REVIEW OF PAST WORK

ROCKBREAKING

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STOPING REVIEW OF PAST WORK - ROCKBREAKING

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PREFACE

Rockbreaking is the most basic of the operations required in mining and over the years has been the subject of extensive research and development. This document is a summary of the work carried out by the Chamber of Mines Research Organization supplemented by studies of work carried out by others, both in South Africa and overseas. The document has been prepared as part of a process of critically reviewing what progress has been made in solving the Stoping Problem and forms the basis for determining the direction for future research and development.

The work is divided into the three main groupings of breaking rock with explosives, rockdrilling, and non-explosive methods of rockbreaking. Each chapter has been prepared by different authors, each an expert in his own field. The quality of some of the reproductions of the illustrations is poor due to them having been taken from old reports.

This document will be of interest to persons engaged in improving methods of mining hard rock in narrow stopes.

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ROCKBREAKING - SCHEME

- ROCKBREAKING
  - Cyclic mining (with explosives)
  - Rockdrilling
    - Fundamentals
    - Hand operated
    - Mounted
  - Continuous Mining
    - With explosives - AECI continuous drill/blast system (see Intergrated Mechanized Systems)
    - Non-explosive

CHAPTER 1

BLASTING AND BLASTING METHODS

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BLASTING AND BLASTING METHODS (CYCLIC MINING) - SCHEME

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    - Slurry
    - Emulsion
  - Initiation Systems
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INTRODUCTION TO CYCLICAL BLASTING

Stoping is the primary unit operation in gold mining and rockbreaking is the fundamental stoping function. Blasting has served as the only economic means of rockbreaking since gold was discovered on the Witwatersrand over 100 years ago. Each day the gold mining industry fires well in excess of 500 000 stoping blastholes.

The largest drawback to explosives is that they all produce noxious fumes. Hence the mines are evacuated of all personnel before blasting and a few hours are required to clear the explosion gases from workings before re-entry of workers. These requirements impose a rigid cyclical nature on the entire mining operation and are the cause of a large unproductive period of time between the completion of face preparations for blasting and the start of cleaning operations. A practical method of continuous mining based on explosives could revolutionize mining operations. Refuge bays, fumeless explosives and changes to mine and ventilation layouts are potential elements of such a method.

The cost and productivity of conventional stoping operations are dependent on virtually all the products and side effects of blasting. There are also large potential benefits for conventional mining if blasting is controlled as part of an integrated stoping system.

The following reviews the important features of past and present work in the areas of applying new explosive types, developing improved initiation systems, investigating new explosive rockbreaking methods, and understanding better the explosive rockbreakage process.

1 EXPLOSIVE TYPES

The traditional explosive type used in South African gold mines has been nitroglycerine (NG) based dynamites. An alternative explosive consisting of ammonium nitrate prills mixed with a carbonaceous fuel (ANBA) was introduced in the late 1950's and through widespread collaboration between COMRO and other parties ANBA was widely used by gold mines by the mid 1960's. The advantages of ANBA relative to NG explosives are that it is simpler and safer to manufacture, safe to handle and use and is significantly less costly. However ANBA explosives are not resistant to water and must be used only in dry holes.

Water resistant slurry explosives (water gels) were introduced in the late 1970's and COMRO conducted extensive trials to establish their performance characteristics, usage aspects and economics relative to NG and ANFO (ANBA) explosives. With the success of slurry explosives came the introduction of competition to the explosives market when National Explosives began marketing Tovex. Prior to this an exclusive explosives contract existed between the mines and AECI, dating back to the early 1900's.

The latest generation of explosives, emulsions, have been available to gold mines since 1985. The future of explosive appears to be in the form of slurry and emulsions due to ease and safety in manufacturing, resistance to water, safety in handling and use, non-headache properties, reduced fume production and versatility in formulation to achieve various performance characteristics. The development of an explosive that produces no fumes is very attractive from the perspective of continuous mining operations. Explosives companies worldwide are pursuing this goal vigorously and have done so for decades. The difficulty lies in controlling the reaction totally and in dealing with external substances, such as water in holes, participating in the reaction. The NOX and CO fume characteristics of emulsion and slurry explosives are approximately one-tenth (≈5 ppm) and one-half (≈550 ppm) of the respective amounts produced by NG explosives (≈50 ppm) and (≈950 ppm).

1.1 ANBA (Ammonium Nitrate Blasting Agents)

The introduction in the early 1950's of ANBA explosives in the USA made available a relatively safe, simple and cheap form of explosive energy for blasting. Prior to this nitroglycerine based explosives were used almost universally for all types of rock blasting. Overseas developments in ANBA during the 1950's centered mainly around applications in the large diameter holes used in surface blasting. Safety and cost advantages of ANBA aroused local interest in gold mining applications. During the late 1950's and early 1960's there was collaboration by COMRO, the mining department of the University of the Witwatersrand, African Explosives and Chemicals Ltd (AECI) and a number of mining companies with the following major objectives:

(i) identify the performance characteristics and economic benefits of using ANBA, instead of conventional NG explosives, in small diameter holes (28-42 mm) in gold mines.

(ii) determine the most cost effective form of ANBA (i.e. porous or dense prills. At this time only dense prill manufacturing facilities exist in South Africa).

(iii) develop the equipment needed to charge small diameter holes.

(iv) compile a code of practice on ANBA for use by the mines.

The outcome of this work was fruitful and is summarised as follows (1-5):

(i) it was found that the performance characteristics of ANBA in small diameter holes compared favourably to that of NG explosives. Owing to a better filling of holes and therefore more energy per metre of hole, ANBA explosives were capable of producing a larger magnitude strain pulse and equal or greater breakage.

(ii) field studies indicated that both dense and porous prilled ANBA products were very effective and were cheaper to use than NG explosives. In addition, it was shown that porous prills were more effective than dense prilled ANBA. (This is due to a more even distribution of the fuel component in the final mix as well as to greater surface area when porous prills are used. This offers a more energetic and reliable reaction). As only dense prilled ANBA was available in South Africa at that time, the justification for a factory to manufacture porous prills was supported by these tests and was subsequently built by AECI.

(iii) safe forms of loading ANBA explosives were devised.

(iv) this work culminated with a symposium on ANBA conducted by the SAIMM being held during 1964 at which was issued a code of practice for the underground use of ANBA.

At present ANBA, now more commonly known generally as ANFO (ammonium nitrate/fuel oil), and by a local trade name ANFEX, accounts for about 30-40% by mass of the explosives used in gold mining. It is marketed by AECI and Sasol Explosives (SMX) in a porous prilled form.

Although ANFO explosives are cheaper there is a tendency towards over charging holes and wastage due to its bulk form. This can lead to excessive damage to surrounding rock (hangingwall and footwall) and a diminished cost advantage over other explosives. Another disadvantage of ANFO explosives is their lack of resistance to water.

1.2 Slurry Explosives

One method to render ammonium nitrate products water resistant is to physically shield them from water. Slurry explosives, also commonly known as water gels, evolved out of research in this area. They consist of more or less fluid compositions where ammonium nitrate is dispersed in a gelatinous sensitizer/fuel matrix. Advantages over NG explosives include non-headache properties, less fumes produced (especially NO, NO_2) and reduced sensitivity to initiation and therefore increased safety in handling and use.

Slurry explosives for small diameter holes (Sinex) were introduced to gold mines by AECI in the mid 1970's. Soon after this the explosives monopoly which AECI held was challenged by the National Explosives company. They proposed to manufacture locally and sell a slurry explosive known as Tovex under licence to Du Pont in the USA.

COMRO conducted extensive field tests on slurry explosives to assess economic, performance and operational aspects as compared to NG and ANFO explosives. These studies focused on underground, small hole diameter applications in coal, gold, platinum and chrome mines. However trials at surface mines and quarries were also conducted.

The overall outcome of these trials was that slurry explosives were accepted by the underground mining industry. It was found that the general performance, breakage capacity and economics of slurry explosives compared favourably with the NG and ANFO explosives in use. The non-headache properties of slurry explosives were appreciated by miners and the improved safety characteristics were also widely recognized.

One very significant result of these trials was that the long standing contract between the gold mining industry and AECI was rescinded in 1983 and that National Explosives began marketing Tovex explosives in competition with AECI.

At present slurry explosives are being sold by National Explosives (Tovex; not produced under licence), Dantex Explosives (Tovex; licenced by Du Pont, USA), AECI (Energex; licenced by Du Pont, USA). AECI discarded the Sinex range in 1985 when it also gained access to Du Pont technology. All of the above explosives are based on Du Pont technology which is widely recognized as the leading technology in slurry explosives. The main reason for this is the sensitizing agent (MMAN) which is versatile and allows for a wide range of explosive products with different properties and performance characteristics (density, energy content or strength, V.O.D.).

1.3 Emulsion Explosives

Another way of making ammonium nitrate based explosives resistant to water is offered by emulsion explosives. These consist of an emulsified mixture of ammonium nitrate solution, oil and wax. An intimate mixture is obtained giving a very large contact area between the fuel (oil and wax) and the oxidiser (ammonium nitrate) and this results in a very rapid and complete reaction. The sensitivity to initiation and propogation is controlled largely by creating micro-voids (air) in the mixture. Handling and safety characteristics of emulsions are generally similar to slurry explosives.

Emulsion explosives were introduced to gold mines in 1985 by AECI (Powergel) and in 1986 by Sasol (Emulite). The performance characteristics of these explosives are currently being studied by the Blasting Section of the Stoping Technology Laboratory. Initial test results have shown that emulsion explosives produce finer fragmentation (average size is 20% smaller) than NG based and slurry explosives. A qualitative picture emerging from industry is that in comparison to other explosives, emulsions produce finer fragmentation, cause less damage to hangingwall and footwall rock but are less capable of breaking out the burden rock near the toe of the hole.

At present emulsion explosives for gold mines are available only in cartridged form however efforts are underway by AECI and SASOL to offer a bulk emulsion that is pumped into holes as this may offer a lower unit cost explosive than cartridged products.

2 INITIATION SYSTEMS

The basic requirements of an initiation system for stoping are that all holes explode, and in a predetermined sequence. At present, fuse-igniter cord systems are used almost exclusively as this is the simplest, most inexpensive and easiest to use method available. Alternative initiation systems which incorporate millisecond-delay firing of holes have been and are still being developed and their performance evaluated. However the full benefits of these systems must be quantified to identify whether their higher cost as compared to fuse-igniter cord systems can be justified.

2.1 Fuse-Igniter Cord Initiation Systems

The following gives a brief description of important aspects of fuse-igniter cord systems. The ends of capped safety fuse protruding from blastholes are connected in order with a trunkline of igniter cord that, as it burns, sequentially ignites the fuses (see Figure 1). Provided the correct combination of igniter cord burning speed, fuse length and spacing of fuse connections along the igniter cord (interconnector spacing) is observed, holes are fired in sequence at 4-8 second intervals. Hence, there is no dynamic interaction between holes. The distance which separates an exploding hole and the igniter cord flame front is known as the burning front or lag distance (see Figure 2) which is generally between 3-5 m in length, depending on fuse length, interconnector spacing and the fuse and igniter cord burning speeds. As the burning front increases, the likelihood of blast induced igniter cord trunkline cut-offs decreases. These cut-offs are caused chiefly by rock loosened from the stope face and hangingwall by ground vibration.

A basic problem of fuse-igniter cord systems is that strict sequential firing and a long burning front represent opposing interests and a compromise is necessary. As the interconnector spacing is decreased, the probability of out-of-sequence shots increases (due to inherent burning speed variations of both fuse and igniter cord), but the burning front increases. Likewise, as the interconnector spacing is increased the probability of out-of-sequence shots decreases while the burning front also decreases. A graphic representation of this relationship is given in Figure 3.

2.1.1 Initial application of igniter cord for sequential firing(6)

Igniter cord was developed by Nobel's Explosive Company in the U.K. and was introduced in 1951 in South Africa for sequential initiation of blastholes in stoping and development. The introduction of igniter cord assisted efforts in developing longwall mining methods which began in 1945. Igniter cord facilitated sequential lighting of fuses without the need for hand lighting of fuses while travelling down the stope face. The use of igniter cord obviated the requirement of cutting fuses to various lengths for timing shots and it opened the way for delayed blasting. Consequently, blasting accidents and the exposure of personnel to fumes were greatly reduced. In addition, holes could now be drilled straight into the face instead of in benches and this provided for greater face advance and more concentrated mining with its associated advantages.

2.1.2 Improvements to fuse-igniter cord initiation systems

Following the application of igniter cord to sequentially initiate the burning of fuses during the 1950's it was found that inherent burning speed variations in both fuses and igniter cord were the cause of out-of-sequence firing of blastholes. Such non-sequential firing causes overburdening of subsequent blastholes and can lead to a host of problems including poor advance, increased hangingwall damage and irregularly shaped faces that hamper cleaning and require a lower producing blast to straighten the face. Non-sequential firing can also lead to miners compensating by over drilling and over charging, resulting in an inefficient operation. A long term effort was mounted during the 1960's to improve the quality and reliability of fuse-igniter cord systems. Extensive collaboration took place between COMRO, AECI and various mines.

Pigtails

Work on a pigtail initiation system was conducted from 1966 to 1968. This system was aimed at eliminating the timing errors introduced by the burning speed variability and incorrect application of the igniter cord (variable interconnector spacing between fuses). The system consisted of capped fuses with special connector clips that incorporated a chemical delay element and a short length of igniter cord. After loading, the igniter cords were coupled from one unit to the next in sequence and served only to ignite the delay units in sequence. Unexpectedly wide variation in the burning speed of fuses rendered this system ineffective.

Consecutively cut fuses

Safety fuse marketed in South Africa is designed to burn at 110 s/m ± 10 per cent (statutory limits = 99-121 s/m). In the manufacturing process fuse is made in 1 km reels and, while the burning speed of fuses cut from different reels may fall anywhere within the statutory limits of 99-121 s/m, the burning speed variation between fuses cut consecutively from a reel is small (<4 s/m). This feature gave rise to the concept of bundling fuses in consecutively cut packs of 25 fuses each. Fuses within these 'special pack' bundles had a burning speed variation of less than 4 s/m and offered greatly improved sequential firing when used properly. Special pack fuses were marketed from the early 1970's up to 1984 when they were replaced by colour coding of fuses.

