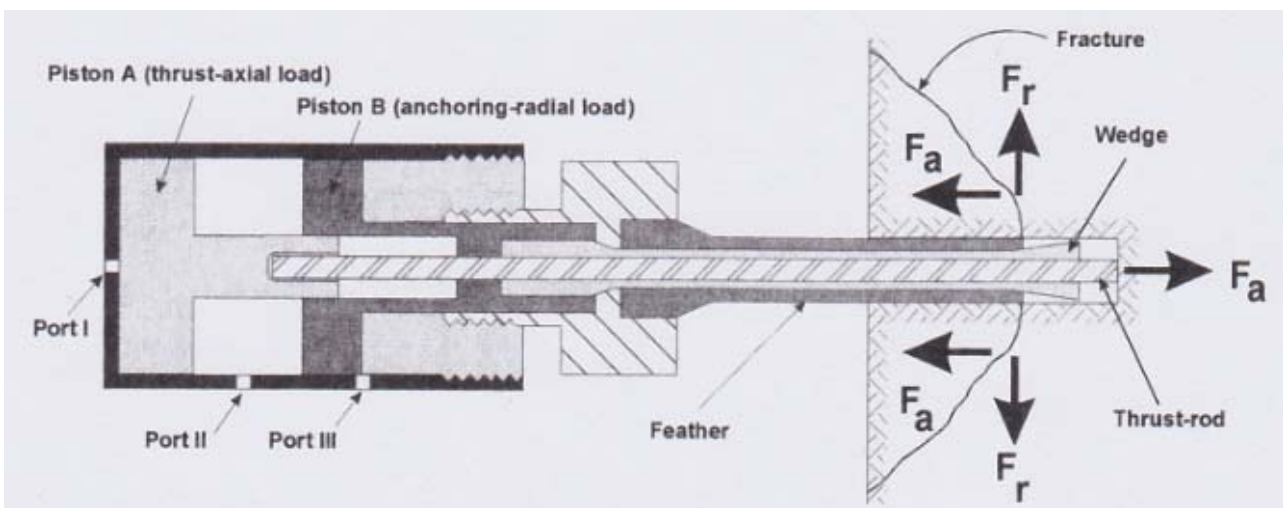
Figure 13: Working principle of a radial-axial splitter (after Anderson, 1982)

Paraszczak et al. (2003) concluded the following after experimentation in an underground environment:

1. Radial-axial splitters are capable of breaking highly resistant rock within various mining scenarios and confined spaces without imposing changes to the mining infrastructure.
2. Breakage induced by the splitter is predictable and can be controlled, enabling mining-specific areas with limited dilution.

3. The splitter action causes much less disturbance to the environment with little undesirable fracturing and fewer disruptions to the environment, making the area safer for the miners.

4. The handheld prototype splitter used in the underground environment produces limited broken rock fragments with an average of 30 – 40 kg rock from one drilled hole.

5. The splitter's availability and reliability have been found to be satisfactory with no major problems encountered.

Radial splitting

Radial splitters consist of a double-acting hydraulic cylinder, a wedge and twin feathers as shown in Figure 14. The wedge and feathers assembly is inserted into a drilled hole. The wedge is driven by a piston between the feathers, and this action exerts an outward force against the walls of the hole. As a result, a radially loaded stress starts building up and fractures the rock mass along its axis as shown in Figure 15. Previous research has focused on using radial splitters to break rock to free surfaces that are perpendicular to the face and parallel to the axis of the holes drilled.

Figure 14: Hydraulic rock splitter (DARDA, 2017)

One of the major advantages of the technique is the fact that the direction of breakage along a split line is predetermined (DARDA® Rock Splitters, 2016). The method does not produce any shockwaves, dust, noise or vibrations during the fracturing process. Splitters are well

known as feathers and wedges because they consist of one wedge and two semi-circular feathers on either side.
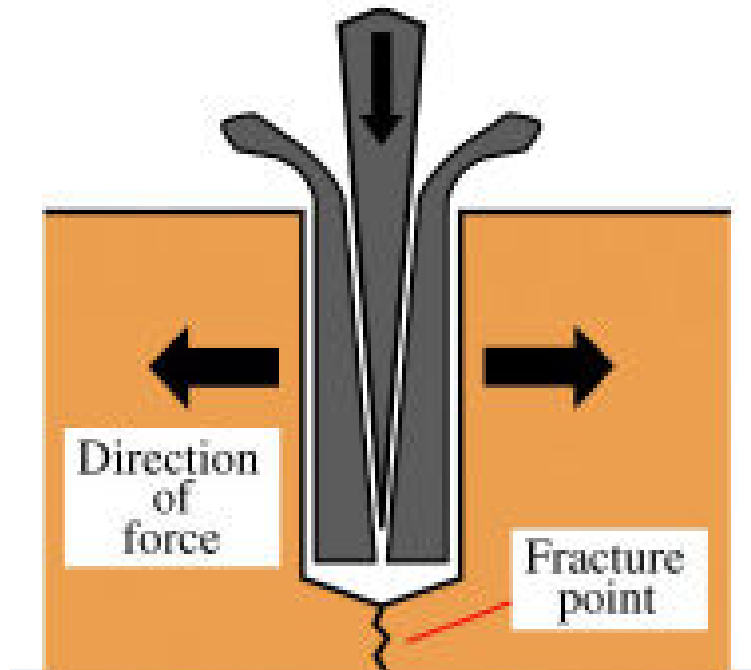
Figure 15: Wedge and feathers of a rock splitter (http://www.jpaleontologicaltechniques.org)

The radial splitter application and lessons learned in the narrow reef hard rock mining environment will be discussed in more detail in the next chapter.

2.2 Summary of non-explosive rock-breaking systems without the need for drilling holes.

This section summarises non-explosives rock-breaking systems and techniques without the need to drill holes into the rock mass. The purpose of this section is not to capture every detail of the system, but to provide an overview of alternative systems not discussed in the previous section.

The overview of the technologies is organised according to the mechanism by which the rock is broken, weakened or cut:

- Abrasion cutting – examples include diamond saw cutting, diamond wire cutting and water cutting.
- Mechanical cutting – examples include impacting method, rolling and dragging.

The above-mentioned methods are described in more detail in the following section.

2.2.1 Abrasion cutting

Diamond saw cutting

Diamond saw cutting (Magri and McKenna, 1986) was introduced into the gold mining industry to obtain in situ samples from underground stopes and development ends. As the name implies, a machine is used with a single-bladed diamond circular saw (diameter of up to 800 mm) to cut into the rock mass. Tests at Randfontein Estates Gold Mine (REGM) produced results during cutting of 3 m^2/h using a cutting power of 24 kW and water flows of 10 l/min. The method demonstrated that it had the potential for viable rock cutting.

The method is not currently used as a number of challenges were experienced regarding blade wear, jamming of the blade and rig design.

Diamond wire cutting

Diamond wire cutting has been successfully introduced into the dimension stone industry to obtain rectangular blocks with high recovery rates (Bortolussi et al., 1994). The initial diamond wire machines consisted of single-strand cutting wire and currently consists of a multiple diamond wire. Initial trials were conducted with a prototype multiple diamond wire machine which was developed by Yamana Company, Japan, in 1994. Smith (1992) concluded that diamond wire rope cutting was viable but costly, and a number of challenges were encountered, the main one being that the diamond wire became trapped as the stope was closing due to the weight above.

Trials conducted at Harmony's Saaiplaas No. 5 shaft showed that the waste produced from narrow reef cutting was less compared to that produced by the conventional drill-and-blast stoping (Smith, 1992). The trials were discontinued as the method of cutting could not produce 400 m^2 per month due to the difficulty of the removal of the loosened rock. The wire constantly jammed and breakage occurred due to localised seismicity.

More recent trials were carried out at a platinum mine (SAMERDI implementer reports, 2018) with an 11.5 mm diameter diamond wire used to cut the rock. The system achieved, in a rock mass with a uniaxial compressive strength of 188 MPa, a penetration of 1 m per minute over a 300 mm advance. It took 1 hour 30 minutes to cut 36 m^2 of rock. The life of the wire was around 45 m^2 per metre of wire. The inefficiency of the cleaning of the rock and the supporting of the area prevented the maximum cutting and therefore the successful implementation of the method.

Waterjet cutting

Waterjet cutting (Gauert et al., 2013) is a water jet with an added abrasive agent sprayed at extremely high pressures onto the material to be cut. Both shattering and cutting of mineral grains and cements occurs (Momberg and Kovacevic, 1998). The system comprises a high-pressure pump and a diamond-plated orifice (See Figure 16).

Figure 16: Streamline TM PRO-I 60/125HP Pro Intensifier, ultra-high-pressure pump and orifice size of 0.35 mm (Gauert, et al.)
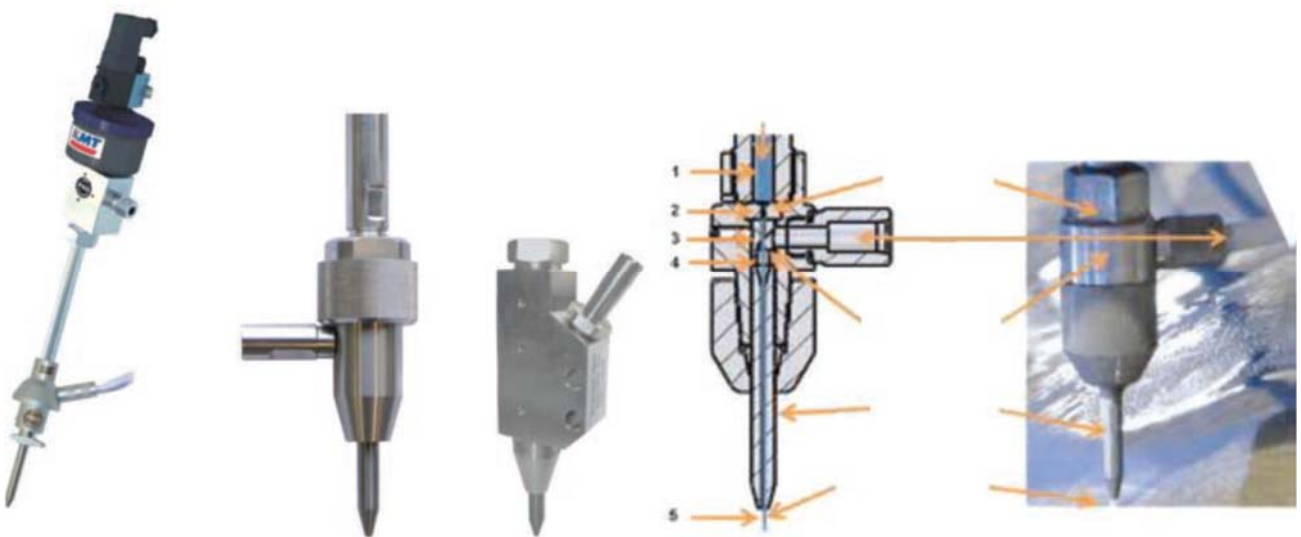

Figure 17: Pro-cutting head by KMT (WARDjet)

The pressures for waterjet cutting during trials varied between 3.5 kbar and 6.2 kbar. Limitations to the system include the high pressures (~6 kbar) required to cut the rock, the slow advance rate and the high water consumption.

2.2.2 Mechanical cutting

Mechanical cutting of rock is currently the most popular and common non-explosives rock breaking according to Vogt (2016). Shao (2016) states that all rock-excavating machines operate by forcing the cutting tools into the rock and causing it to break. The cutting tools of the various machines vary depending on the rock type. Examples include drag bits, discs, and roller and button cutters. Huang et al. (2013) emphasised that the direction of the tool motion with respect to the rock surface is an indication of the tool and rock interaction, described as indentation or cutting.

The mechanism of mechanical cutting can be divided into three main actions to break or weaken the rock mass:

- Dragging – High forces are needed to break the rock mass, which require larger machines and cause increased wear on the tools. Drag bits are more efficient at cutting rocks compared to indenters (Vogt, 2016). The picks are made of extremely strong, expensive brittle material prone to early failure. An example of dragging is a continuous miner (CM) used to cut coal. The CM has a rotating drum with a number of strategically placed drag bits and rotates while being pushed against the coal face (see Figure 18).

- Rolling indenters – The indentation of the tool onto the rock mass is the method used with the majority of rock cutting tools, included all cutters which roll, for example disc cutters, rolling cones and button bits as seen in Figure 18. The tools are forced against the rock face. Rolling discs are used mainly in tunnel boring machines, shaft borers and raise borers and in blind hole drilling – they are suitable for breaking very strong rocks.

- Impacting tools – The tool is compressed against the rock mass and subjected to a hammer action, i.e. pulses rather than continuous pressure. Examples include the impact ripper, the hammer & chisel method and demolition machines.