Colour coded fuses

Although safety fuses were made available in bundles of 25 consecutively cut fuses of similar burning rate, a high probability of out-of-sequence shots still existed between bundles of different burning rates used on the same face, and between individual fuses when fuses from different bundles were mixed. To alleviate these operational problems a system of colour coding all fuse bundles according to burning rate was developed in 1984(7). Fuse reels are graded during manufacturing and bundles of 25 fuses are colour coded so that the burning speed range in any one colour band does not exceed 8 s/m. All fuses within the same bundle maintain a 4 s/m burning speed variation. This facilities the use of bundles with similar burning speeds on the same face so that a low probability (0,1 per cent) of out-of-sequence shots, both inter-bundle and intra-bundle, is virtually guaranteed provided the correct interconnector spacing is adhered to.

Stopesets

It is often difficult to ensure that the correct interconnector spacing is adhered to by miners. For this reason the Stopeset system was developed and made available in 1980. It is comprised of immoveable plastic adaptors fixed to igniter cord at regular intervals which mate with fuses fitted with special snap connectors (see Figure 4). The Stopeset system together with colour coded fuses ensures a low probability of out-of-sequence shots because it ensures the correct combination of fuse length igniter cord type and interconnector spacing. However, the Stopeset system is currently being used by only a few mines due to its higher cost over normal fuse igniter cord products. Large scale production trials have been conducted on various mines in an attempt to quantify the benefits of using Stopesets as compared with normal accessories. These trials indicated marginal benefits from the use of Stopesets in terms of efficiencies and face advance, however the variable nature of conditions encountered during mining obscures a direct and full comparison.

Wetspun safety fuse

During 1981 the manufacturing process for fuses was upgraded to produce a more consistent fuse burning speed. This new manufacturing method is referred to as the wetspun process as the black powder is in a slurry form when the fuse is made. The wetspun process in use today results in 90 per cent of all fuses produced falling within the burning speed range of 104-116 s/m (burning time variation < 12 s/m).

Stopecord

During 1978 a new igniter cord, Stopecord, with greatly improved water resistance was introduced. Stopecord is a twin igniter cord which consists of one strand of slow igniter cord that is encased with either medium or IC 57 igniter cord. Prior to stopecord, medium and IC 57 were the two most common types of igniter cord used in stoping. These igniter cords depend entirely upon their sheaths to resist the harmful effects of water which can cause failure of the igniter cord. Slow cord is very resistant to water and in stopecord it serves as a means of relighting the faster burning igniter cords to which it is attached in case the sheath has been damaged and exposed to water.

An additional feature is that stopecords are significantly stronger than normal igniter cords and are therefore less likely to be cut-off during the blast. This product has found wide acceptance on the mines.

Stope fuse - stope line

Stope fuse - stope line, an experimental fuse-igniter cord system which holds the promise of improved timing accuracy and reliability, is currently undergoing initial field trials. This system is based on a twinned slow igniter cord which has both excellent water resistance properties and the most consistent burning rate of all igniter cords. A slow burning fuse to match the stope line is made using the wetspun fuse manufacturing process. The overall result is a system that is more reliable, produces better sequential firing and has a greater burning front than systems currently used. Initial findings from field tests have been most promising and it is likely that this system will replace normal fuse-igniter cord systems in the future.

Underground blasting gallery

To assist in the design and testing of fuse-igniter cord systems an underground blasting gallery in the form of a simulated stope face was operated from 1978-1985. The gallery was instrumented to measure the burning times of each loop of igniter cord and of each fuse as well as the resulting firing times of individual shots and the burning front(8). No explosives were used in these tests. An important feature of the gallery was that it offered similar environmental conditions (temperature, humidity and atmospheric pressure) to those found in stopes as these can affect significantly the performance of fuse-igniter cord systems. The gallery was equipped to test narrow-reef, wide-reef and development end initiation systems.

A computer model of sequential firing in stopes was developed and used in conjunction with the blasting gallery to design sequential firing systems. More than 750 000 safety fuses and 250 km of igniter cord were tested at the gallery and it played a primary role in most of the above mentioned developments in fuse-igniter cord systems.

By 1985 the quality of fuse-igniter cord systems had reached the stage where less than 2,5 failures (out-of-sequence shots plus misfires) per 1000 shots would result when systems were used properly. Furthermore the stope fuse-stope line system holds the potential of reducing this failure rate. It was therefore decided in 1985 to discontinue work on improving fuse-igniter cord systems because it appeared that further improvements would be marginal at best.

2.2 Millisecond-Delay Initiation Systems

In addition to the inherent problems and limitations of fuse-igniter cord systems discussed in previous sections, a further limitation is that this system does not make use of dynamic interaction between holes. With fuse and igniter cord, the burden rock of each hole has been broken and the fragments have come to rest long before the next hole fires. However, when holes are fired at sufficiently short intervals, the time span of breakage and heave of neighbouring holes will overlap. Thus the breakage process of adjacent holes can be made to interact in a constructive fashion.

Work has been and is being conducted on millisecond-delay initiation systems with the objectives of improved sequential firing, improved reliability and the utilization of dynamic interaction between holes to improve throw and fragmentation of rock. By increasing the amount of blasted rock thrown towards the gully the time necessary to clean the stope can be reduced, allowing shorter cycle times and greater mining rates. A further reduction in cleaning times may also accrue from a reduction in the amount of large rocks that are difficult to handle and which require secondary breakage.

2.2.1 Electronic and electric initiation systems

During the mid to late 1960's an electrical and an electronic method of sequential firing were investigated[9]. In the electrical method a sequential electric firing device located remotely from the blast was used to initiate ten banks of ten electric delay detonators. Successful sequential firing by this method depended upon the delay accuracy of the detonators and upon ensuring that the detonator wires remained unbroken until the detonators were energized. This system was unsuccessful due to a lack in the availability of precision electric delay detonators and to cut-offs of lead wires before detonators were energized.

In the electronic method, individual timers were used with instantaneous detonators in each blasthole to overcome the problems associated with the electrical switching method described above. This system functioned successfully however it was found to be too costly and complicated for routine use. Although a commercial system was not produced from these efforts, primarily because of limits in electrical/electronic technology, these early investigations laid the ground work for current efforts.

During 1985 emphasis shifted away from efforts to improve the quality of fuse-igniter cord systems in favour of the development of alternative initiation systems. As mentioned earlier it is believed that the quality of fuse-igniter cord systems has reached a plateau, with only minor improvements forthcoming, and therefore efforts would be better utilized in the development of alternative initiation systems.

The most promising type of new initiation system at present is based on electronics. Collaboration with a local manufacturer has reached the stage where this system is performing well in surface tests and a mine worthy unit is now being developed. Underground trials are expected to commence during the first quarter of 1988.

The architecture of this system is such that it meets fully the requirements of an initiation system for stoping. Identical timing units are placed in each blasthole and these are hooked to a trunkline in the desired sequence of firing. Before energizing the system, a timing signal is passed from the first hole to the last. Only after all in-hole timing units have been energized does the first hole fire and therefore no blast induced cut-offs of the initiation system will occur. The timing period between shots is uniform, it can be easily changed in manufacturing and it is accurate to within one millisecond of the specified delay time. It is expected that this system will be easier to use and perform more reliably than fuse-igniter cord systems. The largest unknown at this stage is the cost of a final unit but at present this is projected to be approximately R2 per unit.

2.2.2 The Nonel system

Currently the only commercially available millisecond-delay initiation system that is suited for stoping is the Nonel system. Nonel consists of a small diameter plastic tube which is internally coated with a thin film of reactive material. When initiated, either by a detonator or detonating cord, a low energy shock wave is transmitted along the inside of the tube at about 2 km/s to a detonator crimped onto one end.

The commonly used layout of the Nonel system in stoping is shown in Figure 5. A double detonator arrangement (Unidet) is used where the delay of the in-hole detonator is about 10 times that of the surface detonator and this provides a detonation front (equivalent to a dynamic burning front). Holes are harnessed in pairs to enhance reliability as the initiation process will continue onto subsequent holes even if one unit fails. Nonel does not fully meet the requirements of an initiation system for stoping as it is subjected to the inaccuracies of pyrotechnic delay units in detonators and also to blast induced cut-offs of the initiation system.

Since its introduction as an experimental product for stoping during the early 1980's a number of trials have been conducted by mines and the results have been monitored by COMRO. Until recently a meaningful evaluation of this system has been hampered by quality control problems in experimental units. A production version of the Nonel system is currently being used by one mine (Harmony) at a rate of 30 000-40 000 units per month. In addition efforts are underway to characterize the performance of, and the benefits from, this system at the COMRO test site at Doornfontein. Results of this work are discussed in a later section (4.3).

3 ROCK BREAKAGE METHODS

There are many ways in which to apply explosives to rockbreaking in stopes. This section describes work that has been done on different rock breakage methods. Most of these involved a change in one or more of the following:
Burden rock geometry, hole dimensions and charge geometry.
These factors have a direct bearing not only on the breakage process itself but also on other stoping functions such as cleaning and support. For example, the blasting method determines the amount of rock produced and the way in which it is displaced into the stope. The blasting layout dictates the shape and extent of muck pile to be cleaned, the way in which support is loaded by the blast, whether or not a blast barricade is needed, the importance of sequential firing and so forth.

3.1 Long Hole Blasting

Experiments in long hole blasting were conducted on a number of mines during the late 1950's and early 1960's(6). A stope layout for this method is given in Figure 6. Some of the advantages envisaged were that the face would require no daily examination, barring down and support; there would be no socket to clean; only a few well drilled holes would be drilled instead of hundreds of short holes that were frequently badly marked, inaccurately drilled and often not properly charged. Consequently, better supervision and more rapid face advance was expected.

Holes of 50 mm diameter were drilled parallel to the stope face by pneumatic drills from advance strike gullies which were in effect sub-levels. Holes were charged with ammon gelignite cartridges and the holes were traced with detonating cord to ensure propagation of the explosive reaction along the entire charge length. Rock was blasted onto a rolling scatter pile.

A number of reasons have been cited for the failure of this method and the two most important of these seem to be: firstly, problems in controlling drill hole deviations with long holes, and this was made worse in fractured ground; and secondly, holes became unchargeable in deep fractured stopes due to relative movement of rocks along fracture surfaces over the hole length.

3.2 Wide Reef Guts Hole

Work was done during the mid 1970's on reducing the number of holes drilled in wide reef stopes, the main objectives being to reduce the time spent on drilling and to improve the performance of fuse-igniter cord initiation systems in wide reef stopes.

Standard drilling patterns normally used in wide reef stopes are comprised of small diameter holes (30-40 mm) drilled about 400-600 mm apart both horizontally and vertically. It was found in underground tests that the number of holes could be reduced by up to 40 per cent by employing a row of large diameter (guts) holes at mid stoping width. These holes were capable of breaking about half of the rock being mined with the remainder then being broken out by small diameter holes near the hangingwall and footwall (see Figure 7).

Diameters of up to 64 mm were required for the guts holes and this was found to be beyond the productive ability of the pneumatic hand-held rock drills in use. Thus a clamping device was designed for linking the bores of small diameter holes drilled in parallel (see Figures 8 and 9). An attractive feature of fewer holes being drilled was that the design and implementation of the fuse-igniter cord initiation system was made simpler.

The main problems associated with the non-acceptance of this method by the mines were that:

(i) the drilling of the guts hole was an arduous task that required considerable skill on the part of the operator.

(ii) Close supervision was required as both accurate hole patterns and a high degree of sequential firing were critical to the achievement of acceptable breakage results.

3.3 Short Hole Blasting

During the early 1970's an extensive programme of work was conducted to investigate short hole (750 mm) blasting methods(10,11). It was believed that there would be advantages in using short holes in preference to longer holes (1,1-1,5 m) whenever the difficulties arising from the use of long holes resulted in a face advance of less than 6 m/month. The following benefits were expected with the use of short holes:

(i) smaller charges could be used with less damage being caused to the hangingwall and to stope supports.

(ii) strict sequential firing would not be necessary since rock is essentially 'cratered' from the stope face and therefore each hole breaks virtually independently of the others.

(iii) the amount of rock broken would be more compatible with the capacities of scraper cleaning systems and might make possible the drilling and blasting of the entire face each day.

(iv) A smaller advance per blast together with more frequent blasting was expected to assist strata control because after the blast a shorter span would be left unsupported for a shorter period.

Computer simulation of mining operations together with underground experiments were used to evaluate short hole blasting methods. As a result of these simulation studies it became clear that significant improvements in stope production could be achieved by the use of shorter holes, but that these were dependent on five related improvements, namely, increases in cleaning and drilling capacities, reductions in the times taken to prepare the face for cleaning and to charge up after drilling, and overall reduction of the time required for completion of a cycle.

In order to effect these improvements the panel layout was altered to that shown in Figure 10 . In addition light weight pneumatic rock drills (20 kg) and unitized explosive cartridges (1 of 25 x 600 mm cartridge per hole) were used.

It appears that the principal reasons for this mining method not being widely adopted is that more cycles were required for the same face advance and cycle completion on a daily basis was hampered due to difficulty in reducing the cleaning times to acceptable levels.

3.4 Unitized Cartridges

The concept of unitized cartridges arose out of the work on short hole blasting discussed above. It was believed that the loading of a single cartridge in each hole rather than numerous shorter cartridges would reduce the charging-up time(11). In addition, the unitized charge concept was seen as a potential method of controlling the amount of explosive used per hole. These cartridges were used extensively during the short hole blasting evaluations.

The factors which lead to the failure of this approach to gain acceptance were the higher unit cost of explosives in this form due to packaging and the ease by which miners could segment unitized cartridges to circumvent the control of the amount of explosive used per hole.

3.5 Guar Gum as a Coupling Agent

During the mid 1970's field studies were conducted to determine the benefits of using guar gum as a coupling agent in blastholes(12).

Because of the severe gauge wear of drill bits caused by the abrasiveness of quartzites in gold mines, diameters of drilled holes may vary from about 28 mm to 42 mm. This results in the presence of an air gap between the stick explosives inserted in the drilled hole and the sides of the hole. This air gap is filled to some degree by consolidation of cartridges in holes by the charging stick. Such consolidation was however more difficult with utilized cartridges because of their length. The air gap decouples the explosive from the rock and decreases the breaking effectiveness of the explosive.

The aim of this study was to investigate the effect that a coupling agent would have in minimising or overcoming decoupling of the stick explosive, without increasing the bulk of the explosive charge. The coupling agent consisted of a guar gum formulation (3 per cent powder, 97 per cent water) which was mixed by hand with water on site.

A suite of blasts with 600 mm long Gelignite cartridges (unitized cartridges) of various diameters was conducted under controlled conditions to minimise the number of variables between blasts. The tests indicated a minimal apparent benefit which did not warrant the extra supervision and man-hours required for use of the coupling agent.