Drag bits and disc cutters are the most common cutting tools. Multiple tools are mounted in a pre-determined pattern on the drum or cutter head of machines (Roxborough, 1986). Two

main classes of rock-cutting tools, i.e. drag bits/picks or cutters and indenters are shown in Figure 18.
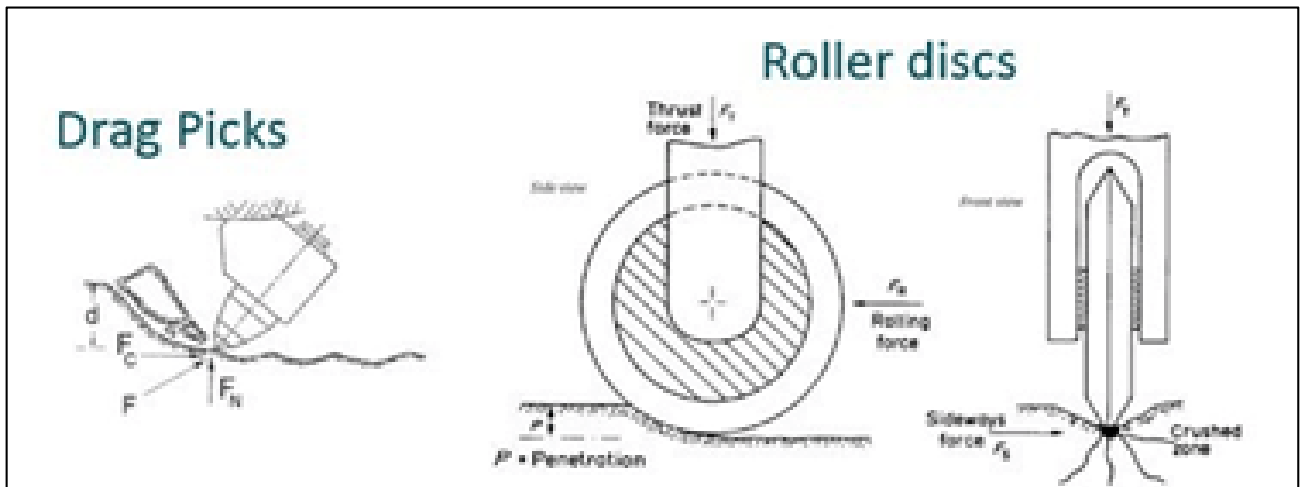
Figure 18: Main classes of cutting tools (Source: CSIRO Hard Rock Cutting Research Group)

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3   FIELD TRIALS

3.1   Introduction

An extensive literature research was conducted by the Gold Fields Mining Innovation team to investigate the possibilities of practical autonomous non-explosives rock breaking methods in the deep-level hard rock environment. The decision to explore the radial splitter was based on the simplicity in design, robustness of the splitting unit and the power supply. Furthermore the ease of deployment and handling made the decision easier. The potential to mount the unit onto an autonomous vehicle was a huge advantage. The project was initiated under strict confidentiality conditions and the author was brought on board after the decision was made to conduct trials with the DARDA® hydraulic splitter.

An extensive internet literature research did not find any experimental work or trial results using a hydraulic splitter in a deep-level mine environment. Therefore the trials performed in the deep-level hard rock mines were purely initiated to evaluate the DARDA® hydraulic splitter's functionality and applicability. The trials were conducted on a clean slate, with no benchmarks set to measure the results achieved.

3.2   DARDA® hydraulic splitter

Detailed descriptions and specifications of the various DARDA® splitting cylinder devices are given on the company's website (www.darda.de). Figure 19 shows the DARDA® splitting cylinder and power supply.

Essentially the concept makes use of the wedge principle, whereby a borehole is initially drilled into the material (concrete, rock), and then the DARDA® hydraulic splitter wedge set (two counter-wedges or feathers and a central wedge) is inserted into the pre-drilled hole. The central wedge is then driven forward between the two feathers under hydraulic power, forcing the feathers outward against the hole wall. Radially loaded stress build-up is created that fractures the rock. This DARDA® hydraulic splitter unit will apply a pressure of up to 400 tons to force the feathers apart (see Figure 20) (Anderson et al., 1982, Duncan et al., 1972 and Paraszczak, et al., 1994). A detailed mathematical analysis of the stresses induced in the rock by the splitter was done by Chollette et al. (1976).

Figure 19: Handheld DARDA® Radial Splitting Cylinder and power supply (DARDA® website)

According to the DARDA® website, the benefits of hydraulic splitting include controlled breaking without the negative side effects seen with impact breakers or conventional drilling and blasting practises.
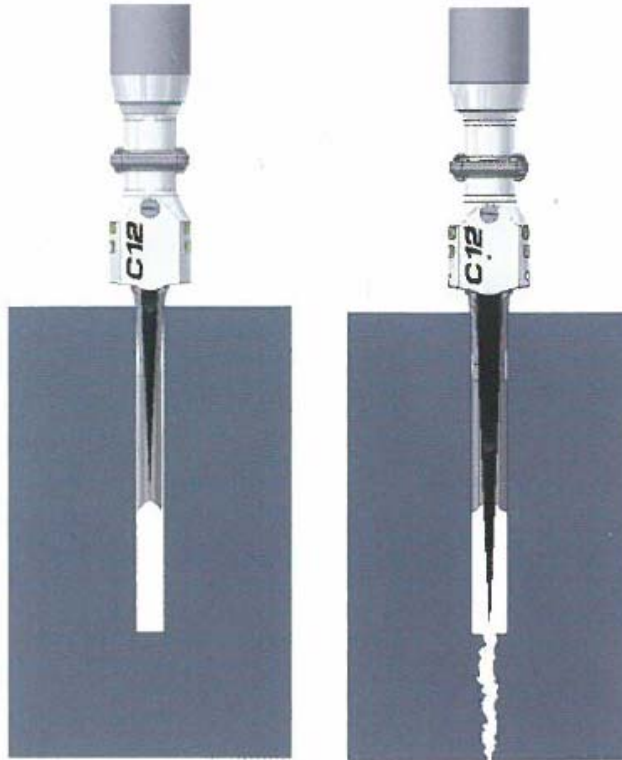
Figure 20: The rock splitting concept (DARDA® website)

The DARDA® hydraulic splitter has been successfully applied in the general civil demolition field for breaking and splitting concrete and rock. A typical application of splitting on surface is shown in Figure21.

Figure 21: Typical application of the DARDA® hydraulic splitter (DARDA® website)

The splitting unit is manufactured from steel and aluminium to reduce the weight. The control valve will either extend or retract the centre wedge. The wedges are manufactured from steel coated with a carbide layer for increased durability.

The hydraulic pump delivers a maximum of 50 MPa to the splitter. The pressure is controlled through a pressure limiting valve. Different drive systems are available for various applications and can be either electric, air, diesel or fuel motors.

During these trials, a 220 V electric motor was used to drive the hydraulic power pack.

The DARDA® hydraulic splitter is constructed for robust environments. The material properties of the components, and the extremely high forces exerted to fracture the rock mass, impose certain restrictions on the equipment size and weight, which in turn impose limits on the operability, such as the minimum hole diameter. The C12L model splitting cylinder was used in these underground trials. This splitter unit weighs 32 kg and is approximately 1.3 m in length, which is not very different from the weight and proportions of a conventional handheld pneumatic rock drill. The applicable hole diameter ranges between 45 mm and 48 mm. The two feathers, or counter-wedges, are 450 mm in length. However, the minimum hole length required is 680 mm to accommodate the extending centre wedge.

The complete DARDA® hydraulic splitter consists of the following:

- Hydraulic splitting cylinder

- Counter-wedges and centre wedge

- Drive system, hydraulic pump unit and hydraulic hoses

3.3   Field work on surface

Background: The DARDA® hydraulic splitter was tested on surface before taking it underground. A suitable test site was identified in the Alberton area, adjacent to a motorway. The site was easily accessible and no permits or permission were required to enter the site. The main purpose of the particular site was to make use of experienced labour in DARDA® hydraulic splitting and to get a basic understanding and feel for the capabilities of the DARDA® hydraulic splitter. A DARDA® hydraulic splitter with a diesel-driven power pack was used on site as shown in Figure 22.

Figure 22: Diesel-driven power pack for the DARDA® hydraulic splitter (Author, 2013).

Holes were marked in a pre-determined pattern and drilled to a depth of 1.2 m using a pneumatic percussion drill. The hole diameter was 46 mm. The hole depths were measured, and once the quality of the holes was found to be satisfactory, the DARDA® rock splitter was inserted into the first hole. Figure 23 shows the DARDA® rock splitter breaking rock towards the centre hole and to the left upper hole.

Figure 23: Application of the DARDA® hydraulic splitter (Author, 2013)

Figure 24: Progress made with the DARDA® hydraulic splitter (Author, 2013)

Figure 24 shows the progress made during the breaking of the 5-hole drilled pattern. The final excavation depth was approximately 140 mm created by the DARDA® hydraulic splitter. Several other hole patterns were drilled and mechanically the DARDA® hydraulic splitter performed well in fracturing and breaking the rock mass (see Figures 25 and 26). The functionality of the DARDA® hydraulic splitter and the ease of operation was demonstrated and proven on surface.

Figure 25: Fracturing during 5-hole trial (Author, 2013)

Figure 26:. Opening created with debris manually cleared (Author, 2013)

3.4   Underground test work

3.4.1   South Deep Gold Mine tunnel excavation

The DARDA® hydraulic splitter and power unit were taken to 95 level at Gold Fields South Deep Gold Mine, situated in the far West Rand near Westonaria. A request from the "new technology" team at South Deep Gold Mine for a demonstration / trial was conducted at the mine on 95 level. The excavation height was in excess of 3.5 m, making it relatively easy for all personnel involved to move around. A random area was selected in the tunnel with electricity (220 V), water and compressed air readily available. The holes were drilled using a pneumatic percussion drill with a drill bit diameter of 46 mm. The DARDA® hydraulic splitter used during the test was equipped with an electrically driven hydraulic power pack, as opposed to the diesel-driven unit used in the surface trials.

The tunnel was lined with a shotcrete layer. A test pattern was marked on the sidewalls of the tunnel excavation. An initial 5-hole cut was drilled, as shown in Figure 26. The distance between holes in the vertical and horizontal direction was 150 cm.

Figure 26: Marked 5-hole pattern on the side wall (Author, 2013)

The 5-hole pattern and the resultant excavation are shown in Figure 27. The depth achieved was 35 cm.

Figure 27: Five-hole pattern and excavation (Author, 2013)

Some additional holes were drilled on the perimeter of the 5-hole cut pattern. The existing cut holes were deepened by inserting the drill steel and extending the holes by 50 cm. An additional advance of 15 cm was achieved after the DARDA® hydraulic splitter was inserted into all the holes. An average depth of 50 cm into the rock mass was achieved during the trial. The tests proved the functionality of the DARDA® hydraulic splitter and the goals set were achieved.

3.4.2 Pitseng trial stope

Trials were conducted over a period of five months at the then Gold Fields' KDC West Gold Mine Pitseng shaft. At the Pitseng shaft a non-production stope was established on level 24 where several new mining initiatives could be trialled and tested. The stope width was on average 1.5 m of unfractured footwall quartzite with a uniaxial compressive strength (UCS) of 160 MPa (personal communication – Prof. F. Malan, University of Pretoria). The stope was equipped with the necessary services, including water, compressed air, electricity (220 V) and wireless networking (WiFi) connections for communication.

Figure 28 shows where the tests were carried out on the rock face: position A in the absence of a second free face, and position B adjacent to the gully, where a second free face was present. During the trials drilled holes were grouped into one of two categories, either "cut" for the creation of a second free face and "slicing" for breaking the rock along the second free face.

Figure 28: Layout of the trial stope on 24 level, where the trials were conducted (Ras Botma, 2013)

All the proposed cut designs were carefully marked on the face and the drilling accuracy was measured and noted. The drilling accuracy was determined using a clinorule to measure the angles, hole depth and horizontal and vertical distances between the holes (see Figure 29).

Figure 29: Measuring the hole depth and inclination of drilled holes (Author, 2013)

A simple handheld spring scale was used to measure the amount of rock broken from the cut designs. Still photographs and video footage of all the splitting operations were taken.

The "cut" holes

The design of the hole layouts for splitting was based on conventional drilling and blasting knowledge and engineering judgement, as well as established techniques in the use of the splitter in surface mining and demolition applications. In mining terms the "cut" is a pattern of closely spaced drilled holes in the rock face, used to create a second breaking face. In conventional drill-and-blast operations, the "cut" drill pattern differs from the production drill pattern in hole layout, drilling angle and initiation timing of the blast holes. The sole purpose of the "cut" is to create a second breaking face into which successive production holes will break outwards to the perimeter of the excavation. A large variation of "cut" designs have been proposed for hydraulic splitting, for example the United States Bureau of Mines (Anderson et al. 1982) suggested a spiral-shaped round drill pattern, whereas Clark et al. (1978) recommended creating a second free face by using a method similar to the V-cut used in an explosive round.

Breaking rock using the DARDA® hydraulic splitter is a much slower process compared to conventional blasting. The rock around each hole has to be individually fractured until the rock breaks into an adjacent drilled hole or a void. Holes must be concentrated in the "cut" area to facilitate the creation of the second free face. In this investigation, the void area was drilled according to several proposed hole layouts. These layouts included a few designs with holes drilled at an angle, a few combinations of straight and angled holes and designs where larger holes were included in the cut pattern. The intention was that the larger holes would act as a "mini cut" into which the splitter holes would break.

A number of drill patterns to create the cut were carefully considered. As the holes have to be drilled to a minimum depth to accommodate the fully extended central wedge, holes drilled perpendicular to the advancing face had the advantage that they could be deepened and used again for splitting.