3.6 Graded Charges

Graded charges are a common method of blasting in surface mining where a greater concentration of explosive energy is used in the bottom portion of holes where breaking is most difficult. Out of a combination of this concept with the unitized charge concept came the idea of a graded, unitized charge for stoping. It consisted of a unitized cartridge with a cardboard sleeve around one half to give an explosive diameter that was larger at the lower half of the charge. A slurry explosive (relatively soft) was used in order to facilitate charge consolidation at the hole bottom.

Although a meritorious concept, the significantly higher cost, and poor performance of the newly developed slurry explosive were the main reasons for the failure of this charge.

3.7 Shaped Charges

Shaped explosive charges hold the potential of providing an explosive system for rock breakage without drilling.

The drilling process is time consuming and is becoming increasingly expensive due to rising labour costs. One potential method of reducing the time and cost to complete a mining cycle is by using shaped charges to simultaneously penetrate and break out the rock. In this application a number of conical shaped charges would be set up in the appropriate pattern with a small stand off distance from the face (see Figure 11). The conical shaped charges would produce an initially thin jet which penetrates the rock while the intense radial pressures produced during penetration breaks out the rock (See Figure 12).

A simple cost calculation based on penetrations reported in the literature has shown that a shaped charge technique could be comparable in cost to drilling and blasting methods(13). Discussions with local specialists in the armaments industry have confirmed that such a technique could be feasible. Exploratory tests with shaped charges in stopes will begin in the first half of 1988.

3.8 Blast Assisted Resue Mining

It has been proposed(14) that the blast be used to cast the waste portion of the stoping width directly into place as backfill while the ore is fragmented but left largely in place. A schematic illustration of this blast assisted resue mining process for various mining conditions is given in Figure 13.

Potential advantages of this system are reduced ERR levels, improved effective grades, reduced labour requirements and increased stoping widths for an improved working environment and mechanization potential. Potential disadvantages are gold losses to the backfill, increased support damage, the need for accurate drilling of holes in more complex pattern and the need to accurately control the charging and initiation timing of holes. In order to achieve the required rock throw and initiation timing, a more reliable and easy to use millisecond-delay initiation system is needed. A further drawback is that the ability of keeping the waste and ore portions separate in fractured stopes is questionable.

However this method does have potential in relatively unfractured rock where parting planes exist between the ore and waste rock. It is intended to investigate this technique in the future following the development of an accurate and reliable electronic initiation system.

4 ROCK BREAKAGE PROCESS

The results of blasting are one of the major factors that determine the cost, productivity and safety of the stoping operation. A full understanding of explosive/rock interactions is necessary for blasting to be conducted in a controlled manner that is compatible with other stoping functions such as stoping width control, strata control and rock cleaning.

However, the explosive rock breaking process has not yet been well quantified for any type of blasting and this is especially so for underground mining.This is largely because blasting does not lend itself easily to investigation due to its violent and dynamic nature, and also because of the large number (and wide range) of important variables which affect blast results. Even for the small charges used in reef mining the power released by one hole is on the order of 10 000 MW, comparable to the South African electricity demand of about 17 000 MW.

Meaningful progress has been made towards improving the understanding and control of blasting by combining results from field investigations with general developments in blasting science. The following summarizes work done on understanding better the rock breakage process in narrow stopes.

4.1 Assessing the Rockbreaking Capabilities of ANBA

As part of the programme to apply ANBA explosives to gold mines during the 1950's and 1960's one objective was to identify the performance characteristics and economic benefits of using ANBA.

In order to quantify the breakage ability of ANBA a good deal of work was done on the mines to measure the dynamic strain produced in rock from ANBA and NG explosives(2). As a direct check on the performance of the explosive, the velocity of detonation (VOD) was also measured. This technique served mainly to determine the effectiveness of loading techniques and initiating devices for ANBA as well the effect of water on this explosive.

Measurements showed that, in the same diameter hole, ANBA explosives produced greater magnitude strain pulses in rock due to a better filling of the hole because of the bulk form. These tests served their basic objective in that the performance of ANBA was established. However, a direct correlation between the character of strain wave and the resulting rock breakage was not in evidence. It was concluded that further work was necessary before a full understanding of blasting would be realised.

4.2 Field Trials of Down Graded Anfex and Other Explosives

This work was conducted during the early 1970's after Anfex had been observed to cause excessive (fine) fragmentation of rock, increased stope widths and damage to the hangingwall and footwall(15). In an endeavour to reduce this excessive breakage, LS Anfex, a down-graded form of Anfex containing 80 per cent limestone chips, was evaluated.

Field trials were conducted to compare Anfex, LS Anfex, a fertilizer grade Anfo (denseprill) Dynagel and Ammon Gelignite. Their performance was determined on the basis of their efficiencies, degree of fragmentation which they caused and the amounts of gold lost as a result of fines produced.

Unfortunately, a true comparison of these explosives was partially obscured by changing stress conditions in the rock due to hearby stoping by the mine. It was found that drilling efficiencies for the cartridged explosives were similar at about 3,4 m/ca, but those for Anfex and LS Anfex were significantly less at 4,2 m/ca and 3,8 m/ca, respectively. The lowest consumption of explosive was of Gelignite (1,6 kg/ca) followed by Dynagel at 1,7 kg/ca. The consumption of granular explosives was much greater, being 2,5 kg/ca for Anfo, 2,6 kg/ca for Anfex and 3,4 kg/ca for LS Anfex. The fragmentation of rock was not greatly different between the explosives but was in line with the explosive consumptions. On the basis of these experiments it was suggested that cartridged explosives offered superior performance (as defined earlier) to granular bulk explosives.

4.3 Millisecond-Delay Blasting

The potential benefits of using millisecond-delay initiation systems which provide dynamic interaction between neighbouring holes was discussed in section 2.2. A pilot study to investigate benefits of using millisecond-delay initiation systems in stoping was conducted during the early 1980's(16). The aspects studied were the effects of various delay times on fragmentation, throw and ground vibration. A short (12 m) up-dip panel in a near-surface site (150 m deep) was used for the experiments. A special initiation system was used (not suitable for production use), comprised of a programmable blasting machine and short period electric detonators.

It was found that interaction between adjacent holes occurred below inter-shot delays of 150 ms resulting in enhanced throw of rock into the gully (see Figure 14). Fragmentation results were inconclusive due to only grab samples (1-2 per cent of total rock broken) being examined. The intensity of ground vibration, and its deleterious effects on strata control, was not enhanced by short delays over the range 10-200 milliseconds.

The above tests identified that there are potential benefits from the use of millisecond-delay firing of shots in stoping. This work evoked interest in industry and a number of mines began using the Nonel initiation system. However conflicting reports as to the merits of millisecond-delay firing made it clear that additional work was needed to investigate the effects of other important variables, namely:

(i) longer stopes

(ii) the fractured rock conditions in deep stopes

(iii) explosive type

(iv) hole patterns and timing combinations.

During mid 1986 a programme to investigate more fully the above variables and quantify the operational benefits of millisecond-delay firing was commenced at the Doornfontein test site (≈2 400 m deep). Initial tests conducted at this site centered chiefly on throw aspects (see Figures 15 and 16). Extensive experimentation will be conducted over the next two years to cover the range of important variables listed above.

4.4 Characterization of Stope Blasting

During the early 1980's when interest was aroused in millisecond-delay firing of stope blasting it was identified that a better understanding of the fundamental processes involved in conventional stope blasting was required, both for the design of fuse-igniter cord and millisecond-delay blasts. A programme was therefore embarked upon in 1983 (still in progress) to characterize the stope blasting process comprehensively. The overall objective is to develop blast design guidelines as an aid in optimizing the stoping operation. The approach is to establish empirical relationships between the important variables encountered in stoping (e.g. explosive type, burden geometry, rock conditions) and the important results of blasting (e.g. fragmentation and throw of rock, air blast, ground vibration), while at the same time to define the mechanisms of explosive rock breakage in order to enhance the general usefulness of the empirical relationships for blast design.

Initial work was conducted at a near-surface test site (185 m deep) in order to develop instrumentation for measuring the dynamic events associated with blasting while at the same time characterize the blasting process in rock under low stress(17). During 1986 a deep experimental stope (2 400 m) was commissioned at Doornfontein in order to characterize the blasting process at depth. A wide variety of instrumentation and measurement techniques has been developed(17,18,19). These include systems for high speed photography of both single and multiple hole blasts, for measuring ground vibrations, airblast and dynamic rock strain close to the blasthole, and for measuring the gas pressure at the hole collar. A brief summary of findings of these studies follows.

4.4.1 Rock fragmentation

The size distribution of rock fragments from stope blasting is important for numerous reasons. Large rocks present difficulties in rock cleaning and subsequent handling and may require secondary breakage in order to pass through the ore pass grizzley. Damage to stope support and blast barricades is increased when large fragments are produced because support reaction forces are much higher when impacted by one large fragment as compared to an equal mass of noncoherent particles.

Intense fragmentation is also undesirable as fines can become trapped in cracks and crevaces in the footwall resulting in gold losses. Furthermore, smaller sized rocks are not amenable to hand sorting and fragmentation that is too fine is known to disable the often used antogenous milling process and this necessitates the addition of steel balls and thus higher milling costs.

A programme of experimentation was undertaken to quantify the size distribution of fragments produced by stope blasting in order to relate stope blasting to important operational variables such as the type, quantify and distribution of explosive and to rock conditions that existed prior to blasting. A summary of findings from the experiments is illustrated by Figures 17-19.

The dependence of average fragment size, d_{50}, on specific explosive energy (MJ explosive/m^3 rock) for various explosives at a near surface (185 m deep) test site is illustrated by Figure 17. With the exception of Powergel, the average fragment size produced by all the explosives tested were comparable on a specific energy basis. Powergel produced an average fragment size that was 25% finer than that of other explosives and this effect is attributed to the comparitively higher velocity of detonation and therefore greater power of this explosive.

Figure 18 shows the effect of depth of mining on the fragmentation measured at different test sites. Note that for the same specific energy, the average size of fragments decreases with an increase in depth of the site. It is believed that this effect is due to the contribution of mining induced stress fractures to fragmentation.

The fragment size at 90,50 and 10 per cent passing versus specific energy is given in Figure 19. Note that range of fragment sizes decreases, toward smaller sizes, as specific energy decreases. Above a specific energy of about 8 MJ/m^3 (approximately 2 kg/m^3) there is little reduction in the amount of large fragments. Blasting with a specific energy value in excess of about 8 MJ/m^3 will not have much effect on reducing the numbers of large rocks but will, however, cause a greater amount of undersirable fines.

4.4.2 Airblast

The character of airblast produced by a typical stoping blasthole is given in Figure 20. Measurements at distances of 2 m from blastholes have shown that air pressure waves exhibit high peak pressure (350 kPa) but low impulse (pressure x time) due to their short duration. The application of basic principles of airblast damage used by the military shows that airblast damage to stoping equipment is highly unlikely. This supports the view that impact by rock fragments is the major cause of damage to stoping equipment.

4.4.3 Ground Vibration

Figure 21 gives relationships that describe the magnitude of ground vibration from stope blasting as a function of distance and explosive mass per hole. These curves were developed from actual measurements of blast induced vibrations at the near surface test site. Also shown in the figure are various damage thresholds reported in the literature which indicate the expected response of the rock to various vibration levels. This information is directly useful in designing the burning front distance required by fuse and igniter cord systems in order to avoid cut-offs of the igniter cord trunkline.

4.4.4 Breaking capacity of charges

The breaking capacity of various explosives as a function of hole diameter and the degree to which the explosive diameter matches the hole diameter is shown by Figure 22. This figure shows clearly that a wide range of charge breaking capacity exists and this highlights the need for guidelines to facilitate proper blast design.

4.4.5 Blast loading of stoping equipment

This section describes briefly the types and character of blast induced loads on stoping equipment. This information was derived from vibration, airblast, and high speed photography studies in experimental stopes. It is shown that impact by flying rock is responsible for blast damage to stoping equipment and important factors that determine the magnitude of rock loads are discussed.

During blasting, explosive energy is partitioned into three areas that are available for interaction with stoping equipment. These areas are airblast, hangingwall and footwall vibration and flying rock. There is no simultaneous action of these forces on stoping equipment because of their differing propagation speeds. The respective speeds of airblast and stress waves in rock are one and two orders of magnitude greater than rock fragment velocities. Figure 23 shows the approximate time span over which blast induced loads are applied to near-face stope support.

As discussed in sections 4.3 and 4.6 of reference 17 and section 4.4.2 of this report, airblast and ground vibration will not have a significant affect on stope support and barricades. The air blast pressure pulse is of short duration and therefore the impulse is too small to cause damage. Also, the magnitude of rock displacement near the charge (<1 m) in both the hangingwall and footwall due to vibration (stress waves) is only a few hundred microns. It is emphasized that support and barricade damage is due to impact by flying rock.

Important factors that will determine the force of rock loading are the amount of rock broken per hole, the size distribution of fragments, the velocity and direction of movement of fragments and the separation of fragments within the flow of moving rock. In general, rock loads will decrease with:

(i) decreasing rock mass broken per hole,
(ii) lower fragment velocities,
(iii) a direction of impact that is at an increasingly smaller angle with the surface of the equipment being struck,
(iv) a finer size distribution of fragments, and
(v) a greater spatial separation of in-flight rock fragments

The first three of these points are obvious and the last two points are illustrated in Figure 24 which shows qualitatively the different character of rock loads by a large rock and by an equivalent mass of incoherent particles. The Figure depicts an elongate support element (hydraulic prop or timber pole) but the same principles also apply to other types of equipment such as barricades. The single fragment will generate a greater load than a stream of smaller incoherent particles because separate fragments will not act on the impacted object at the same time. Also, as the separation of rock fragments is increased, the rock loads will decrease since the kinetic energy density of the stream of fragments has decreased.

4.4.6 Cross-hole pressurization

Another important finding of stope blasting studies was the observation by means of high speed photography that explosive gases can travel along fractures to invade adjacent holes before they fire (See Figure 25). This can lead to breakage problems caused by the explosive charge in adjacent holes being desensitized to initiation, ejected from the hole before it fires, or being detonated sympathetically.

This phenomenon is expected to be most common in deep stopes where rock is highly fractured. Further work under these conditions is necessary to understand the frequency and implications of cross-hole pressurization.

4.4.7 Blasthole liner technique

Another significant achievement has been the development of a research technique, based on a metal blasthole liner, of separating the two basic energy sources available for rock breakage, i.e. shock wave and subsequent gas expansion. The blasthole liner prevents the penetration of explosive gas products into the rock mass surrounding the charge. These gases are exhausted through the blasthole collar after the initial blasthole expansion following detonation. With this technique the shock wave stage of breakage is virtually identical to that of a normal hole while the breaking action due to subsequent gas expansion is eliminated. The layout used for blasthole liner tests is given in Figure 26. The blasthole liner technique has assisted greatly in understanding the mechanisms of explosive rock breakage in stopes. For instance, from the use of this technique an understanding of the difference in breakage affects of emulsion explosives as compared to Dynagel and Anfex (ie. differences in fragmentation, rock damage and advance) has been gained.

The findings of the characterization studies completed so far have widespread implications for stoping operations as discussed above. A preliminary design methodology has been developed which includes the major variables encountered in stoping(17,21). A final version of this blast design method is scheduled to be completed during 1988.

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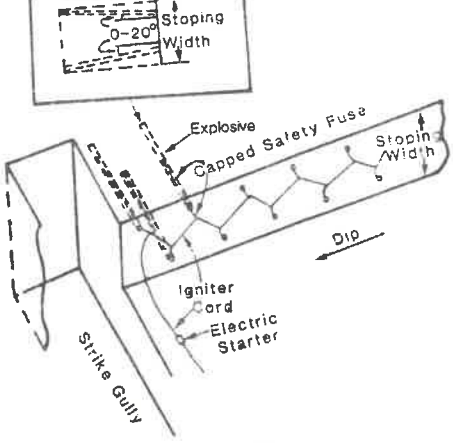


Figure 1 A REPRESENTATIVE DRILLING PATTERN, AND BLASTING ARRANGEMENT IN NARROW STOPES

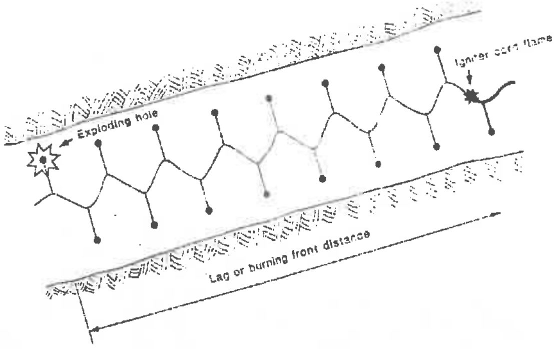


Figure 2 ILLUSTRATION OF THE IGNITER CORD BURNING FRONT DISTANCE OR LAG

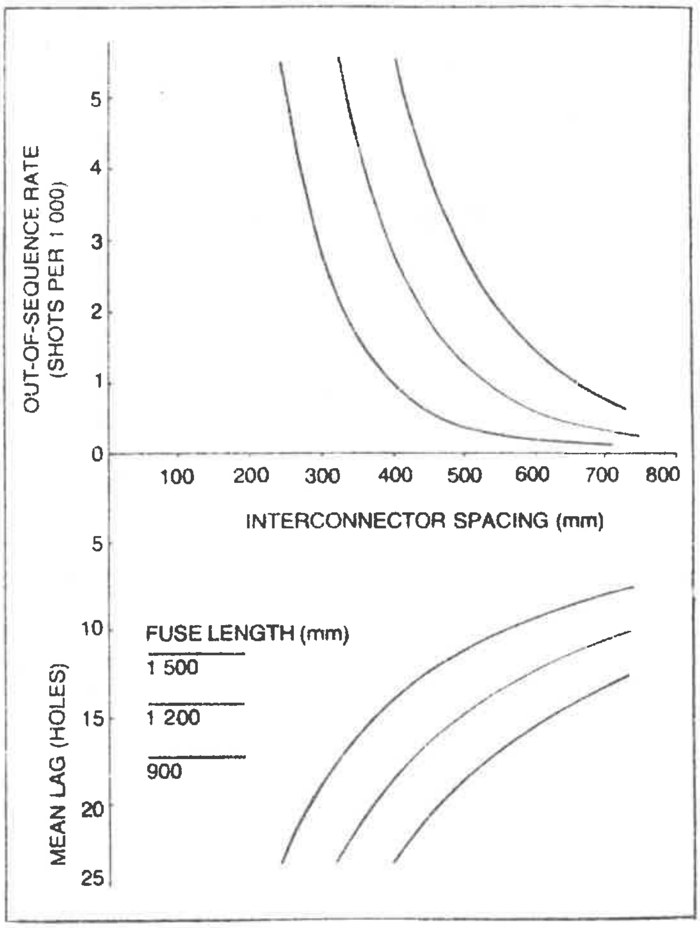


Figure 3 APART FROM FAILURES OF SYSTEM COMPONENTS, OUT-OF-SEQUENCE SHOTS AND TRUNK LINE CUT-OFFS DUE TO AN EXCESSIVELY SHORT LAG DISTANCE ARE THE MAJOR CAUSES OF BLASTING INEFFICIENCIES. THE DIAGRAM SHWS THAT THE OUT-OF-SEQUENCE RATE CAN BE REDUCED SIGNIFICANTLY BY INCREASING THE DISTANCE BETWEEN CONNECTORS. ON THE OTHER HAND, THE DISTANCE OF THE IGNITER-CORD FLAME FRONT FROM THE EXPLODING SHOT (LAG DISTANCE) DECREASES WITH INTERCONNECTOR SPACING. TO FIND AN OPTIMUM MATCH OF THE TWO OPPOSING FACTORS IS ONE OF THE MAJOR PROBLEMS IN DESIGNING SEQUENTIAL FIRING SYSTEMS FOR GOLD MINE STOPES