In conventional drill-and-blast tunnel development, creating the cut requires a high explosive energy concentration (Explosives and Rock Breaking, 1987), hence the high concentration of blast-holes. Once the "cut" has been created, hole burdens can be increased because a second free face is now present. To reduce drilling time during these trials, an attempt was made to create the most efficient hole layouts for the "cut" area in terms of the least number of holes for effective breaking using the DARDA® hydraulic splitter. As mentioned, the holes were drilled to a length that would accommodate the protruding centre wedge. After the first round of breaking the remains of each hole would have to be deepened for the next round. Holes drilled at an angle could not be deepened, as they would extend beyond, and break outside the perimeter (see Figure 30).

The one area where angled holes would be necessary would be in the case of perimeter holes, i.e. holes on the edge of the stope. These holes would have to be drilled at a slight angle, referred to as the "look out" angle (Cooper, G.A. et al.). The holes would be collared inside the planned perimeter and angled outwards with the toe of the hole situated on the outside of the perimeter (see Figure 30) to accommodate the centre wedge of the splitter but to retain the contours of the stope. In these trials perimeter holes were not drilled or investigated.
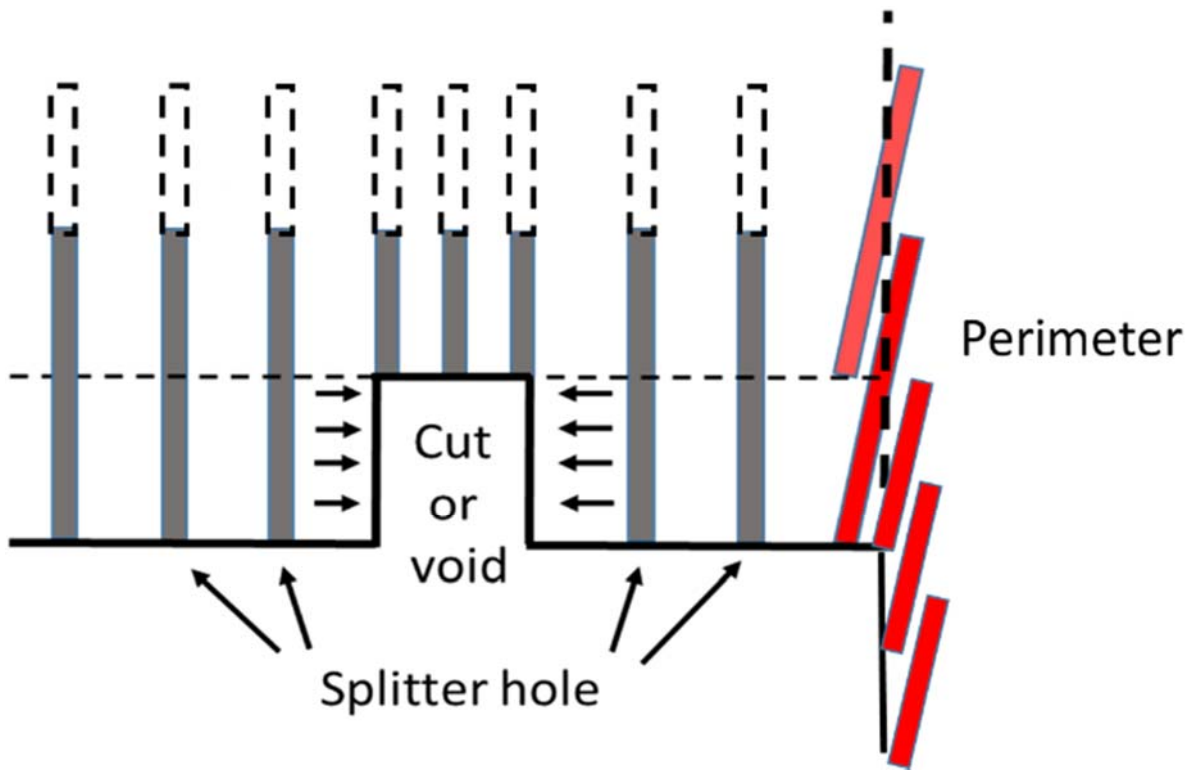
Figure 30: Plan view of splitting rock into newly created void and perimeter holes drilled at an angle

A number of "cut" patterns were proposed, which are described in the appendix. During this investigation four of the cut designs were considered and tested underground. The choice of these four trial patterns was based on simplicity of the drilling pattern and the minimum holes drilled per cut. The four designs are described below.

Cut - Hole layout 1

The hole layout 1 drilling pattern consisted of five parallel holes to create the "cut". The four outer holes were drilled in a 300 mm diamond pattern (see Figure 31). The fifth hole was drilled in the centre of the diamond. The holes were drilled to a minimum depth of 700 mm.

Hole diameter: 46mm
Hole length: 700mm

'A' dimension: 300mm

Plan View

Front View
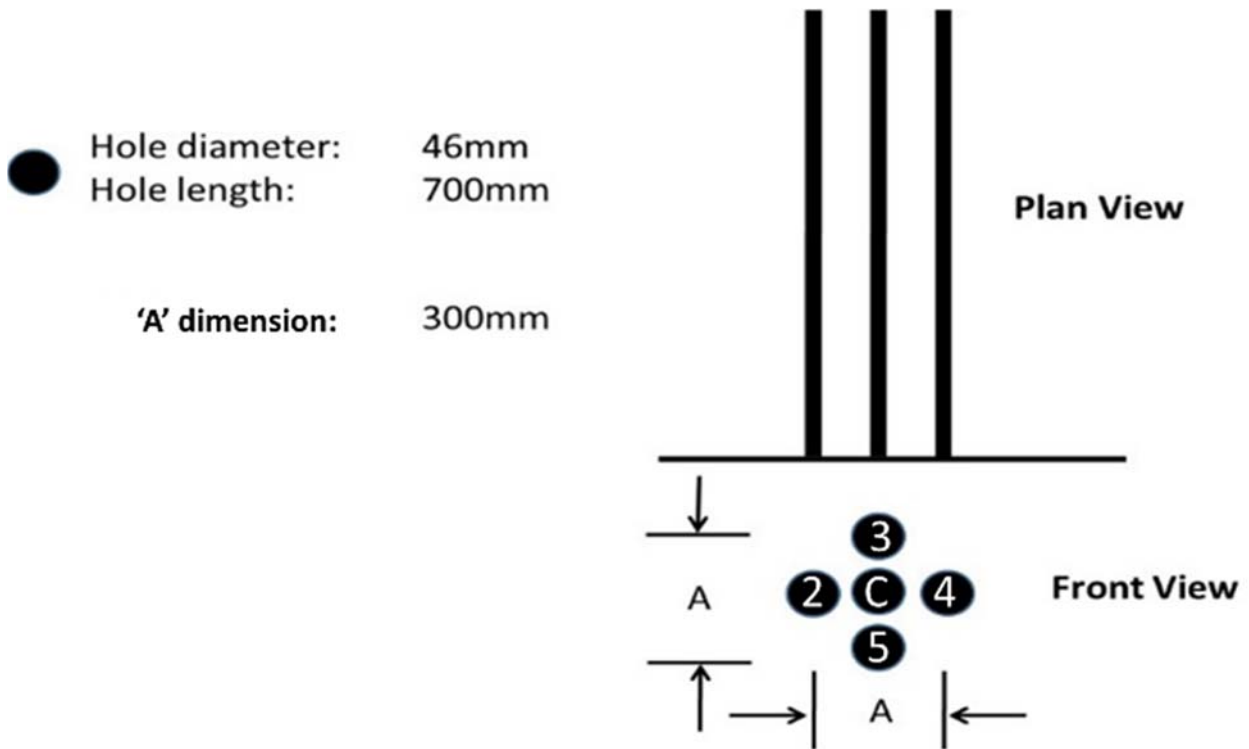
Figure 31: The five-hole diamond pattern

Cut - Hole layout 2

The second pattern trialled consisted of seven holes. All the holes were drilled to a depth of 700 mm parallel to one another and perpendicular to the stope face. The outer holes were drilled in a hexagon pattern with dimensions of 450 mm (Figure 32). One hole was drilled in the centre. Breaking was initiated by inserting the splitter in the centre hole, hole "C".
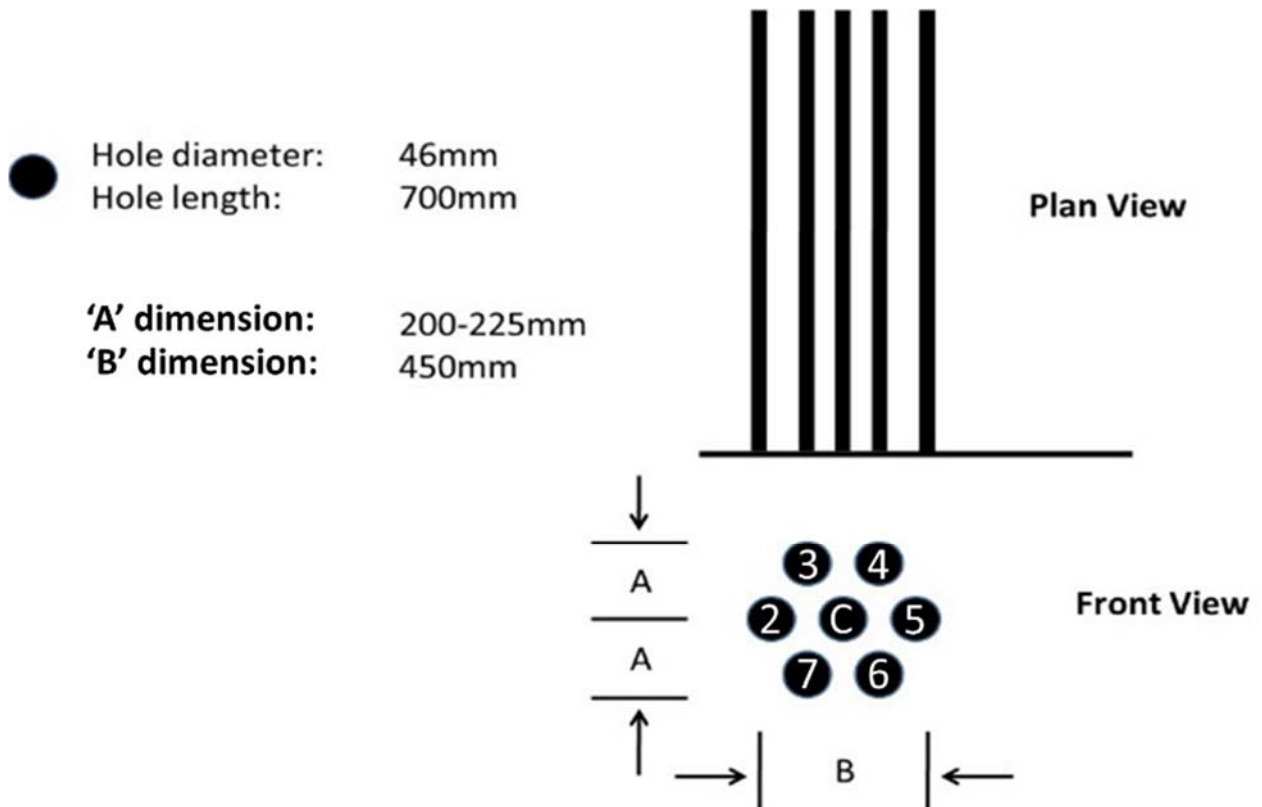
Figure 32: The seven-hole hexagon pattern

Cut - Hole layout 3

Figure 33 shows a drilling pattern of 11 holes. The two outer rows of four holes each were drilled in a rectangular box pattern to a minimum length of 700 mm. The outer holes converged to the centre row of holes at an angle of between 3º and 6º to the perpendicular. Three holes were drilled to a depth of 350 mm, parallel to one another, between the two outer rows of holes. The splitter was inserted in the outer holes and the inner holes acted as break or relief holes.
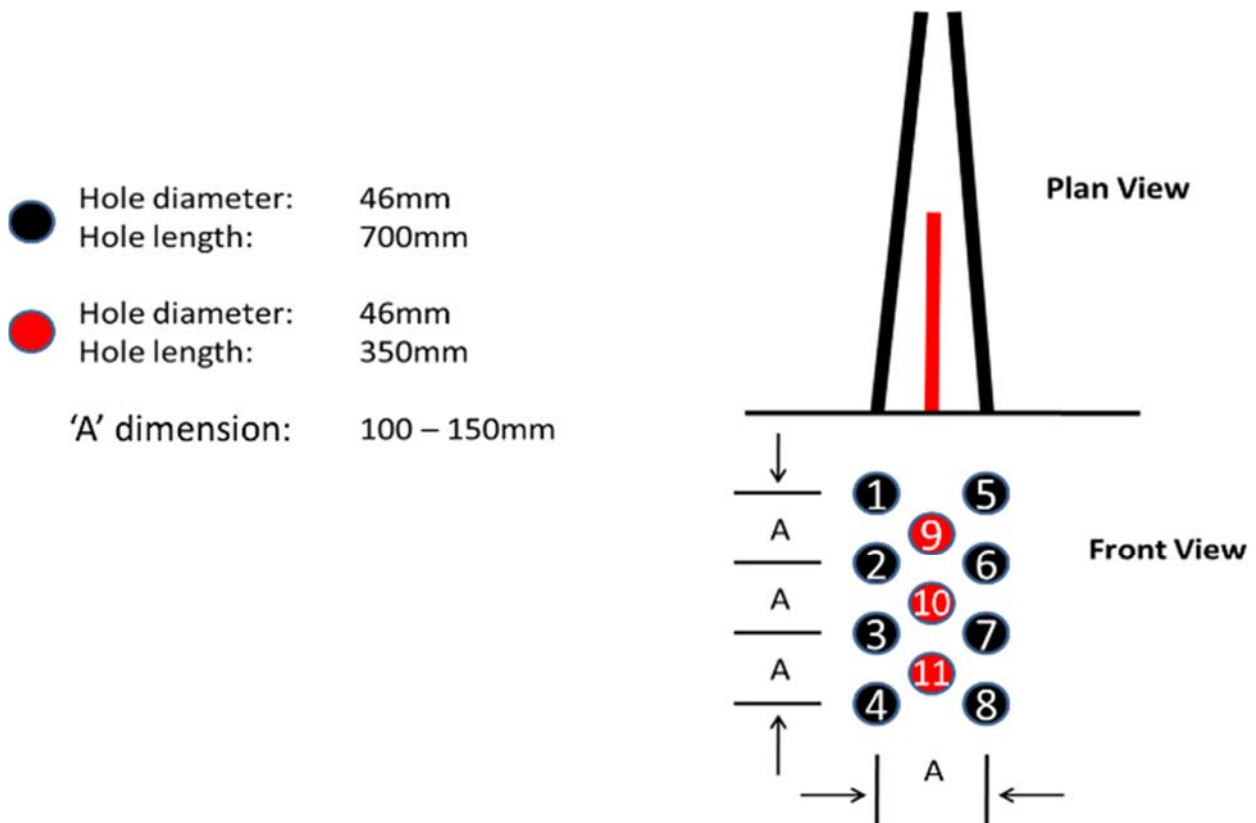
Figure 33: Rectangular box pattern with staggered (non-splitter) centre holes

Cut - Hole layout 4

Figure 34 shows another combination of perpendicular holes and angled holes. The perpendicular "relief" holes were drilled parallel to one another, and perpendicular to the rock face to a short depth of 350 mm. The angled holes (700 mm depth) into which the splitter was inserted, were drilled as a rectangular nine-hole pattern converging towards the perpendicular short holes at an angle of between 19º and 23º to the perpendicular. The angles were dictated by the ease of collaring and to minimise the burden between the first row of angled holes and the adjacent perpendicular holes (dimension C in Figure 34). The splitter was initially inserted into hole number 2.
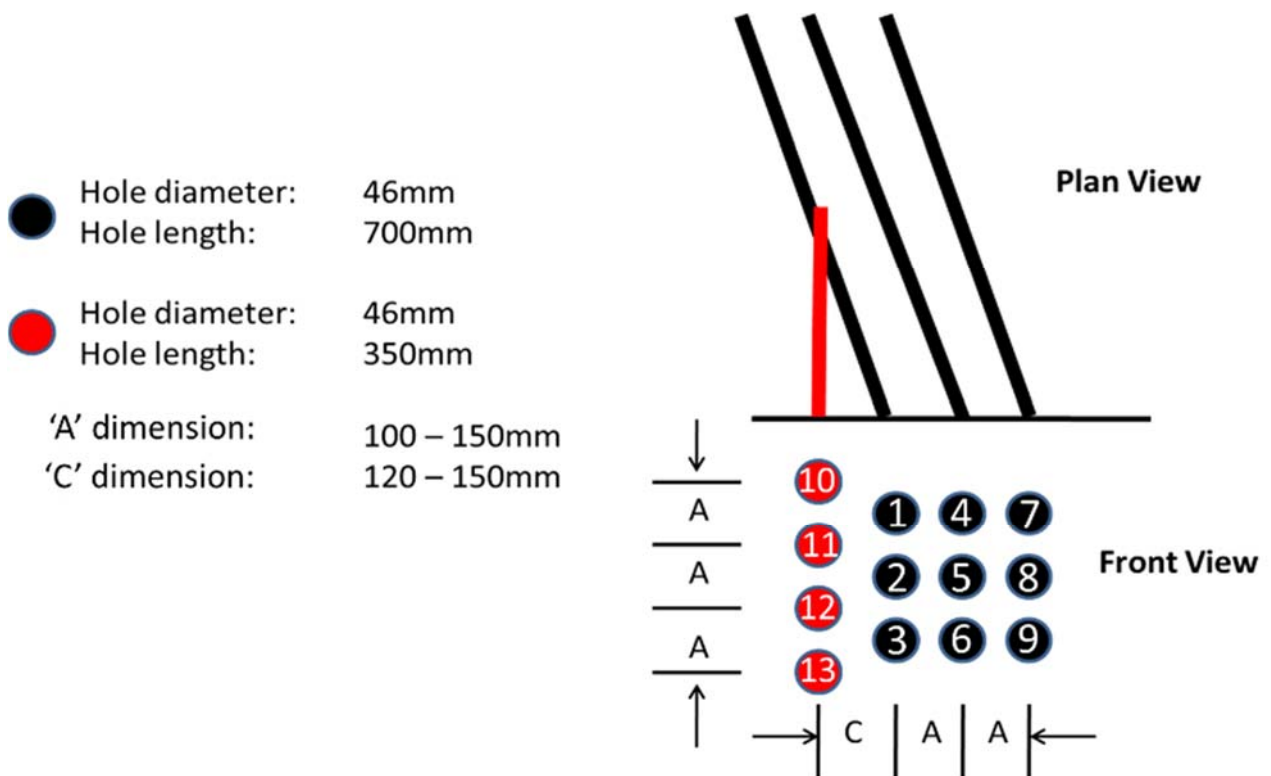
Figure 34: Nine-hole square pattern and four-hole non-splitter holes

3.5 The production or "slicing" holes

Two trials were carried out in an area of the stope where the second free face had previously been created by blasting. The rock face was scrutinised and the position of each hole was carefully selected and marked and drilled parallel to or as close to parallel to the second free face (as shown in Figure 28, position B above and as shown in Figure 35 below). No plans were presented for the production holes. Instead, the pattern was determined according to the face conditions. The distance between the slicing holes and the free face (burden) varied from 300 mm to a maximum of 500 mm. The results are discussed in the next section.

Figure 35: Free face slicing with visible drill barrels parallel to the free face (Author, 2013)

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4   RESULTS AND OBSERVATIONS

The observations of each individual test were evaluated and documented during the trials and further analysed in conjunction with the video footage. One critical common observation from all the tests was the dependence on accurate drilling regarding the collar position and direction of the hole. With divergence of the hole direction from the planned pattern, the amount of rock that has to be fractured and broken may increase, which reduces the performance of the splitter. Contrary holes spaced closer will increase the performance of the splitter; however, more drilling is required, which decreases the overall efficiency.

4.1   Cut hole layout

Hole layout 1

The DARDA® hydraulic splitter was inserted into the centre hole, "C". As the wedge was extended, no visual cracking or breaking of the rock mass was observed. The wedge was retracted and inserted into hole 2 of the pattern (see Figure 31 above). This hole was randomly selected, and as the splitter's centre wedge started extending, some minor cracking developed between the splitter hole and the centre hole. The DARDA® hydraulic splitter was then removed from the hole and inserted into hole number 3. As the wedge started protruding, more visible cracks developed towards hole 2 and the centre hole. The diameter of the centre hole decreased due to the fracturing in the centre of the "diamond" and the expansion into the centre hole of the drilled pattern. The same effect was observed when the splitter was inserted into holes 4 and 5. During the splitting process some small rock fragments fell to the footwall; however, the majority of the rock fragments had to be removed manually. The cut advance produced 30 kg of rock with an average depth of 36 cm (compared to the splitter's feather length of 45 cm).

Figure 36: The five parallel hole cut

Hole layout 2

The DARDA® hydraulic splitter was similarly inserted into the centre hole, "C". As per the five-hole diamond pattern (layout 1 above, Figure 31), initially no visible cracking or fracturing occurred. The splitter was inserted into hole 2 of the pattern. As the wedge was extended, visible hairline cracks developed towards the centre hole. Inserting the splitter into hole 3 saw increased cracking occurring, but not sufficient for fragments to fall easily to the footwall. The splitter was inserted into all the holes and random hairline cracking was noted. However, attempts to break and loosen the rock were unsuccessful. The splitter was then re-inserted in turn into each of the holes. During this process visible fracturing of the rock was noted and some small fragments fell to the footwall. The DARDA® hydraulic splitter was removed and the void area was cleaned by removing the loose fragments using a steel rod. This cut design produced 35 kg of rock fragments with an average depth of 38 cm.

Hole layout 3

In hole layout 3 the drilling was slightly more complex compared to hole layouts 1 and 2. The DARDA® hydraulic splitter was inserted into hole number 2 (see Figure 33 above) of the drilled pattern. Immediately after the centre wedge started protruding, a fracture developed diagonally across the centre hole, hole 9 and hole 5. Upon closer inspection it was evident that an existing fracture was present in the area where the holes were drilled.

This existing fracture assisted the splitting process, consequently all the holes into which the splitter was inserted generated cracks. Even in this case, however, fragments had to be removed by inserting a steel rod into the cut area and wedging the pieces out until the entire area was cleared. Thirty-eight kilograms of rock fragments were removed from the cut to an average depth of 35 cm.

Hole layout 4

In this particular test, the uneven nature of the face created difficulties during the drilling of the nine-hole pattern, as the holes were not collared on the marked positions as per the plan. This was, however, the only suitable area for the trial.

The DARDA® hydraulic splitter was initially inserted into hole 2 of the nine-hole grid (see Figure 34 above). As the centre wedge was extended, fractures developed between holes 2 and 11 and between holes 2 and 12. The same result was observed when the DARDA® hydraulic splitter was inserted in holes 1 and 3. Again small rock fragments fell to the footwall but the balance of the fragments had to be removed by hand. With the splitting of the grid holes, holes 4 to 8, difficult breaking was experienced. This was due to the large variation between the marked position and the actual drilled position of each hole. Hole 9 was abandoned due to the splitter not being able to create any fractures during the splitting process. Some 65 kg of rock was removed from the void area. The larger amount of rock removed compared to the other cuts above was due to the larger cross-sectional area drilled. The average depth of the cut was 28 cm.

4.2   Production or "slicing" holes

The hole layout designs at position B (see Figure 28) were not formally planned and the holes were marked and drilled by evaluating the free faces and the condition of the rock face. As can be seen in Figure 35, the measured distances from the free face to the DARDA® hydraulic rock splitter hole positions varied between 200 mm and 500 mm. Any existing cracks visible in the rock mass were used to assist in the breaking process. Compared to the "cut-hole" trials above, a relatively higher success level was achieved in breaking rock chunks towards the free face. The majority of the fragments broke on existing fractures. Initially the distance of the holes from the free face was set at 200 mm, but as the trials progressed and breakage was easily achieved, the distance was incrementally increased to 500 mm. Spalling of the rock into the second free face was quick and effortless

for the splitter and at the same time cracks appeared behind the row of holes which were exploited during the second round of drilling and splitting.

4.3   Discussion

The trials conducted underground identified the limits as well as the potential of the DARDA® hydraulic splitter in this application. A number of lessons were also learned, including the need for drilling accuracy. Operational aspects of the DARDA® hydraulic splitter and the results achieved are discussed in the following section.

4.4   Cut hole trials

As mentioned above, creating the cut or second free face is a crucial step allowing subsequent efficient rock breaking into the newly established free face. Within the confines of the various cut patterns trialled in this investigation, difficulty was encountered with the use of the splitter. Of the four patterns tested, none was shown to be significantly better than the other three in the ease of establishing the initial cut. Certain factors need to be considered in the design of the cut, particularly the need to minimise the number of holes required so that drilling time is kept to a minimum. The 5-hole burn cut was drilled in the shortest time, but produced the least amount of broken rock. The greater the initial cross-sectional area of the cut, the easier it is to split the following holes (which now become "slicing" holes). The pattern of holes extending radially outwards around the cut area can then be wider, i.e. with holes further apart. Furthermore, by increasing the cross-sectional area of the cut, broken fragments are removed more easily from the void created. In general, however, creating an initial second face or "cut" using the splitter has been found to be difficult to achieve and none of the four cut patterns tested were found to give satisfactory results. A possible solution would be to increase the number of holes and decrease the distance between them to create the initial cut. An alternative would be to increase the hole diameter of non-splitting holes. This aspect needs further investigation.

4.3.2 Slicing hole trials

The slicing operation was found to be more amenable to the use of the splitter. The tests proved that provided a second free face has been established, the use of the splitter becomes more viable and efficient. No difficulties were experienced with rock fragments being detached after cracking and fracturing had occurred.