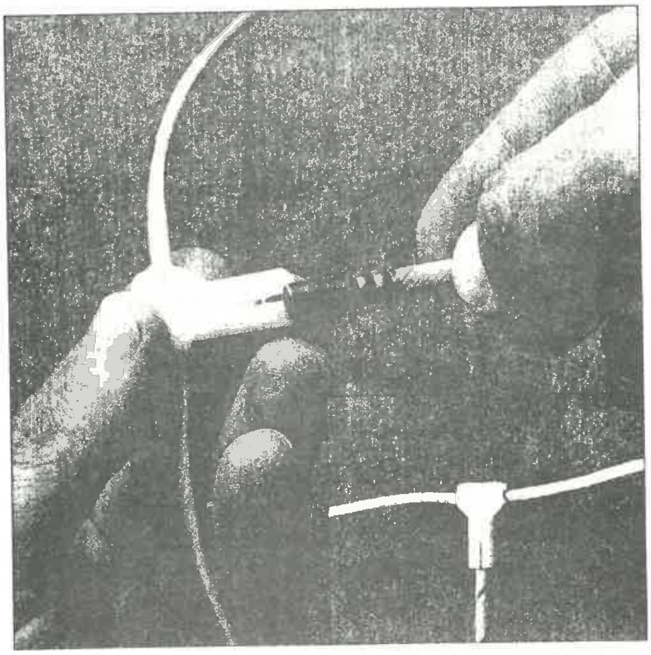


Figure 4 ILLUSTRATIUON OF THE STOPESET INITIATION SYSTEM

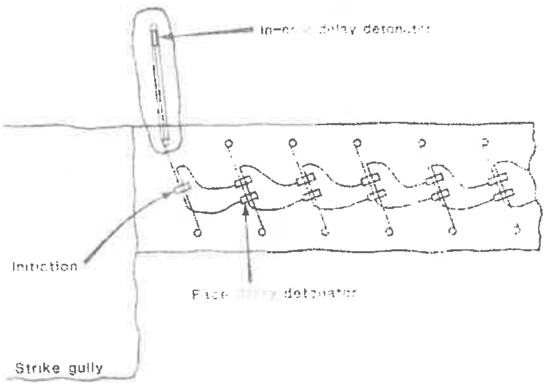


Figure 5 INITIATION SYSTEM LAYOUT FOR NONEL BLASTS USING THE UNIDET SYSTEM

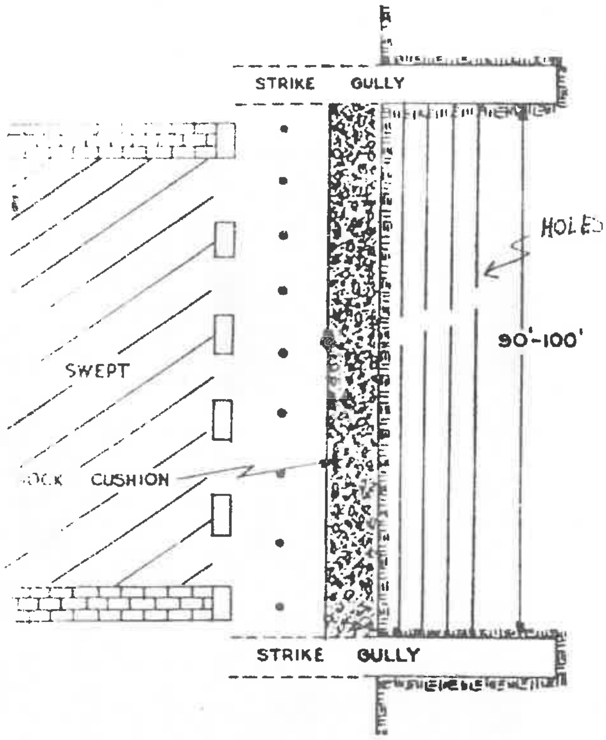


Figure 6 STOPE LAYOUT FOR LONG HOLE BLASTING EXPERIMENTS

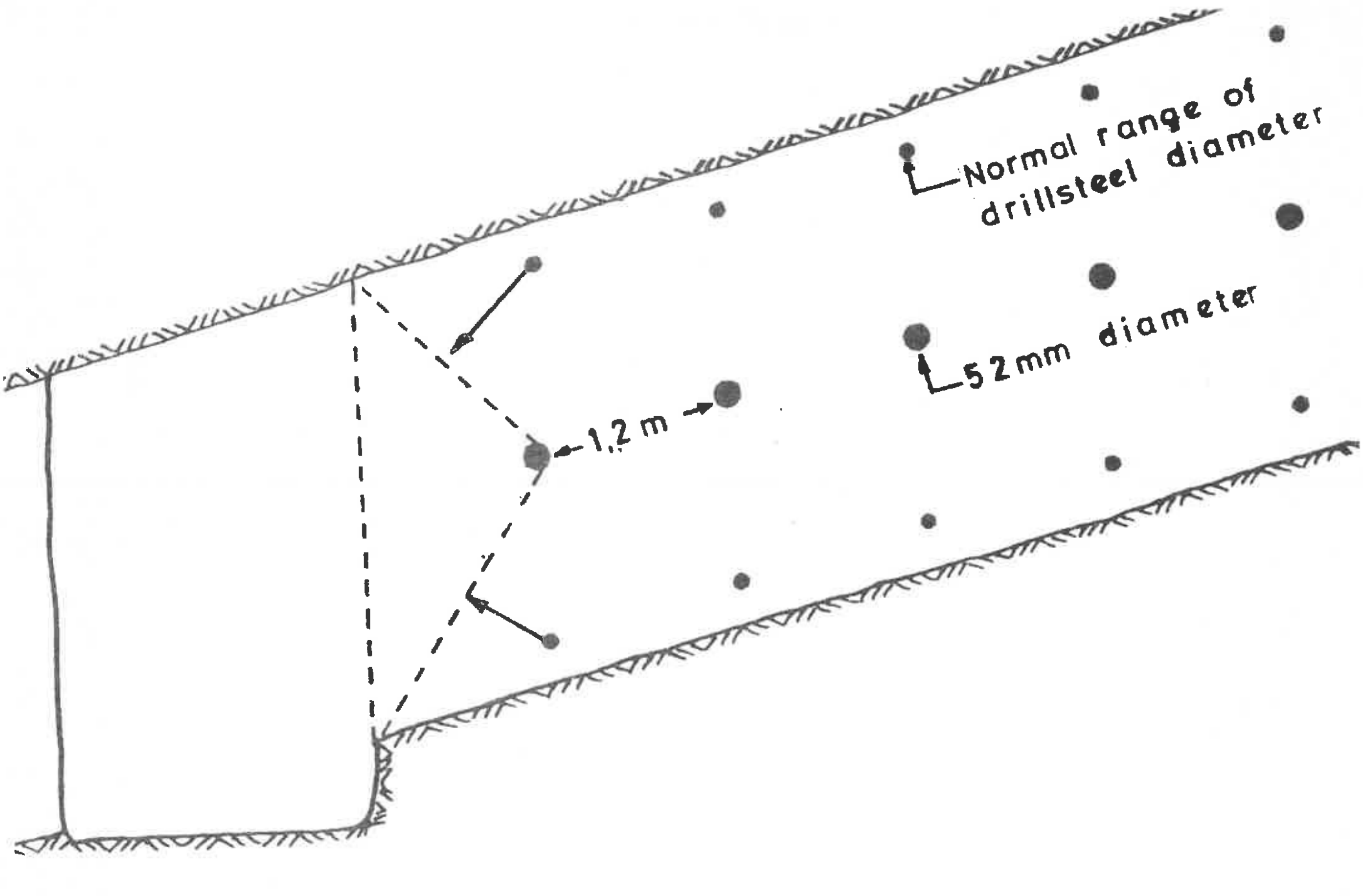


Figure 7 HOLE PATTERN FOR WIDE REEFS BASED ON A LARGE DIAMETER (GUTS) HOLE IN THE MIDDLE ROW

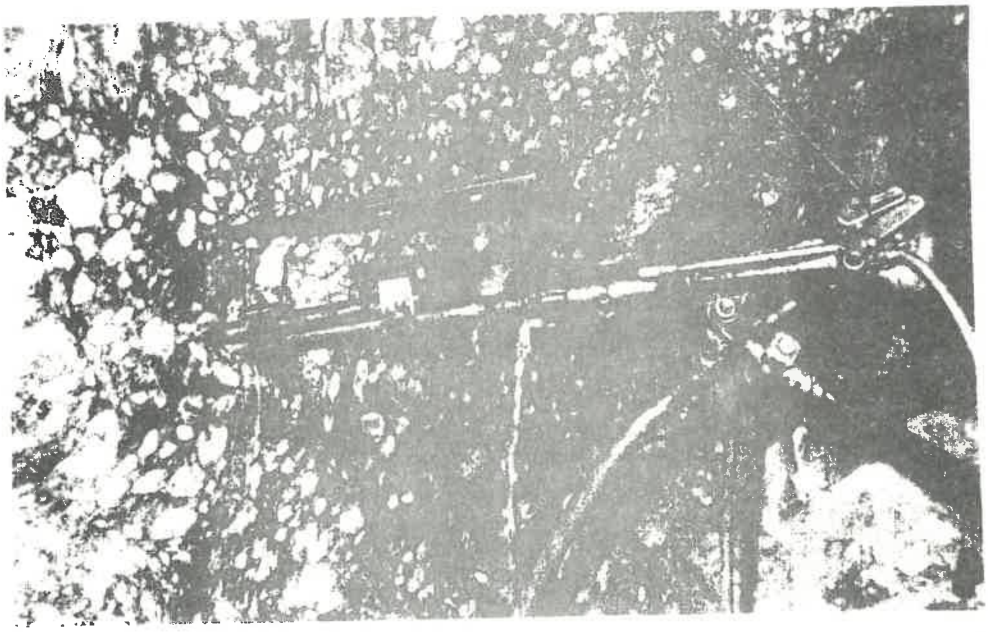


Figure 8 THE CLAMPING DEVICE FOR INTERLINKING THE BORES OF TWO HOLES DRILLED IN PARALLEL

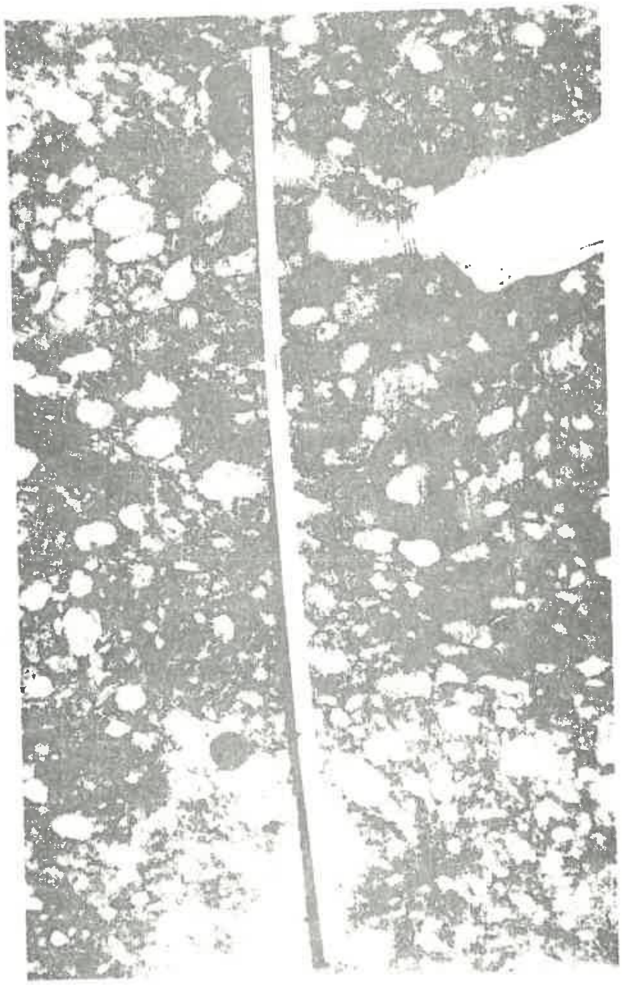


Figure 9 THE EFFECTIVELY LARGER HOLE RESULTING FROM DRILLING TWO PARALLEL HOLES WITH THE CLAMPING DEVICE

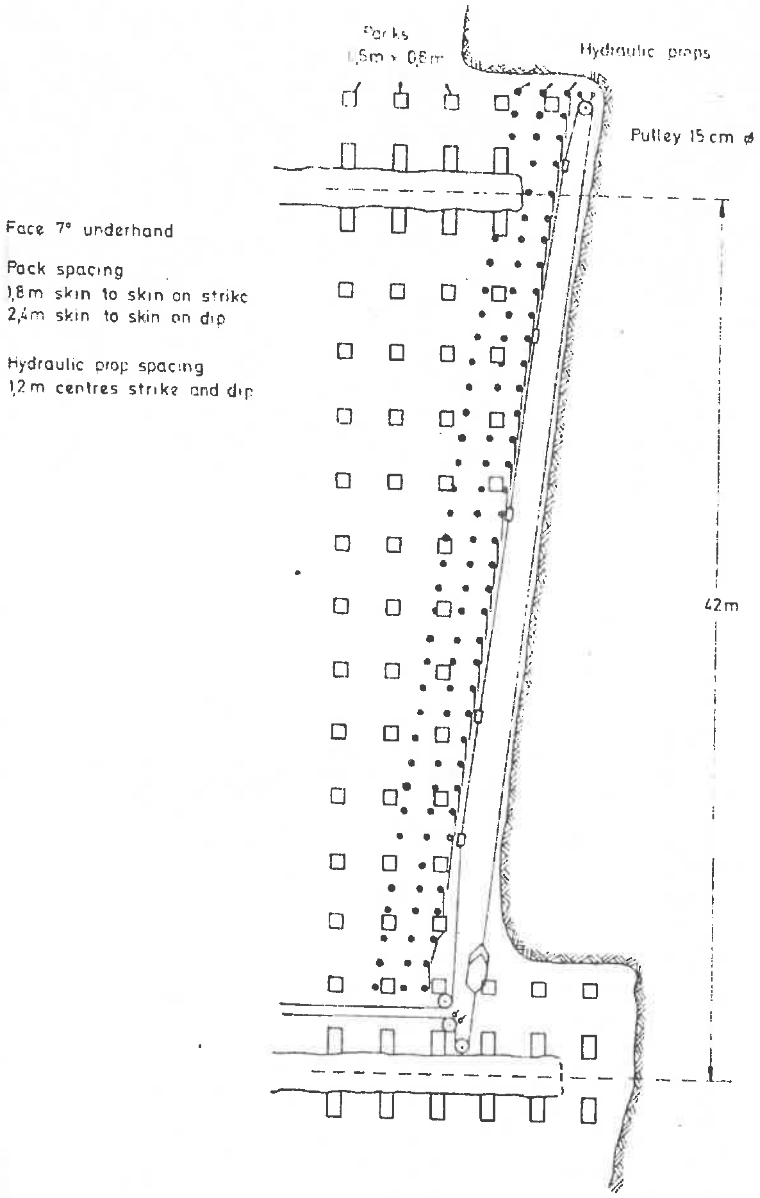


Figure 10 A PLAN SHOWING A TYPICAL PANEL LAYOUT FOR SHORT HOLE BLASTING EXPERIMENTS, INCLUDING THE POSITIONING OF BOTH SCRAPER ROPES ON THE FACE SIDE OF THE BARRICADE AND THE RETURN SNATCH-BLOCK CHAIN ANCHORS

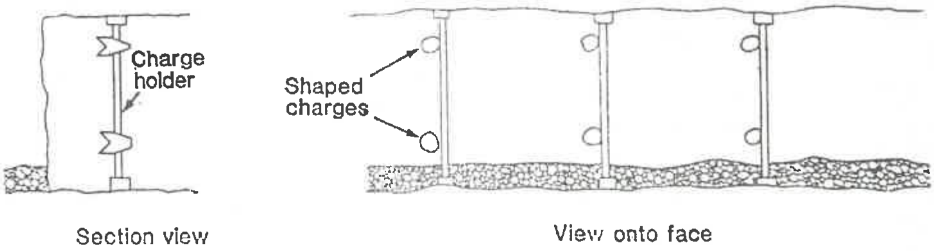


Figure 11 AN ENVISAGED LAYOUT USING CONICAL SHAPED CHARGES FOR ROCKBREAKING WITHOUT DRILLING

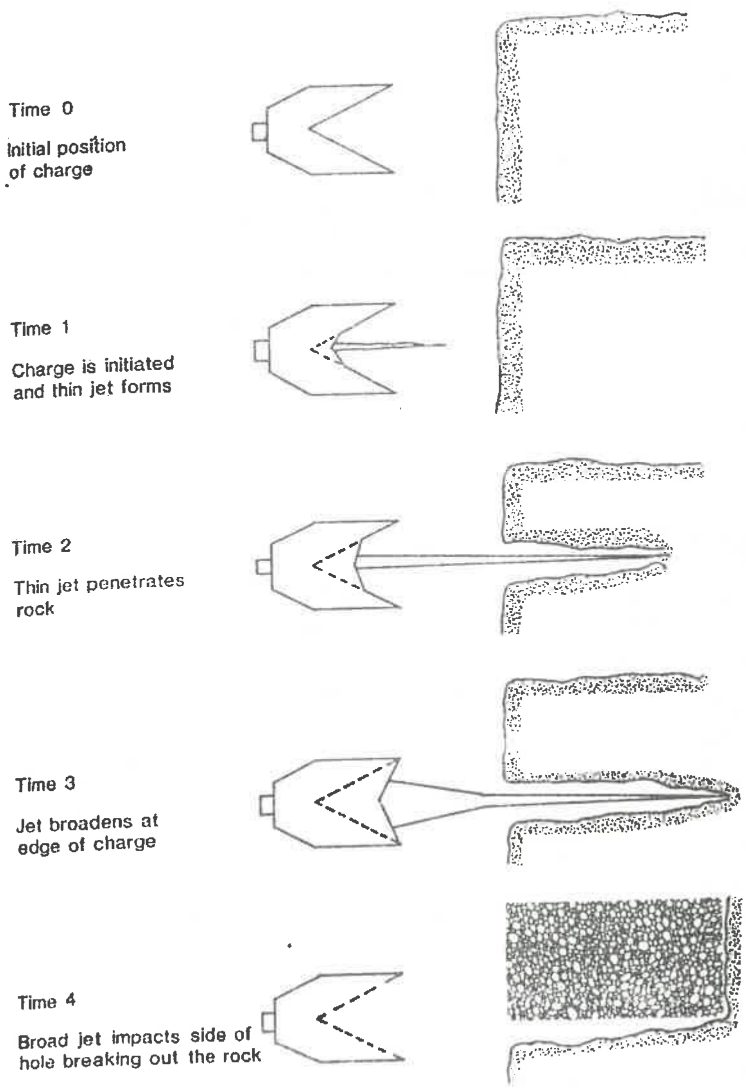
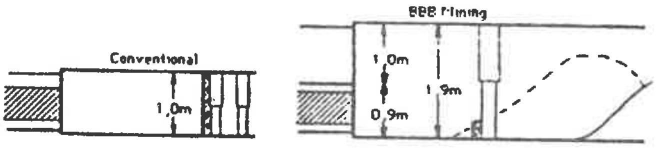
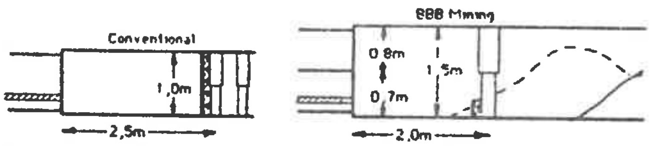


Figure 12 SEQUENCE OF EVENTS ENVISAGED DURING THE PENETRATION AND BREAKAGE OF ROCK BY CONICAL SHAPED CHARGES

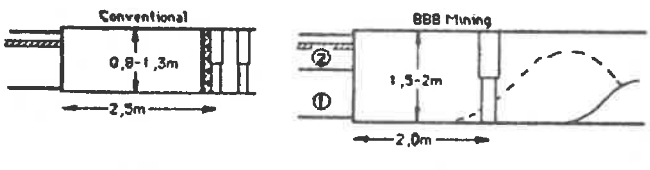
SCHEMATIC OF CUT PLACEMENTS REQUIRED
IN BBB MINING.



MEDIUM (0,6m) CHANNEL ENVIRONMENT



NARROW CHANNEL ENVIRONMENT



DIFFICULT HANGINGWALL CONDITIONS

Figure 13 <u>ILLUSTRATION OF THE BLAST ASSISTED RESUE MINING CONCEPT FOR VARIOUS MINING CONDITIONS</u>

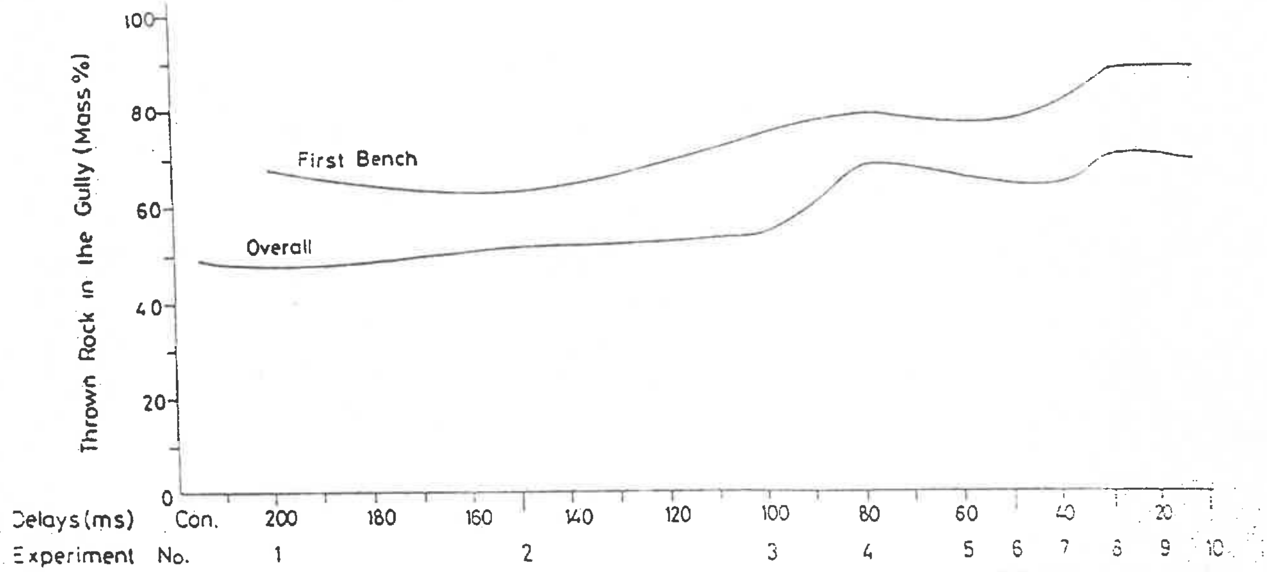


Figure 14 ILLUSTRATION OF THE EFFECT OF INTERSHOT DELAY TIME ON THROW OF ROCK INTO THE GULLY DURING SHORT PANEL UP-DIP MINING (CON. = FUSE-IGNITER CORD INITIATION)