Figure 37: Illustration of rock fracturing during slicing (Author, 2013)

4.5 General discussion – Trial experiences

4.5.1 Drilling accuracy

Drilling holes in the rock face for the use of the splitter posed some specific challenges compared to drilling for blasting operations. These issues could, however, be fairly easily rectified and maintained. Hole length is crucial as the DARDA® hydraulic splitter extends the centre wedge into the hole and it should be able to move freely and not be obstructed by the toe of the drilled hole. A hole that is drilled too short will destroy the centre wedge. The hole diameter should be equal to or slightly greater than the specified hole diameter of 46 mm. Inserting the splitter into a hole with a diameter of less than 46 mm is not possible. Holes need to be drilled as straight as possible. This was highlighted when an early test hole deviated from the planned direction and the splitter centre wedge was consequently bent. Furthermore, as mentioned above, inaccurate drilling can drastically affect the performance of the splitter. Figure 38 shows the feathers and the wedge bent after the DARDA® hydraulic splitter was inserted into a drilled hole which deflected.

Figure 38: Feathers and wedge bent due to a deflected drilled hole (Author, 2013).

4.4.2 Breaking cycle:

Although no detailed time studies were conducted during the trials, several relevant time periods were noted for interest and future reference (see Table 3 below).

Table 3: General time periods required for various DARDA® functions experienced during the trials

Manhandling of *DARDA®* from one hole to the next	17 – 50 seconds
Protruding wedge	53 – 60 seconds
Retracting wedge	44 – 47 seconds
Greasing blades	20 – 55 seconds

4.5.2 DARDA® splitter operational learning

Several challenges were experienced with the equipment during the initial trials, causing the cycle to take longer than planned.

- Firstly, proper lubrication of the feathers and wedge is critical, and in these trials the feathers and wedge were lubricated after approximately every fifth hole. The feathers

were manually opened and the lubricant was squeezed between the moving parts (see Figure 39).This was time consuming and slowed down the entire process. It is suggested that this process be automated by installing a greasing device in the DARDA® hydraulic splitter, which may also increase the usable life of the moving parts.

Figure 39: Greasing of the centre wedge and feathers (Author, 2013).

- The DARDA® hydraulic splitter also had to be supported when inserted into the hole. During the fracturing process, it was found that as the rock broke away from the hole perimeter, the feathers were exposed and the unit fell to the ground. The unit was supported by a steel hook on the handle of the splitter and suspended from the safety netting used in the stope. A supporting rig would be advisable (see Figure 40).

Figure 40: DARDA hydraulic splitter suspended with a hook on the safety netting (Author, 2013)

- On a few occasions some small rock fragments were caught between the feathers and the wedge as the splitter was withdrawn from a hole. These fragments stuck to the lubrication agent as seen in Figure 41 and needed to be removed manually before the wedges could be inserted into the next hole.

Figure 41: Small rock fragment wedged between the feather and the centre wedge (Author, 2013)

- When the rock mass did not give way during fracturing, removing the DARDA® hydraulic splitter from the hole in some cases was extremely difficult, and a pinch bar was required to open fractures and allow the wedges to be dislodged.

- On a few occasions the splitter's feathers did not "grip" against the perimeter of the hole, and as the wedge was extended the DARDA® hydraulic splitter moved backwards.

- During the fracturing process the rock fragments on the free face generally fell to the foot wall. However, the fragments deep inside the void area had to be removed using a steel rod and the last remaining fragments were removed by hand. An improved cut design could alleviate this issue.

- The splitting process should start from the footwall of the stope face and progress towards the hanging wall. In this way the drilled holes in the foot wall are not obscured by rock fragments.

During most of the trials fractures did not develop immediately after the wedge of the DARDA® hydraulic splitter was extended. In some cases it took two or three attempts before fracturing was observed between the splitting hole and an adjacent hole or the second free face.

Figure 42: Rock fragments visible after the splitting process (Author, 2013)

Figure 42 shows the representative fragment sizes produced during both the cut hole trials and the slicing trials. The fragment sizes are ideal for cleaning after the breaking process.

5   CONCLUSIONS

With some improvements to the equipment and technique, rock breaking with the use of the splitter could well have a place in an underground mining operation. It is suitable for niche applications, such as (but not limited to) in areas where conventional drilling and blasting is prohibited, and in removing safety pillars and shaft pillars. The system could possibly be automated and can be used in areas where seismic events are rife.

The static handheld tool can be easily manhandled in restricted spaces. It is also simple in design, can be easily integrated into existing mining operations and infrastructure, and neither does it require a technically competent or skilled workforce, or expensive maintenance.

The trials showed the shortcomings of the equipment in developing the initial cut or second free face. Furthermore, the use of the splitter is highly dependent on accurate drilling. The splitting becomes more effective once a second free face is present.

6   RECOMMENDATIONS

The following recommendations are made regarding the equipment used and the technique applied during the trials.

- The greasing of the wedge and feathers should be automated to increase the available times for the DARDA® hydraulic rock splitter to split rock.

- The DARDA® hydraulic rock splitter should be supported during splitting. This support mechanism may be in the form of a light-weight tripod or hooks which are suspend from the hanging wall.

- Small stones may lodge between the wedges and feathers after rock fragments have been broken free. The wedged stone needs to be cleared before the DARDA® hydraulic rock splitter can be inserted into the drilled hole.

- On a few occasions the feathers did not grip against the perimeter of the drilled hole. The outside surface of the feathers should be machined to increase the friction between the feathers and the rock surface.

- The cut design hole layout should be such that all the holes are drilled parallel, which means the holes can be deepened. Angled holes will "distort" the cut design when the holes are deepened.

- The cross-sectional area of the cut should be as large as possible to assist with the clearance of the rock fragments.

- On a few occasions the DARDA® hydraulic rock splitter could not be manually removed due to the wedge and feathers being jammed in the fractured rock mass. The wedge needs to be protruded and retracted several times to loosen the unit.

- The breaking process should be started from the footwall and progress towards the hanging wall so that the rock fragments falling to the floor do not conceal the drilled holes ready for splitting.

7   SUGGESTIONS FOR FURTHER WORK

The suggestions for further work are as follows:

- Conducting trials with the DARDA® hydraulic splitter in the reef horizon.

- Making use of two DARDA® hydraulic splitters in series.

- Investigating the effect in the stope of seismic events during the slow advance rate of rock splitting.

- Use of the effect of inherent rock stress at deep levels to assist during the rock splitting process.

- Investigation of the layout of "cut" holes, drilling a "slot" of closely spaced parallel holes adjacent to each other.

- Conducting of trials where larger diameter holes are drilled (non-splitting holes) to improve the cross-section of the void for improved rock fracturing.

Proposed cut designs not trialled

The following cuts were proposed to be drilled and the DARDA® hydraulic splitter was inserted into the holes shown to fracture the "cut" area.
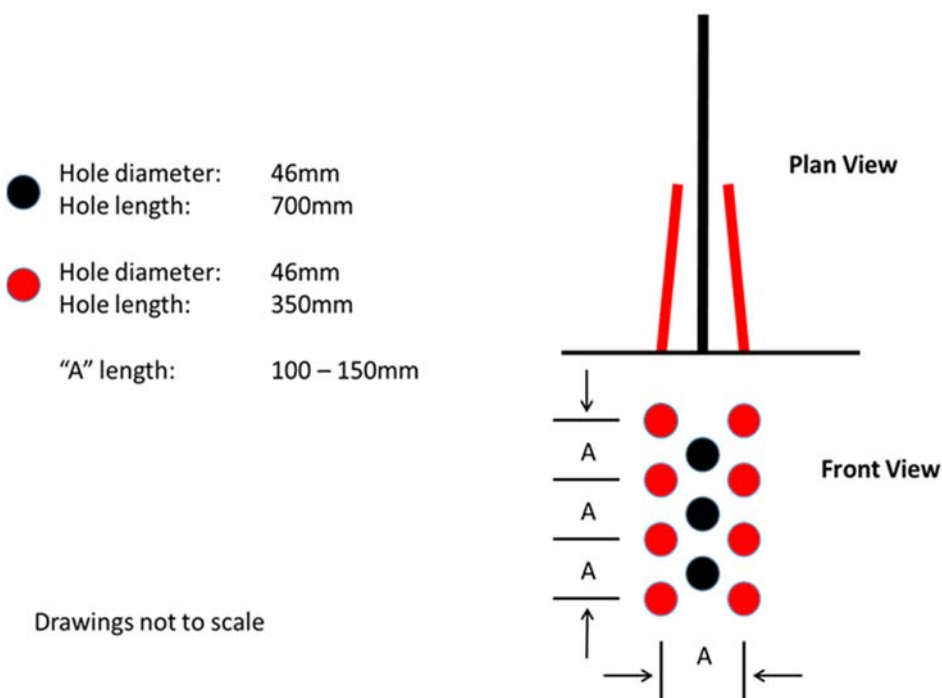

Figure 43: Hole layout - Cut 5

The above proposed design is fairly similar to cut 3, Figure 33 described in Chapter 3 and Chapter 4 of the document. The main difference is that the hole lengths are different but the hole pattern layout is similar. The outer two rows of holes are drilled short and the centre row of holes are drilled to 700 mm. The DARDA® hydraulic splitter will be placed in the centre row of holes to fracture the rock outwards into the openings of the short drilled holes. The angled holes are drilled between 84 ° and 87° to the face.

The centre row holes are used for the hydraulic splitter and fracturing can occur anywhere in a 360º area around the hole, depending on the rock mass jointing and/or the rock weaknesses. Once the rock in the cut area has been removed can the centre row of holes be deepened with the rock drill to create the follow-on cut, which will reduce drilling time.

Hydraulic splitter – cut pattern #6

Hole diameter: 46mm
Hole length: 700mm

Hole diameter: 57mm
Hole length: 350mm

"A" length: 100 – 150mm

Drawings not to scale

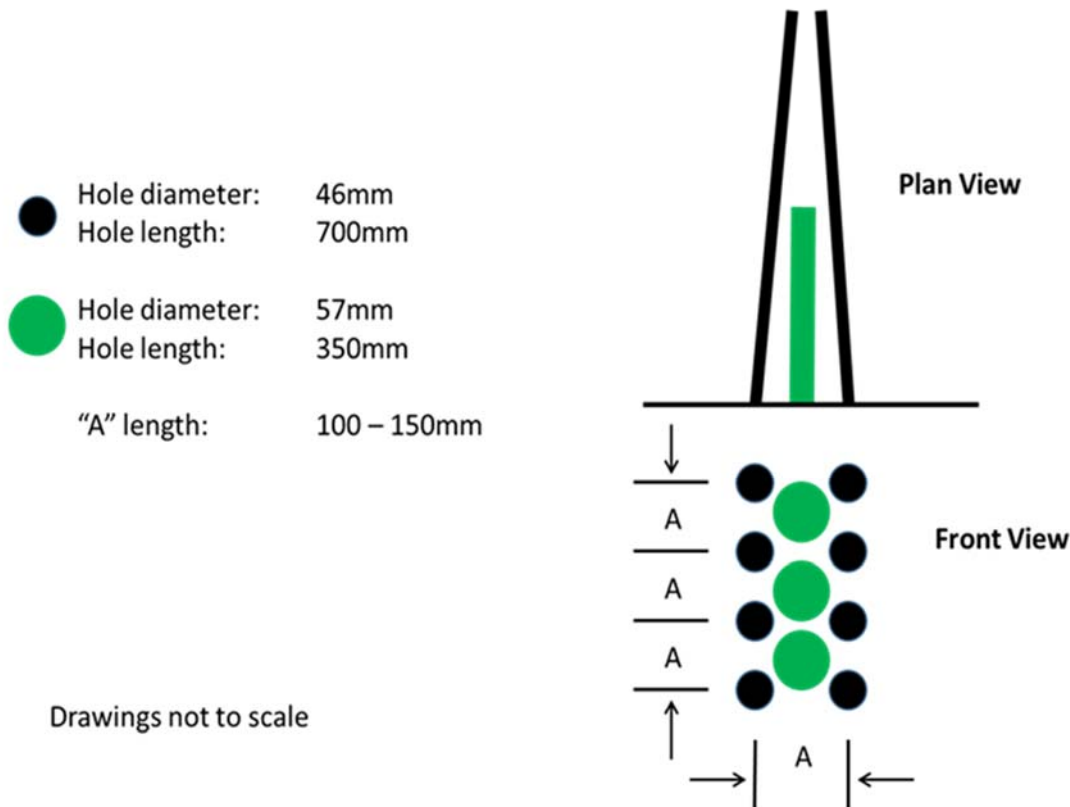Plan View

Front View

A

A

A

A

Figure 44: Hole layout - Cut 6

Figure 44 shows a staggered pattern of holes drilled with two different diameter drill bits. The centre row of holes are drilled with a larger bit, i.e. 57 mm, and the holes on the outside are drilled to 700 mm with a 46 mm diameter drill bit. The DARDA® hydraulic splitter will be placed in the outer holes and will break inwards to the centre row of holes. The angled holes are drilled between 84° and 87° to the face.

The centre row of holes are drilled with a larger diameter drill bit. This gives an increased void area in the rock mass for the hydraulic splitter to break into. The drilling will take slightly longer, but it is envisaged that the breaking of the two outer vertical rows of holes into the centre row of holes will be easier. Once the rock in the cut area has been removed and the rock mass broken into the cut area, a row of vertical holes can be re-used to create the following cut into the face.
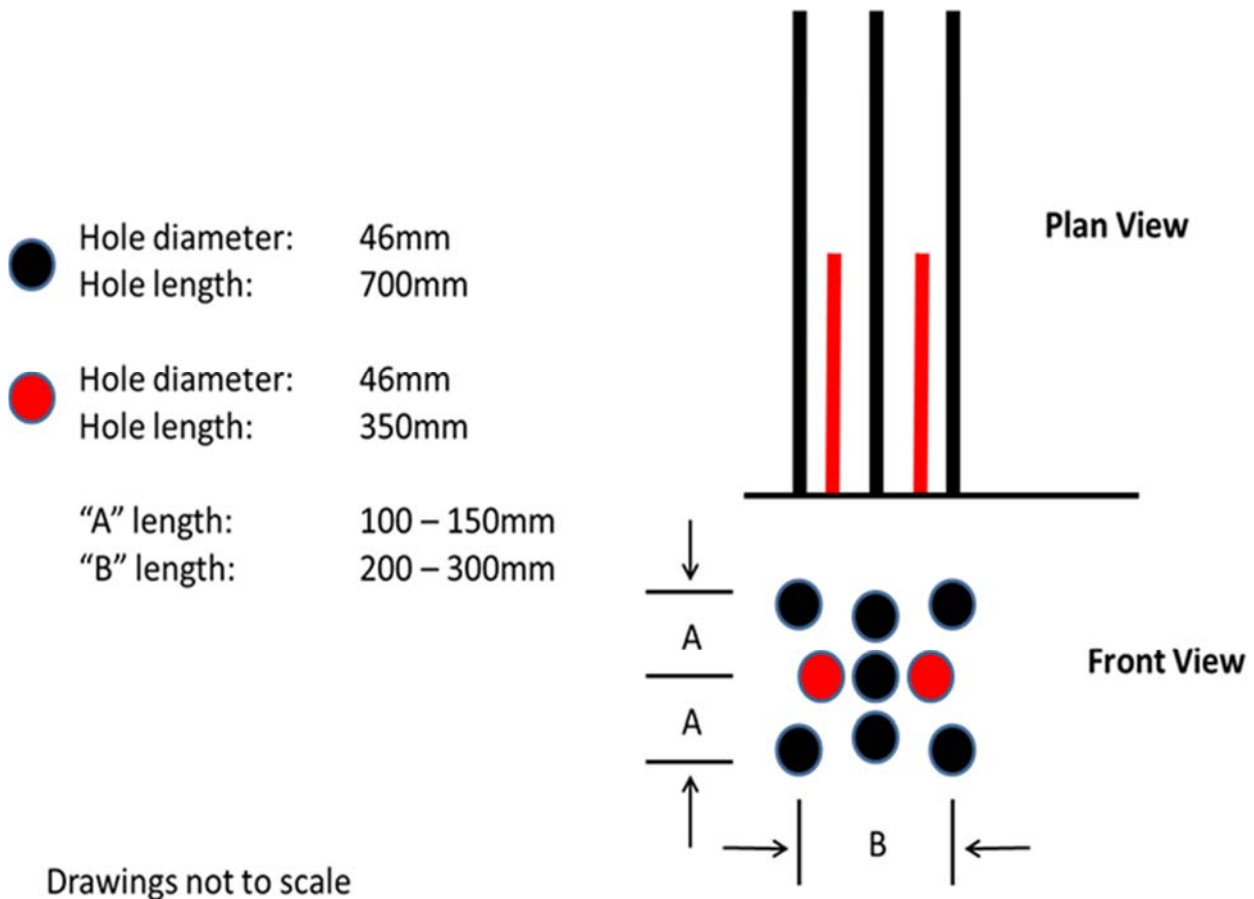
Figure 45: Hole layout - Cut 7

A total of nine holes are drilled, seven holes are drilled to 700 mm and two holes are drilled to 350 mm. The centre row of holes will be the main holes for the DARDA® hydraulic splitter. The four corner holes are drilled off the square cross pattern in the centre. All the holes are drilled with a 46 mm drill bit. The two short holes are non-splitting holes and are used to assist in easier breaking.

The two extra short drilled holes will assist the breaking process as the rock mass could expand into this area. A number of holes are drilled and these can be re-used to extend the cut. The rock mass properties and the existing joints will determine in which holes the hydraulic splitter will be used.
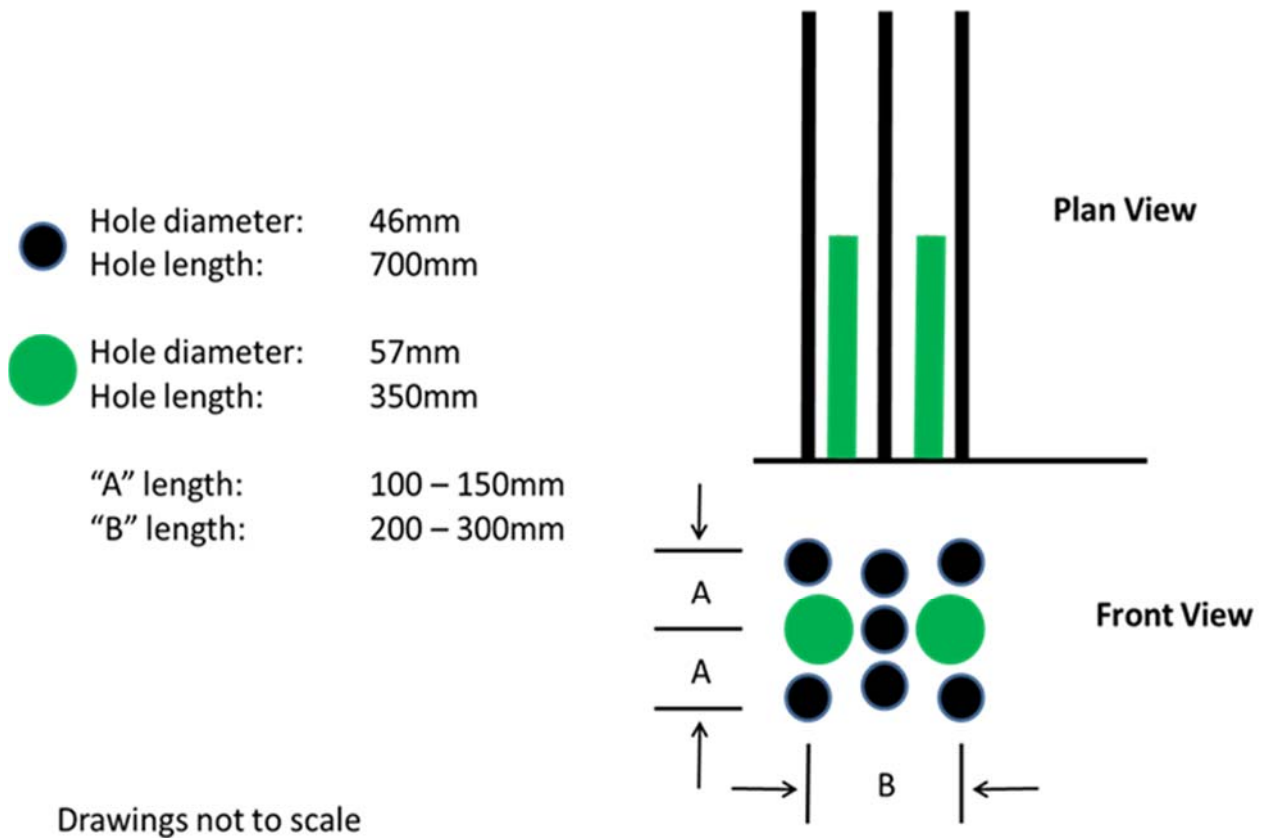
Figure 46: Hole layout - Cut 8

Figure 45 and Figure 46 are fairly similar except for the two short holes that are drilled with a larger diameter to create a bigger opening for the rock to expand into. Hole lengths and angles are similar to those shown in Figure 45.

Two larger holes are drilled, creating a larger space for the fractured rock to expand into. The small diameter holes will be re-used to extend the cut area. All the holes are drilled parallel, making it easier to drill.

Hydraulic splitter – cut pattern # 9

Hole diameter: 46mm
Hole length: 700mm

"A" length: 100 – 150mm
"B" length: 200 – 300mm

Plan View

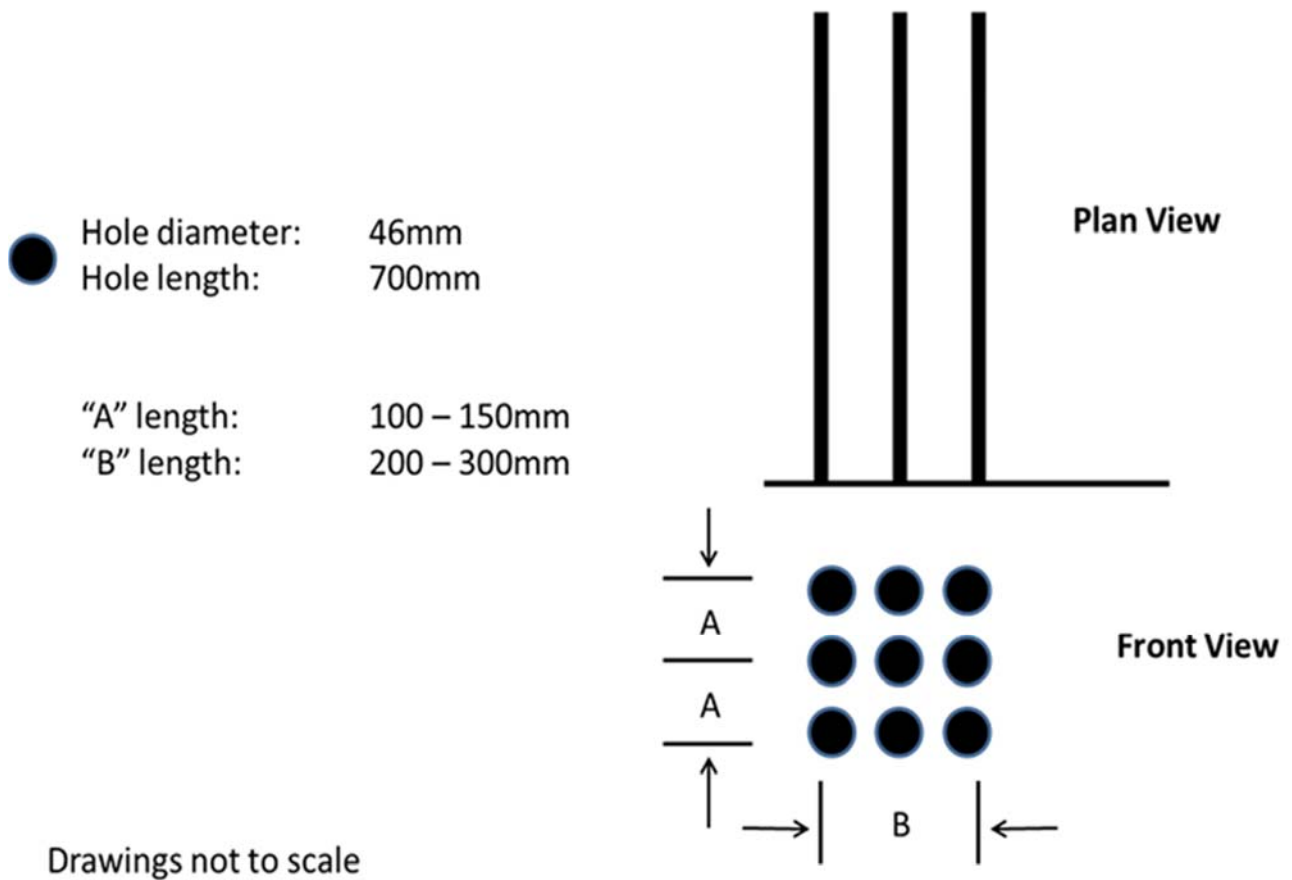Front View

A

A

B

Drawings not to scale

Figure 47: Hole layout - Cut 9

Figure 47 shows a nine-hole cut; all the holes (46 mm diameter) are drilled to 700 mm and perpendicular to the face. The DARDA® hydraulic splitter will be inserted into the centre row of holes to expand the rock into the neighbouring row of holes. To improve efficiency, the DARDA® hydraulic splitter could be inserted into any of the holes as they are all of the same length. The hole can be re-used to extend the follow-on cut.