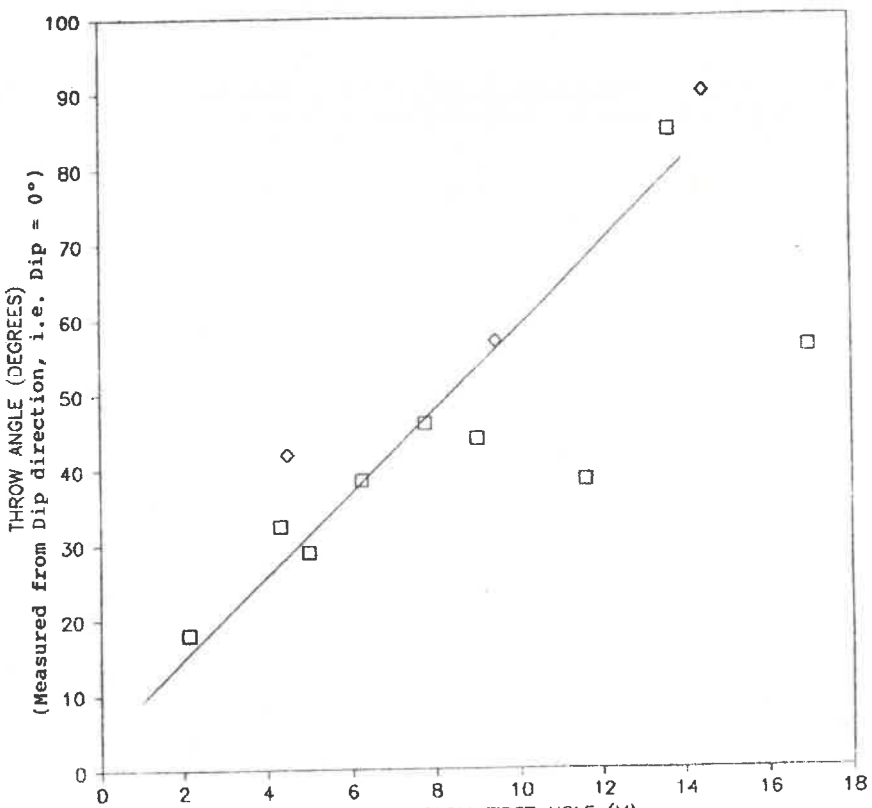


Figure 15 THROW ANGLE OF ROCK VERSUS UP-DIP DISTANCE FROM FIRST HOLE FROM TESTS AT DOORNFONTEIN. TEST CONDITIONS: DEPTH = 2400 m, STOPING WIDTH = 1.0 m, PANEL LENGTH = 22 m IN BREAST MINING CONFIGURATION, HOLE LENGTH = 1.0-1.2 m, NONEL DELAY = 50 ms WITH HOLES CONNECTED IN PAIRS, STAGGERED DRILLING PATTERN. NOTE THAT THROW ANGLE (MEASURES FROM DIP) BECOMES GREATER AS THE UP-DIP DISTANCE FROM THE GULLY INCREASES.

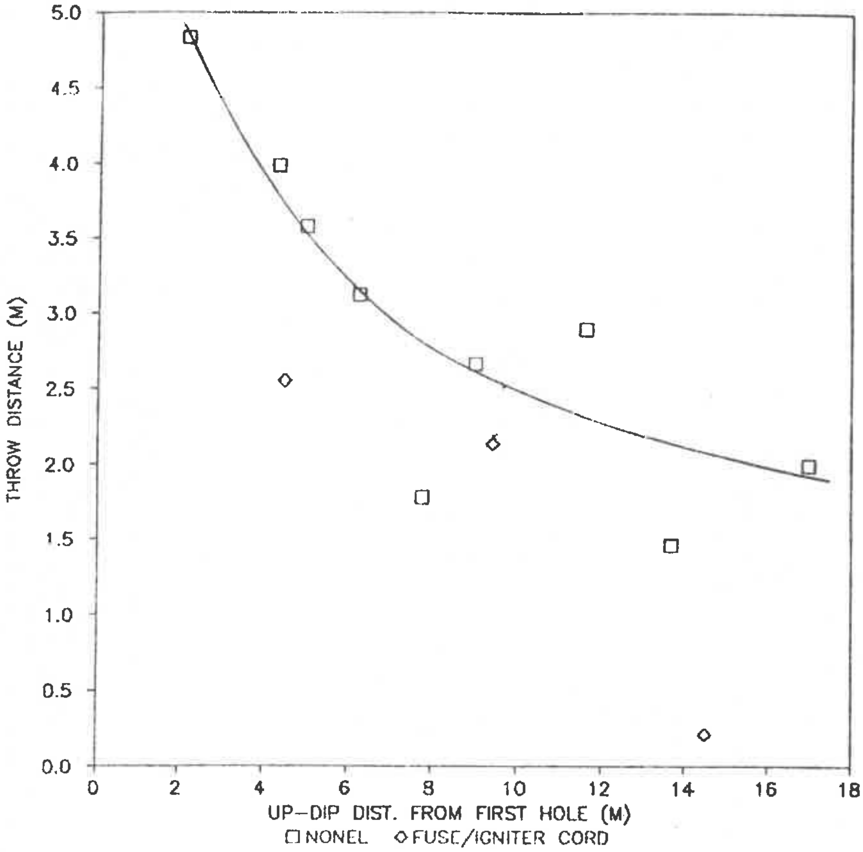


Figure 16 <u>**ILLUSTRATION OF THE DECREASE IN ROCK THROW DISTANCE AS THE PANEL LENGTH INCREASES. (SAME CONDITIONS AS STATED IN FIGURE 15)**</u>

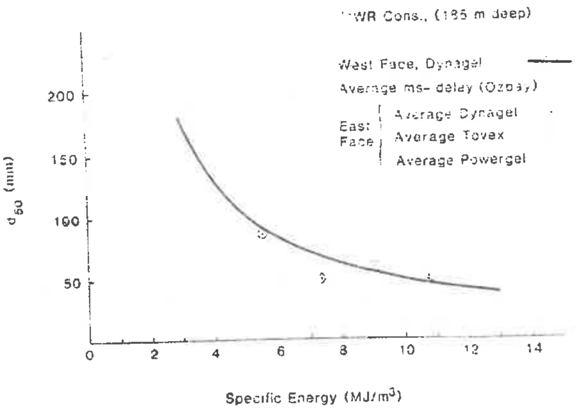


Figure 17 THE DEPENDENCE OF AVERAGE FRAGMENT SIZE, d_{50}, ON SPECIFIC EXPLOSIVE ENERGY (MJ EXPLOSIVE/m³ ROCK) AND EXPLOSIVE TYPE

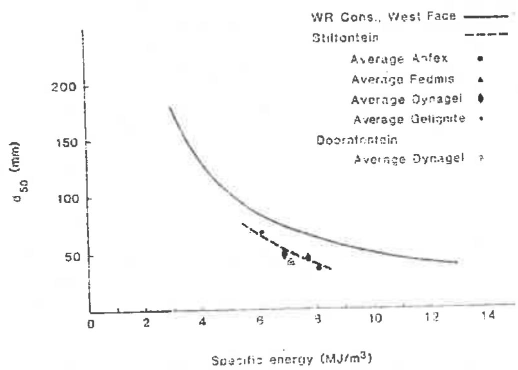


Figure 18 AVERAGE FRAGMENT SIZE AS A FUNCTION OF SPECIFIC ENERGY FOR TEST SITES AT VARIOUS DEPTHS (WRC 185 M, STILFONTEIN 1850 M, DOORNFONTEIN 2400 M)

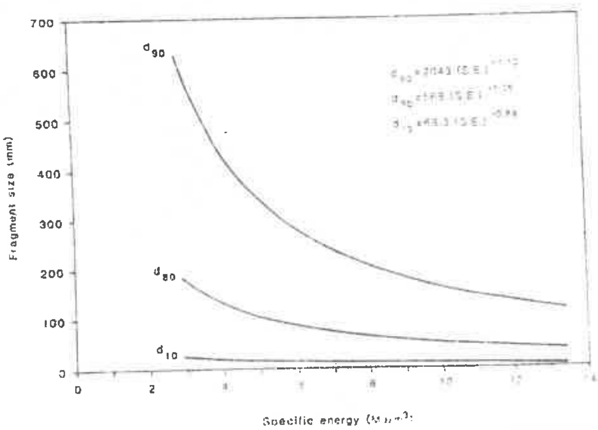


Figure 19 FRAGMENT SIZE AS A FUNCTION OF SPECIFIC ENERGY FOR 10 PER CENT, 50 PER CENT, AND 90 PER CENT PASSING SIZES. NOTE THAT THE RANGE OF FRAGMENT SIZES DECREASES, TOWARD SMALLER SIZES, AS SPECIFIC ENERGY INCREASES

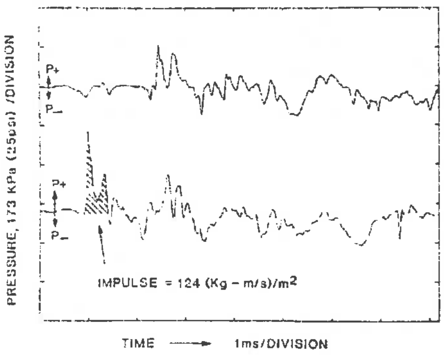


Figure 20 <u>AIRBLAST HISTORIES MEASURED AT 2 m FROM A SINGLE HOLE. PEAK PRESSURES ARE LARGE BUT IMPULSE IS SMALL AND THEREFORE BLAST BARRICADES WILL NOT BE DAMAGED</u>

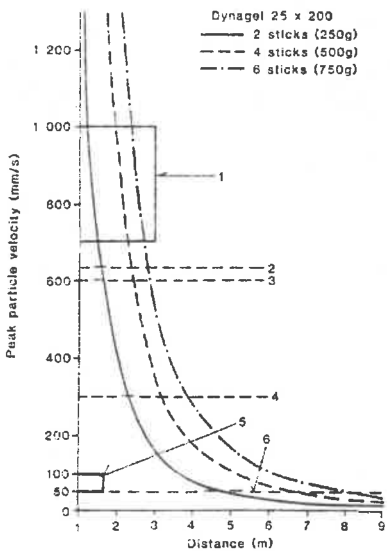


Figure 21 <u>MEASURED GROUND VIBRATION INTENSITY, IN TERMS OF PEAK PARTICLE VELOCITY, AS A FUNCTION OF EXPLOSIVE MASS AND DISTANCE FROM THE CHARGE. NUMBERS 1-6 INDICATE DAMAGE THRESHOLDS RANGING FROM DAMAGE TO HARD INTACT ROCK (NO. 1) TO CONSERVATIVE LIMIT FOR AVOIDING DAMAGE (NO. 6). THIS INFORMATION IS DIRECTLY USEFUL IN DESIGNING THE BURNING FRONT DISTANCE REQUIRED BY INITIATION SYSTEMS IN ORDER TO AVOID CUT-OFFS</u>

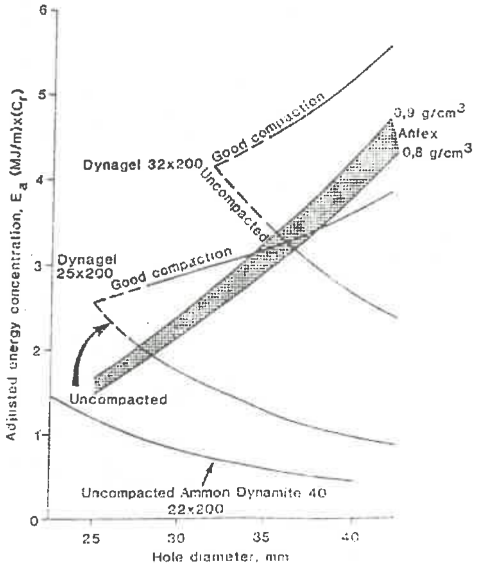


Figure 22 **ILLUSTRATION OF THE BREAKING CAPACITY (E_a) OF VARIOUS EXPLOSIVES AS A FUNCTION OF HOLE DIAMETER AND THE DEGREE TO WHICH THE EXPLOSIVE DIAMETER MATCHES THE HOLE DIAMETER. IT IS CLEAR THAT A WIDE RANGE OF BREAKING CAPACITY EXISTS, HIGHLIGHTING THE NEED FOR GUIDELINES TO FACILITATE PROPER BLAST DESIGN**

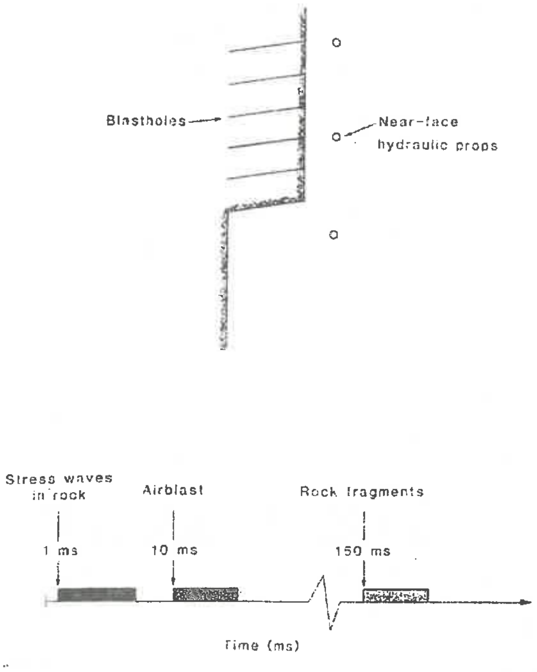


Figure 23 **APPROXIMATE TIME SCALE OF BLAST-INDUCED LOADING OF NEAR-FACE STOPE SUPPORT. THIS INFORMATION WAS DERIVED FROM VIBRATION, AIRBLAST AND HIGH SPEED PHOTOGRAPHY STUDIES, AND IT ILLUSTRATES THE APPLICATION OF FINDINGS FROM COMPREHENSIVE STOPE BLASTING INVESTIGATIONS TO THE DESIGN OF STOPING EQUIPMENT**

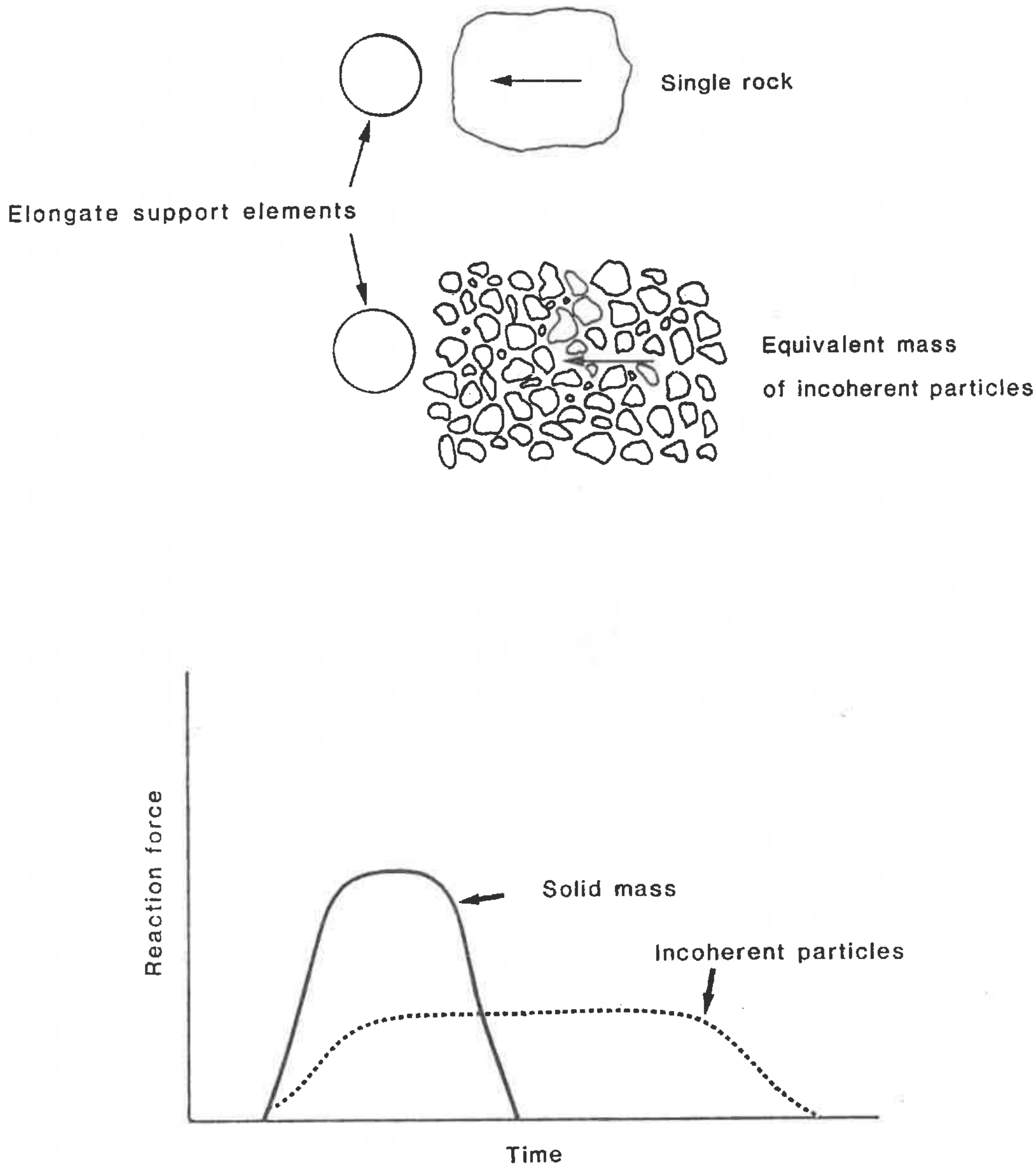


Figure 24 AN ILLUSTRATION OF ROCK LOADS FOR A SINGLE ROCK AND FOR AN EQUIVALENT MASS OF SMALLER FRAGMENTS

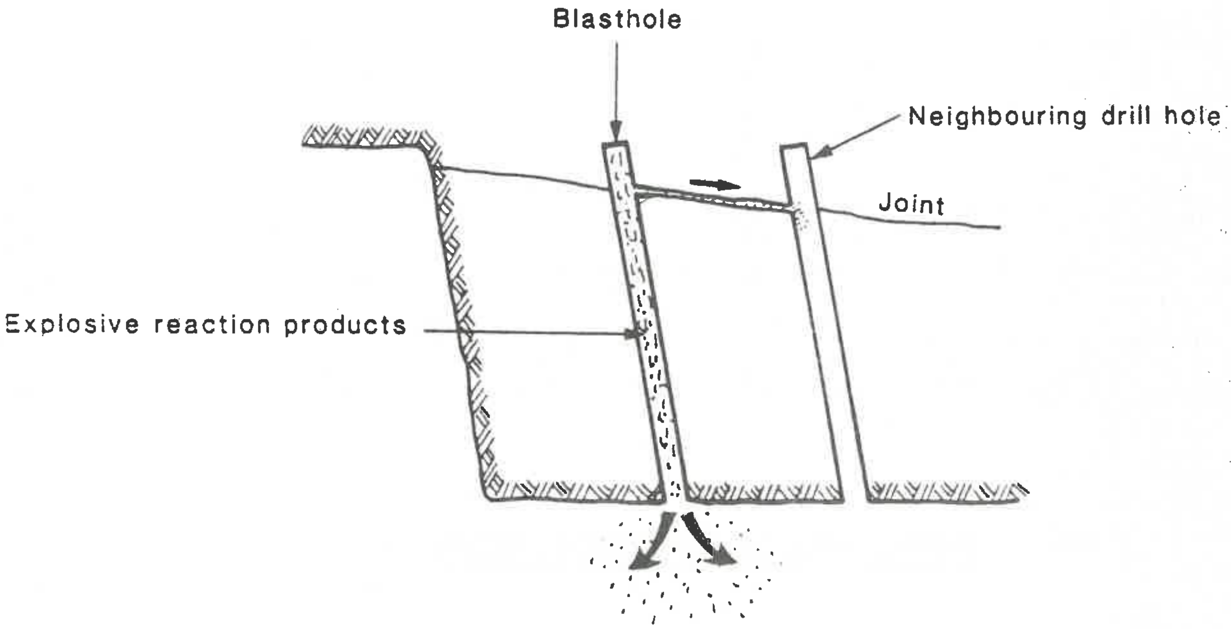


Figure 25 BY MEANS OF HIGH SPEED PHOTOGRAPHY IT WAS OBSERVED THAT EXPLOSIVE GASES CAN TRAVEL ALONG GEOLOGICAL AND STRESS FRACTURES TO INVADE ADJACENT BLASTHOLES BEFORE THEY FIRE. THIS CAN LEAD TO BREAKAGE PROBLEMS CAUSED BY THE EXPLOSIVE CHARGE IN ADJACENT HOLES BEING DESENSITIZED TO INITIATION, EJECTED FROM THE HOLE BEFORE IT FIRES, OR BEING DETONATED SYMPATHETICALLY

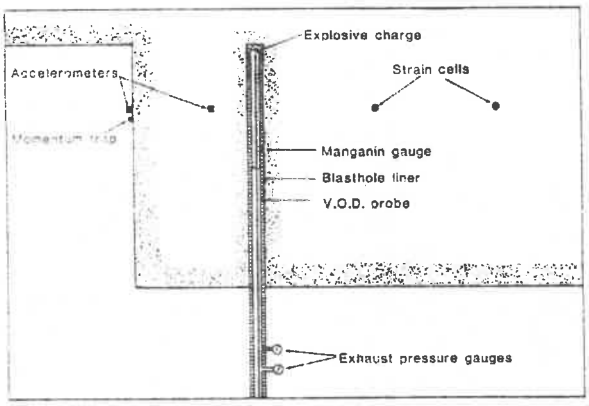


Figure 26 <u>**EXPERIMENTAL LAYOUT USED DURING BLASTHOLE LINER TESTS FOR QUANTIFYING THE MECHANISMS OF EXPLOSIVE ROCK BREAKAGE**</u>

CHAPTER 2

HAND-OPERATED ROCKDRILLING

P F HIND
Stoping Technology Laboratory

HAND-OPERATED ROCKDRILLING - SCHEME

- ROCK-DRILLING
  - Fundamentals
    - Rock Condition
    - Percussion/Rotation Requirements
    - Drill bits
    - Long-hole Drilling
    - Wide Reef Drilling
    - Water Assisted Drilling
  - Hand-Operated
    - Manual Thrust
    - With Thrustleg
    - Pneumatic Limitations
    - Electric Limitations
    - Hydraulic
      - Emulsion
        - Performance
        - Costs
        - Production Trials
        - Application in Mines
      - Water
        - Potential Advantages

1 INTRODUCTION

In the early days of gold mining in South Africa, blast holes were produced by striking a short drillsteel with a hand held hammer, generally 1,8 kg (4 lb) mass, and rotating the steel between successive blows. Water was introduced into the holes to remove chippings and allay dust, this was self cleaning from holes drilled above the horizontal but had to be removed by a cloth rag on the end of a stick for holes drilled below the horizontal,(1). With manual drilling it was only possible for a man to drill one 0,9 m hole in a shift, making the operation very labour intensive.

In 1892 the first pneumatic rockdrills were introduced into the gold mines on the Witwatersrand,(2), these were large machines weighing 90 to 135 kg that had to be rig mounted, with a screw feed for thrusting. With these drills it was possible for a two-man crew to drill 7,3 m (24 ft) per 8 h shift, the low performance resulting from the poor mobility of the drill/feed assembly and laborious method of changing drillsteels.

Over a period of years these pneumatic rockdrills were developed and resulted in a 'light', 32 kg, hand held drill that was manually thrust by the two-man crew. Torque was produced in this design of drill by mechanically linking the rotation and percussion mechanisms via the rifle-bar and the output was adequate for drilling in the competent rock encountered. By 1935 these machines were in use throughout the South African gold mining industry for drilling blast holes, and a two-man crew could drill 30,5 m (100 ft) in an 8 h shift, see Table 1 for comparisons.

Table 1 COMPARISON OF PERFORMANCE FOR EARLY ROCKDRILLING TECHNIQUES(2)

Method	Fathoms Broken Per Machine Shift	Tons Broken Per 'Drilling Boy' at 4ft (1,2 m) Stoping Width
Hand Drilling	-	0,5
'Old' Reciprocating Machines	0,5 (2 natives)	3,0
'Modern', (1946), Jackhammers	3,4 (2 natives)	18-24

In the mid 1950s there was a general acceptance by the Industry that significantly improved drilling performance could be achieved if the thrust on the pneumatic drill was improved. A 'simple' pneumatic thrustleg was designed and introduced which resulted in a drilling rate of 10 m/h being achieved, however, it was 10 years before this improvement was fully implemented.

As the depth of the mines increased the rock conditions changed, the compressed air systems became inefficient and as a result the drilling performance of the thrustleg mounted pneumatic rockdrills deteriorated. In order to improve the drilling operation the Research Organization of the Chamber of Mines instigated a programme of work to determine the factors affecting drilling and thus the design specifications for an effective rockdrill and thrustleg assembly, and subsequently develop an improved rockdrilling system.

2 FUNDAMENTALS

The ability of a rockdrill to drill a hole can be affected by:

(i) rock condition
(ii) rockdrill performance characteristics
(iii) drillsteel bit configuration
(iv) thrusting mechanism characteristics
(v) hole diameter and length

2.1 Rock Condition

The rock encountered at the face of the stopes in the South African deep levels mines is generally highly fractured, with spacing from 10 to 300 mm for distances up to 7 m in advance of the face[3]. The fractures in the rock result from the high stresses occurring around the discontinuity of the stope[4] and affect the drilling of blast holes in the stope face at all stages:

(i) collaring the hole; pieces of the face spall off, slabs come away and the hole makings are lost causing holes to be drilled inaccurately, the spalling and slabbing also make collaring the hole difficult and physically tiring for the operator.

(ii) drilling the hole; pieces of rock around the already drilled hole spall off and bind on the drillsteel, increasing the torque required from the drill.

(iii) removing the drillsteel; the pieces of rock that spall off in the hole jam between the rear of the bit and the hole side and make drillsteel removal difficult and tiring.

(iv) moving equipment between holes; the poor hangingwall conditions require support close to the face, making it more difficult for the operator to move the equipment between holes and increasing the physical effort required.

(v) drilling from rigs; the close face support and high density of support required in the stope make the use of drill rigs extremely difficult.

From the mid 1960's the ±400 000 m of stope blast holes drilled daily in South African gold mines have been produced exclusively by pneumatic rockdrills mounted on simple thrustlegs. The inherent problems resulting from the use of pneumatic rockdrilling systems, discussed later, led to the Research Organization instigating a research programme to determine the requirements for effectively drilling in fractured rock.

2.2 Percussion and Rotation Requirements for Drilling in Fractured Rock

(i) Requirements of a percussion mechanism.

A theoretical study was undertaken by Research Organization personnel in 1969 to derive a relationship between the penetration rate of a drill, its principal mechanical parameters and the rock conditions[5]. Assuming ideal thrusting conditions, the relationship shown below was derived and later verified experimentally.

$$PR \propto \frac{E_i \times \bar{f} \times T_R}{A_H \times E_V} \qquad ---- \qquad (1)$$

where PR = penetration rate

E_i = piston blow energy

$\bar{f}$ = piston blow frequency

T_R = coefficient of energy transfer from the drillsteel to the rock

A_H = area of hole

E_V = specific energy value for the bit/rock combination

For a given rock condition, hole size and coefficient of transmission, equation (1) can be simplified to:

$$PR \propto E_i \times \bar{f} \qquad ---- \qquad (2)$$

$\propto$ percussive output power

<u>NOTE</u>: The specific energy of rock for any rock breaking technique is the energy consumption per unit volume of rock broken. The value of specific energy varies with the uniaxial compressive strength of the rock, the type of rock and its condition, the particle size produced and method of rockbreaking employed(6). Work carried out by the Research Organization in the 1960s indicated a straight line relationship between specific energy and particle size when plotted on a log-log scale,(4,7). Values of specific energy vary from ±4000 kWh/m^3 for fragments of ±0,1 mm produced by diamond drilling to ±3 kWh/m^3 for fragments of ±400 mm produced by an impact driven wedge, these results were measured in quartzite with a compressive strength of ±200 MPa. The uniaxial compressive strength of quartzite encountered in the South African gold mines varies from 142 MPa to 466 MPa,(4).

(ii) Requirements of a rotation mechanism

In percussive drilling a line load is generally applied to the rock surface via a chisel bit. The stresses produced by this force create a small volume of crushed rock in the immediate vacinity of the tool and rock contact, and more extensive shear and tensile fractures in a much larger volume of surrounding rock,(6).

This mode of rockbreaking requires from the rotation mechanism:

(a) an adequate torque output to move the bit from the indentation produce, while the drillsteel remains thrust against the rock, plus the additional torque to overcome the affects of any spalled rock jamming the drillsteel, as already discussed.

(b) the drillsteel must rotate at a sufficient rate to ensure that the bit is in contact with unbroken rock for each successive blow; if this does not occur the bit may slip back into the previous indentation and impact on a cushion of crushed rock producing little, if any, penetration.

(iii) Additional rockdrill requirements

A consequence of the mode of rockbreaking noted above is the requirement for the crushed rock to be effectively removed from the hole, failure to do so adversely affects affects the penetration rate. The crushed rock and any large fragments produced are removed from the hole by flushing water fed through the drillsteel. It is therefore necessary to incorporate in the drill design a means of supplying sufficient flushing water to the drillsteel without affecting the drill performance.