Hydraulic splitter – cut pattern # 10

Hole diameter: 46mm
Hole length: 700mm

Hole diameter: 46mm
Hole length: 350mm

"A" length: 100 – 150mm
"B" length: 200 – 300mm
"C" length: 120 – 150mm

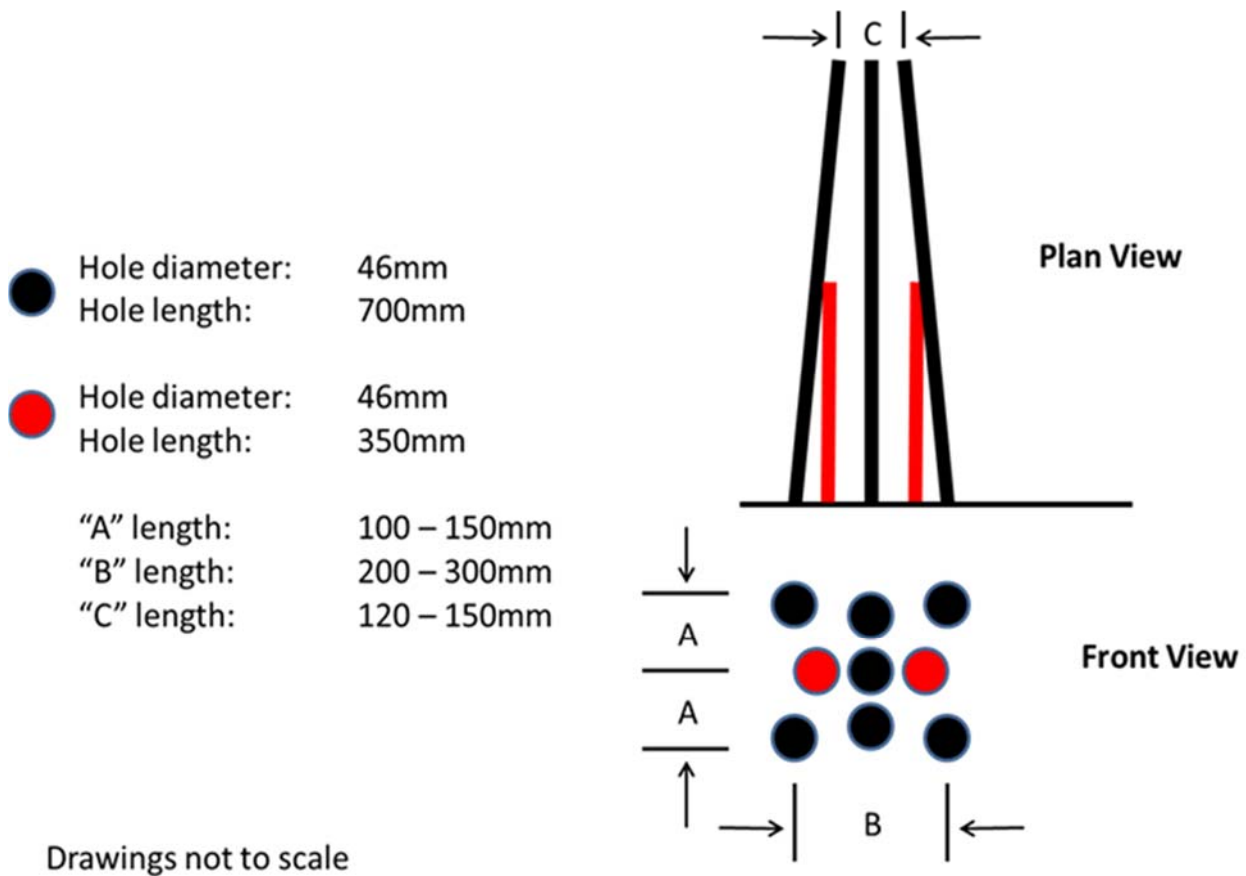**Plan View**

Front View

Drawings not to scale

Figure 48: Hole layout - Cut 10.

Figure 48 shows a similar pattern of holes as in Figure 43, Cut 5. The main difference is that the corner holes converge towards the centre. These corner holes are drilled 700 mm in length and can be used by the DARDA® hydraulic splitter to fracture the rock towards the centre of the drilled pattern. Two holes are drilled short to create some space for the rock to expand into. The angled holes are drilled between 84º and 87º to the face. The parallel holes can be re-used to drill deeper for the follow-on cut.

Hydraulic splitter – cut pattern # 11

Hole diameter: 46mm
Hole length: 700mm

Hole diameter: 57mm
Hole length: 350mm

"A" length: 100 – 150mm
"B" length: 200 – 300mm
"C" length: 120 – 150mm

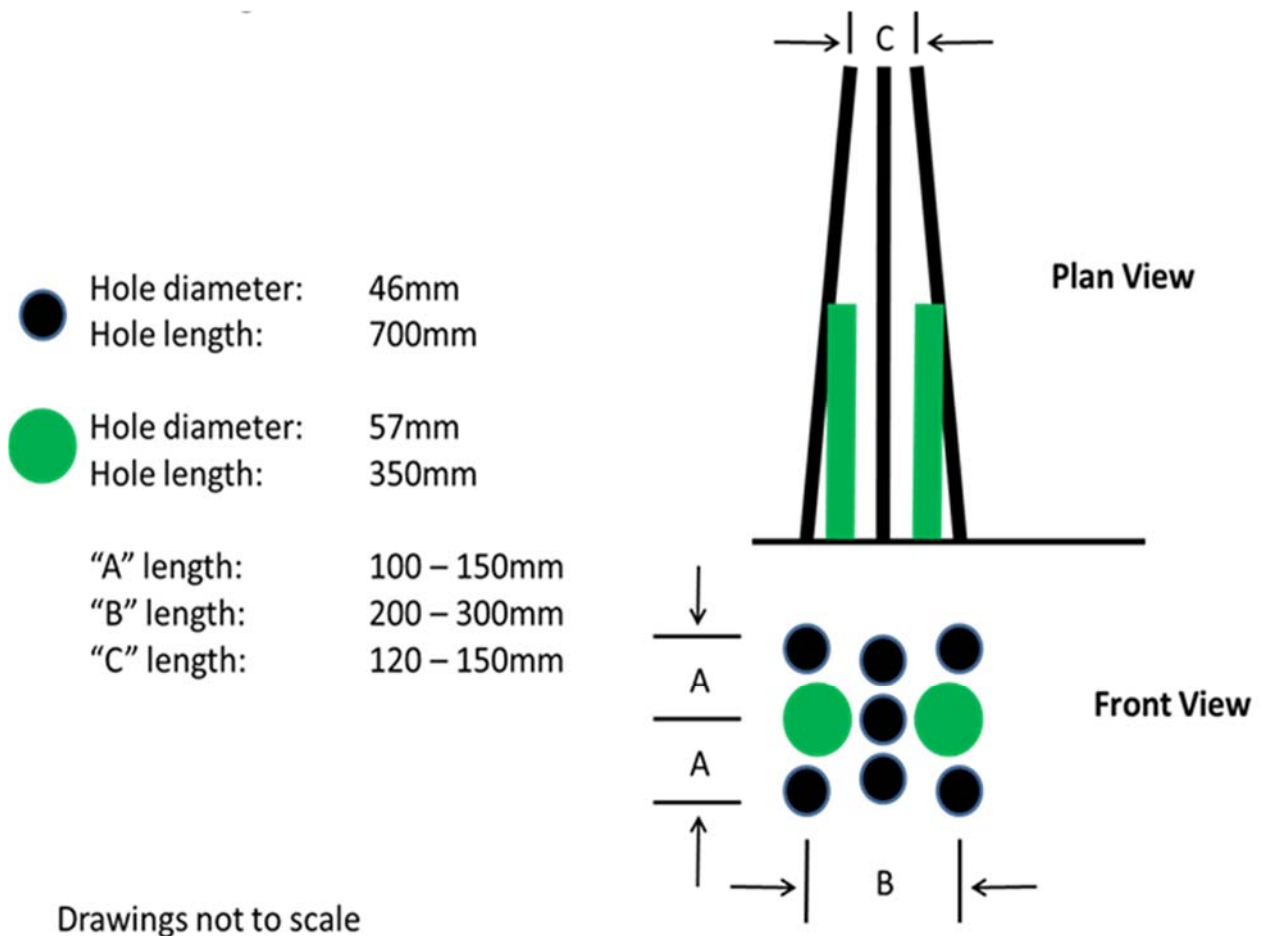Drawings not to scale

Plan View

Front View

Figure 49: Hole layout - Cut 11.

Figure 49 shows a pattern with two drill bit diameters, namely 46 mm and 57 mm. All the 46 mm diameter holes are drilled to 700 mm, and the larger diameter 57 mm holes are only drilled to 350 mm. The larger holes will create an opening for the rock to expand into. The corner holes converge at the centre of the pattern. The angled holes are drilled between 84° and 87° to the face. The parallel holes can be re-used to drill deeper for the follow-on cut. This includes the 57 mm holes.

Hydraulic splitter – cut pattern # 12

Hole diameter: 46mm
Hole length: 700mm

Hole diameter: 57mm
Hole length: 350mm

"A" length: 120 – 150mm
"C" length: 120 – 150mm

Drawings not to scale

Plan View

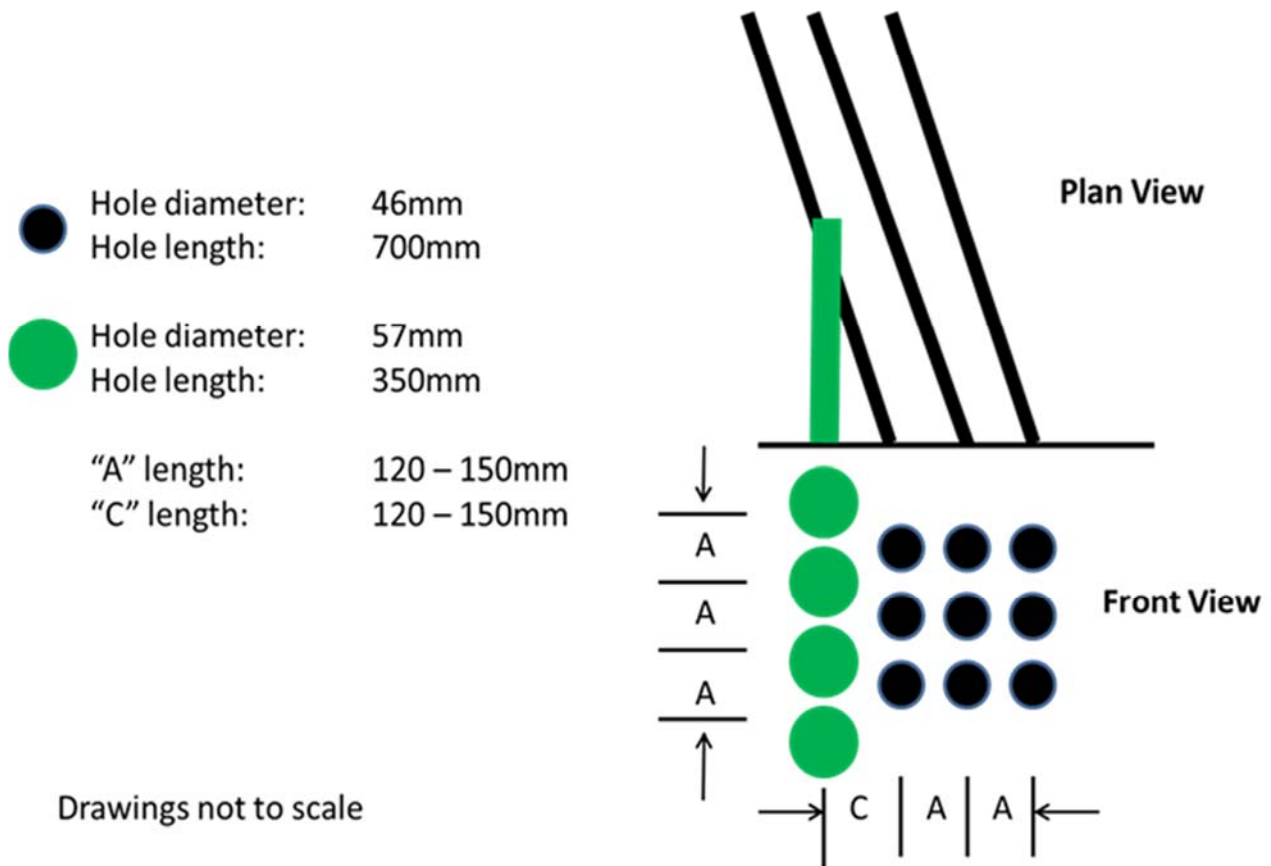Front View

A

A

A

C | A | A

Figure 50: Hole layout – Cut 12.

Figure 50 shows a similar drilling pattern as depicted in Figure 34, Cut 3, except for the holes drilled perpendicular to the rock mass. The diameter of these holes is increased to 57 mm. The angled holes are drilled at angles of 67° to 71° to the face. The angled holes will expand into the larger holes. This is a combination of angled holes and straight holes. The straight holes are drilled parallel and to a length of 350 mm. The angled holes are drilled to a full length (700 mm). The short hole's pattern and the longer hole patterns are staggered. All the holes in the two different patterns are drilled parallel. The angled holes are drilled at an angle of 67° to 71° to the face. This is a slightly more difficult pattern to drill accurately.

Hydraulic splitter – cut pattern # 13

Hole diameter: 46mm
Hole length: 700mm

Hole diameter: 46mm
Hole length: 350mm

"A" length: 100 – 150mm
"C" length: 120 – 150mm

Plan View

Front View
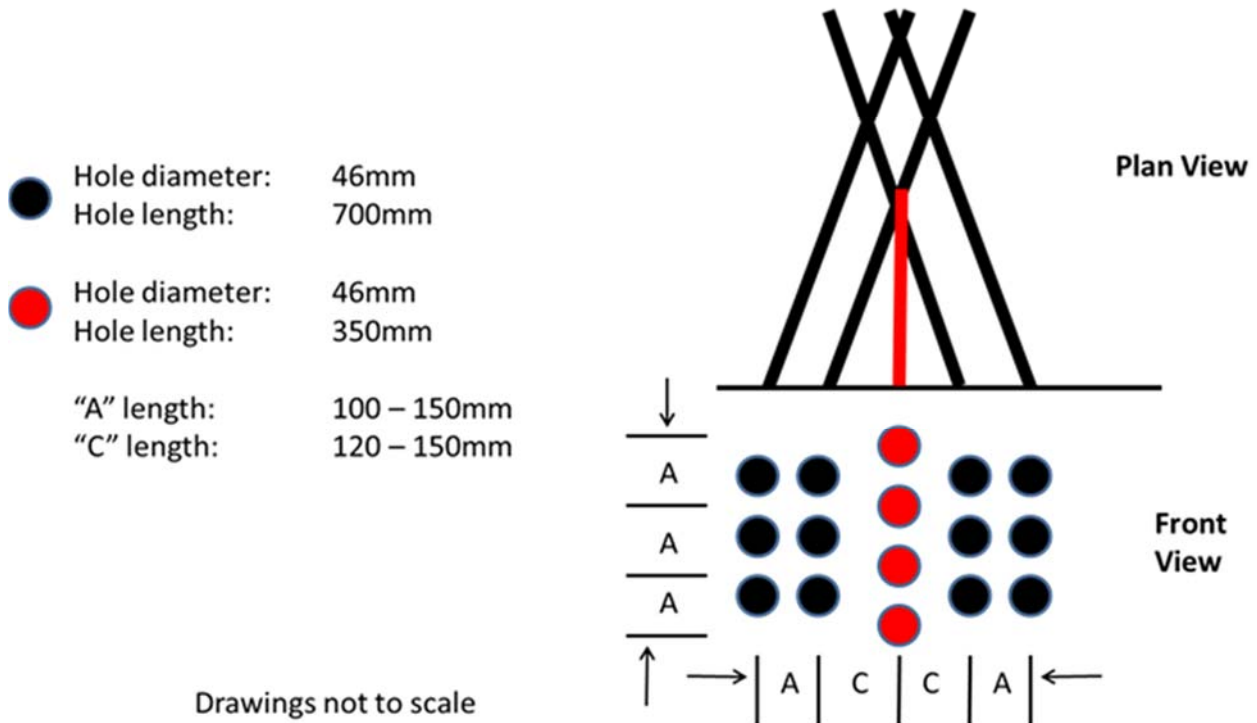
A
A
A

A | C | C | A

Drawings not to scale

Figure 51: Hole layout - Cut 13

Figure 51 shows a drill pattern with two main groups of holes, one row drilled straight into the face, 350 mm long. The other group is drilled in a square pattern on either side of the row of holes. The holes are drilled parallel to one another and at an angle of 67º to 71º to the face. These holes are drilled 700 mm long. Drilling is slightly more complex. It is expected that a "wedge" will be created.

Hydraulic splitter – cut pattern # 14

Hole diameter: 46mm
Hole length: 700mm

Hole diameter: 57mm
Hole length: 350mm

"A" length: 120 – 150mm
"C" length: 120 – 150mm

Plan View

Front View

A

A

A

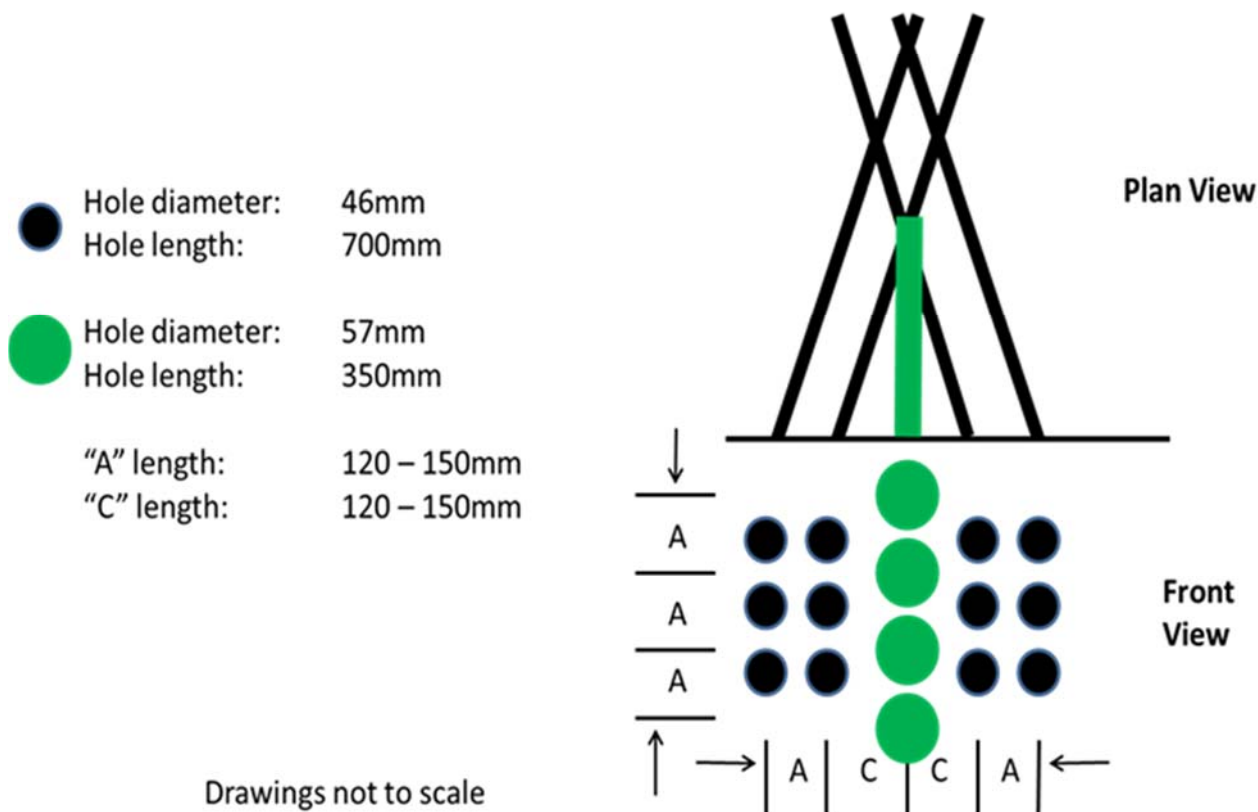A C C A

Drawings not to scale

Figure 52: Hole layout - Cut 14

Figure 52 shows a similar pattern to the pattern illustrated in Figure 51, except for the straight 350 mm holes. The diameter of this row of holes has been increased to 57 mm. The angled holes are drilled on either side of the larger diameter holes at an angle of 67º to 71º to the face. Drilling is slightly more complex. It is expected that a wedge will break. The rock will expand into the bigger diameter holes.
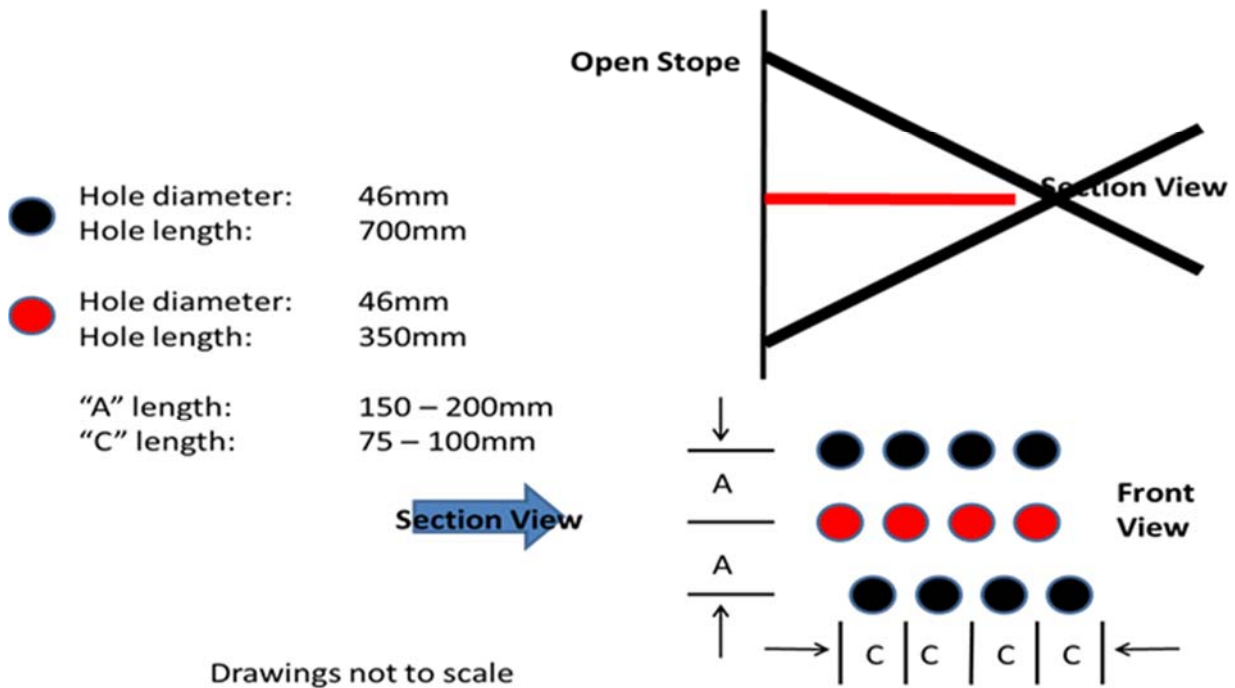
Figure 53: Hole layout - Cut 15

Figure 53 shows that the holes are drilled into the vertical rock face. The holes are drilled at an angle relative to the horizon. The short holes (350 mm) are drilled horizontal and the longer holes are drilled at a negative angle and at a positive angle relative to the short holes. The long holes are staggered so that they do not intersect each other. The short holes are all drilled parallel. The angles of the holes are between 60° and 67° relative to the vertical face, drilled either at a negative angle or a positive angle.
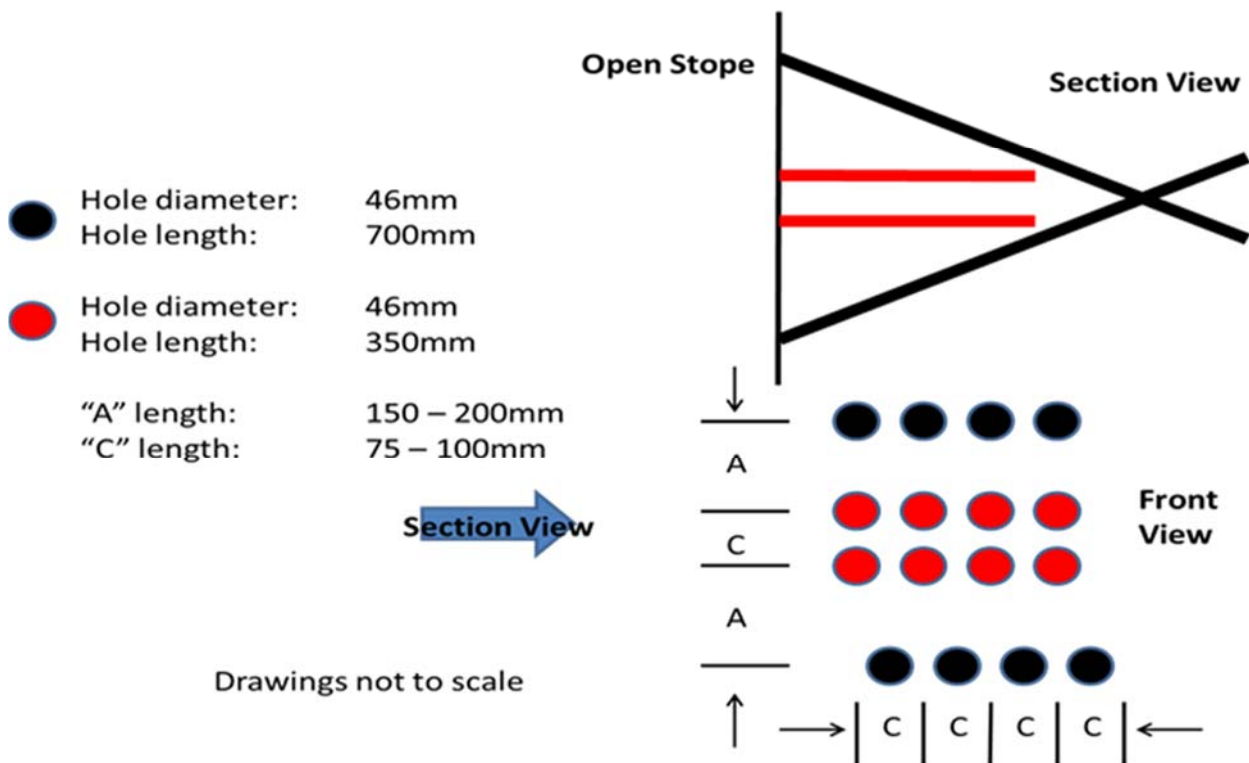
Figure 54: Hole layout - Cut 16

Figure 54 shows a similar drilling pattern to Figure 53, Cut 15, except for an extra row of short holes drilled in a square grid at 90° to the vertical face. The angles of the long holes drilled into the face are between 60° and 67° relative to the vertical face, drilled either at a negative angle or a positive angle. The short holes are drilled with the same diameter drill bit. The expansion of the rock will break into the double row of short holes. It is expected that a wedge will be created.