2.3 Drill Bits

(i) Chisel bit wear

In 1969 a theoretical investigation into the wear of tungsten carbide chisel bits used for drilling quartzite was carried out at the Chamber,(5). A relationship was derived, equation 3, and was found to be in close agreement with experimental work carried out Te Water and Mikulka(8) at West Rand Consolidated gold mine from which the relationship shown in equation 4 was derived:

$$\text{Bit gauge} = d_h - K'(L)^{1/3} \qquad ---- \quad (3)$$

$$\text{Bit gauge} = d_h - K_1(L)^{1/3} + K_2 \qquad ---- \quad (4)$$

where d_h = hole diameter
K' = constant, dependant on hole and bit geometry, and the properties of the rock and carbide

K_1 and K_2 = constants, dependant on initial bit diameter

L = total hole length drilled

(ii) Alternative bit concepts

In the early 1980s the Research Division of Ingersoll Rand carried out an investigation into alternative drillsteel bit concepts as a part of their collaborative work for the Research Organization, their findings can be summarized as

(a) Knock-on bits manufactured with a full circumference at their rear very often became dislodged from the stem when the drillsteel was being removed from the hole because fragments of rock jammed the bit in the hole, (knock-on chisel bits with flats on the major diameter can alleviate this).

(b) Button bits failed because the buttons became loose in the bit.

(c) Cruciform and three winged chisel bits were considered unacceptable because they produced rifling of the holes.

During 1983/84 Sandvik introduced into the South African gold mines a drillsteel concept known as 'divided equipment.' This consisted of a high quality shank/stem, 22 and 25 mm across flats, available with forged or rubber collars, the latter complying with the Research Organization recommendations, and a screw on bit with a modified rope thread. In South Africa the evaluation concentrated on standard tungsten carbide chisel bits.

The initial evaluation at Buffelsfontein using pneumatic rockdrills proved to be a failure, the low thrust and torque available from these machines resulted in bits becoming loose and damaging the threads.

In 1986/87 an evaluation was carried out by Sandvik and the Group Industrial Engineering Department of Gencor at Unisel Gold mine using the emulsion hydraulic rockdrills. Initially forged collars were used but complaints from the operators of 'extra vibration in the drill' led to their early withdrawl. Rubber collared stems were introduced and the evaluation continued, proving to be successful from the engineering and mining aspects; 25 mm A/F stems achieved 550 m before replacement and interim results on 22 mm A/F stems indicate an average life of 380 m, with collar failure being the primary cause for removal, an average life of 80 m on the bits was achieved before they were undergauge. Before the equipment can be considered for commercial application it is necessary for Sandvik to determine equipment costs for large scale supply, an excercise that is currently in progress.

During a recent meeting with Sandvik at which the above was reported, two other aspects of drillsteel technology were discussed. Firstly, Sandvik claimed that their new design of button bits has affectively overcome loosening of the buttons, however, in their opinion button bits should be used only with high blow energy drills of 200/300 J, no benefit would be expected from using 42 mm diameter button bits with drills having a blow energy of 60/70 J. Secondly, they have developed 'three piece divided equipment', consisting of a bit, a stem and a shank, all connected by rope thread; this equipment is currently used only on large drifters where it is essential to have the fronthead flushing water supply to the drillsteel sealed from the rest of the drill in order to reduce the risk of polluting the oil with water. This concept offers no benefit for use with leg mounted drills, they said, and it would introduce extra cost and weight.

(iii) Imparting blow energy to drillsteel bits.

In the early 1980s, as a part of the collaborative programme, Ingersoll-Rand studied alternative methods for imparting blow energy to the drillsteel bit and evaluated two methods, namely struck bit and hurled bit.

In the struck bit drill concept the piston reciprocated and impacted, either directly or via an anvil, onto the shank of the drillsteel while it was thrust into the hole; energy transfer efficiencies to the rock of 50-60% were recorded in this mode of operation.

In the hurled bit drill concept the piston and drillsteel were connected, and reciprocated together; with the correct drill stand-off, energy transfer efficiencies to the rock of 70-80% were recorded.

It was concluded from the investigation that for leg mounted rockdrills the struck bit concept, with direct impact of the piston on the drillsteel shank, was superior because:

(a) The high case reaction of the hurled bit drill made handling extremely difficult.

(b) Maintaining the correct stand-off required by a hurled bit drill was very difficult; if the drill were overthrust it would stall, if underthrust it would loose performance.

(c) The use of an anvil offered no benefit in this application, rather it was a disadvantage in that it increased the complexity, length and mass of the drill.

3 LONG-HOLE ROCK DRILLING

During the 1950s it was proposed that long-hole stope drilling parallel to the face should be evaluated as a means of eliminating the hundreds of short holes normally drilled perpendicular to the face. It was envisaged that with this mining technique the face would require no daily examination, barring down or support, there would be no sockets to clean, and drilling could take place from the gulley where it would not interfere with work in the stope. The expected advantages of this drilling technique were better supervision and more rapid face advance. From the late 1950s onwards several gold mines evaluated long hole stope drilling and concluded, for a variety of reasons, that it was not a viable proposition.

In 1959 a feasibility trial was undertaken at Harmony gold mine,(9), and their findings were:

(i) Rigging the equipment in the gulley interfered with development work and if the gulley were of 'normal' dimensions only short length drill rods could be used, possibly adversely affecting accuracy. However, by using east and west sidings the equipment could be rigged without undue interference and long drill rods could be used.

(ii) A burden of ±1100 mm was the optimum, initially the burden used was ± 1200 mm.

(iii) It was possible to mine with a reduced stoping height and thus reduce the dilution of the ore. (During the trial, which consisted of 11 blasts giving a face advance of 12 m, the stoping height was reduced from 1,8 m to 1,0 m).

(iv) That the blasting technique adversely affected the normal stope support. However, by using a 'rolling rock pile', adequate support could be achieved. (This involved allowing a 10 m wide pile of blasted rock to build up before cleaning from the rear of the pile was started, access was possible between the face and the rock pile).

(v) The drills used initially were underpowered and poor penetration rates were achieved, with the larger drills used towards the end of the trial penetration rates of 0,3 m/min were obtained at hole depths of up to 24 m.

(vi) The measured gauge loss of 1 mm per 5,5 m drilled was considered to be acceptable.

(vii) That lining up the drill carriage was relatively simple, but time consuming, using clino-rules and tapes; some form of 'gun sight' was expected to be a relatively simple development that would significantly reduce setting up time.

(viii) Little deviation occurred in the drilled hole, that measured was only in the vertical plane and the maximum was ± 65 mm in a 24 m hole.

It was concluded that long-hole drilling parallel to the stope face from the top and bottom of a panel and blasting onto a rock pile was a viable mining method. However, the gulleys mined in advance of the face and ledged to accommodate the equipment created major rock mechanics problems, and as a result the trial was stopped.

In 1964 a similar evaluation was carried out at Durban Deep gold mine,[10]. In this instance a stope width as low as 400 mm was achieved using a similar mining method to Harmony gold mine. The work was discontinued because deviations up to 380 mm were measured in the hole and were considered to be unacceptable.

During a discussion with Messrs. J.A. Dilley and V.J. Solomon of the Mining Branch they commented on long-hole drilling trials carried out at Daggafontein and Buffelsfontein respectively. At the former gold mine the trial was conducted in the late 1950s and stopped because of over blasting while in the latter case, in the early 1960s, the trial was stopped because of difficulties in achieving drilling accuracy in the highly fractured rock and the over blasting that occurred due to the large quantity explosives used in the 75 mm diameter holes.

4 WIDE REEF ROCKDRILLING

Two techniques are normally employed for drilling the upper rows of holes in wide reefs over 2 m high. The first is to drill from a platform that is built at the face for every blast, this is both time consuming and labour intensive. In the second method drilling takes place from the footwall, producing inaccurate holes that result in poor blasts and bad hangingwall conditions.

In 1984/5 the Research Organization designed, manufactured and evaluated a 'high stope drill rig' at Western Areas,(11). The rig was self propelled with twin booms each carrying an emulsion hydraulic rockdrill, initially the Ingersoll-Rand FO-80 drifter and subsequently the Ingersoll-Rand WF030 rockdrill. The evaluation was stopped because the rig was difficult to operate, particularly when drilling the lower holes, very complex and difficult to maintain in the underground environment.

Subsequently, in 1986 Western Areas gold mine purchased a diesel powered mobile unit equipped with an adjustable 'sissors lift' platform on which was mounted an electro-hydraulic emulsion powerpack. Two leg mounted emulsion drills operated from the powerpack to drill both the face blast holes and roofbolt holes of the wide reef stopes, operating from the footwall or platform as appropriate.

The equipment is in daily use and the results obtained to date indicate that it is performing well, and increased hole accuracy and drilling rates are being obtained, the rockdrill and thrustleg reliability is significantly better than in other stoping applications with running costs well below expectation.

5 WATER ASSISTED ROCKDRILLING

In 1976 the United States Bureau of Mines contracted the Earth Mechanics Institute of the Colorado School of Mines to develop a new method for drilling small diameter roofbolt holes,(12). The concept of water assisted, rotary mechanical drilling was proposed and evaluated.

After a successful surface development programme the equipment was evaluated in the underground environment where further development requirements were identified and improved designs implemented, it was concluded that:

(i) penetration rates in some types of rock exceeded those of conventional rotary mechanical drilling and pure water jet drilling systems by a factor of two or more.

(ii) bit thrust and torque requirements were reduced to less than 25% of the values required by a rotary mechanical system.

(iii) bit life increased 75 fold over the standard rotary mechanical application, even in the most abrasive of partially metamorphosized sandstone.

(iv) the reliability of the system was considered to be acceptable; the major emphasis of the equipment development had been placed on the water pump or intensifier used, the hybrid bit design and the high pressure rotary coupling.

(v) water assisted, rotary mechanical drilling of small diameter roofbolt holes was a viable system.

6 HAND OPERATED ROCKDRILLING

6.1 Manually Thrust Drilling

In 1969 the Research Organization instituted an investigation into pneumatic rockdrills operating without the aid of mechanical thrust,(5). This technique for drilling a hole required the drill operator and his aide to lay on their back and thrust the drill, each with one leg.

Measurements of the manually applied thrust were compared with the thrust required to achieve the maximum penetration rate for various air pressures. It was observed that at an air pressure of 270 kPa manual thrusting could supply only 75% of the optimum, while at 680 kPa this fell to 43%.

In addition to the practical measurements, a theoretical model for the optimum thrust required by a pneumatic drill was formulated:

$$T_O = K\ (Ap \times Ps) \qquad ---- \quad (5)$$

where T_O = optimum thrust
K = a constant
Ap = area of piston head
Ps = supply pressure

6.2 Thrustleg Mounted Pneumatic Rockdrilling

As a part of the above programme the Research Organization also investigated the effectiveness of thrustleg mounted drilling using pneumatic drills. Observations were made underground at two gold mines using three different drills and tests were carried out in the laboratory for comparison(5), it was concluded that:

(i) The shape of the penetration rate-v-thrust curve was the same for all the machines regardless of air pressure;

ie. for a given supply pressure and machine output, as the applied thrust was increased the penetration rate increased up to a maximum value, thereafter further increase in thrust caused the penetration rate fall off until stall occurred.

(ii) The shape of the curve in the area of maximum penetration rate was normally quite flat. Applying a thrust somewhat lower than that at which the maximum penetration rate occurred would result in only a slight reduction in performance; from this consideration an 'optimum thrust' could be defined as that thrust which produces near peak penetration rate without causing excessive bit wear.

(iii) Pneumatic rockdrills operating underground with supply pressures of 480 to 550 kPa were greatly underthrust and increasing the thrust to the optimum value could increase the penetration rate by a factor of up to two, an increase of similar magnitude in drillsteel life would also be obtained.

(iv) The penetration rate of a drill thrusted by an in-line, rig mounted air leg or a screw feed was the same.

(v) The penetration rate using an ordinary airleg approached that for the in-line air leg as the angle of inclination neared 30°.

6.3 Pneumatic Power Supply Limitations

The pneumatic power supply systems currently used on the South African gold mines consist of compressors on the surface and extensive mine wide underground pipe networks. To power the rockdrills and other pneumatic equipment underground requires large volumes of compressed air and the friction losses over long distances result in low transmission efficiencies.

The extent of the pipe networks and the harsh environment make underground maintenance difficult, as a result there are many leaks that often go unattended to for long periods during which large quantities of compressed air are wasted. It is also very common for the compressed air underground to be used for unauthorized purposes because it is perceived to be a free and expendable commodity.

These factors result in low air pressure being available at the stope to power the pneumatic rockdrills, thus reducing their drilling performance. In many instances the reduced performance jeopardizes the drilling of a panel in a single shift and in order not to loose a 'blast' the miner introduces additional drills, further reducing the air pressure and drill performance.

Over a period of years the Research Organization and the Industry have conducted many surveys in an attempt to determine the electrical energy used for drilling. In a communication from Dr. D.G. Wymer, Director of Engineering Systems Laboratory, the latest findings relating to electrical energy utilization for pneumatic rockdrilling were summarized as follows:

(i) Theoretically, assuming no compressed air losses, the electrical power demand by the surface compressor station should be about 19 kW for each drill in use.

Assuming an overall drilling rate of 10 m/h, this equates to an energy consumption of 1,9 kWh/m.

(ii) Recent measurements of electric energy consumed by compressors on 17 gold mines, compared with drilling distances, indicated energy consumption rates varying from 15 to more than 30 kWh/m, depending to a large extent on the age and size of the mine.

When compared with the theoretical situation, and assuming that a relatively small quantity of air is used for other legitimate purposes, a high degree of wastage is apparent.

It has been estimated that, under the very best conditions, the electric energy required specifically for drilling should be 10 kWh/m.

(iii) Based on the air pressure and flow rate to a pneumatic rockdrill the input energy should be 0,75 kWh/m; assuming a typical percussive output power of 2 kW and a drilling rate of 10 m/h, the output energy is 0,2 kWh/m.

(iv) From the above, the energy efficiency of the power system is 7,5% and the overall efficiency of the complete drilling system is 2% at best. For the older and more extensive compressed air systems these efficiencies are lower still.

6.4 Pneumatic Rockdrill Limitations

The pneumatic drills currently used in South African gold mines stopes were first developed early in the 20th Century when the mines were shallow and the rock unfractured. In these conditions the mechanical interlinking of the rotation and percussion mechanisms via the rifle-bar produced an adequate mean stall torque, up to 15 Nm.

At the depths at which mining is now carried out the rock is highly fractured, discussed in section 2.2.1, and the torque output from the pneumatic rockdrill is inadequate for consistent effective drilling, the rotation often stalls, stalling the percussion at a critical time. In order for the operator to start drilling again it was necessary for him to reduce the thrust on the drill and withdraw the bit slightly to clear the obstruction, a time consuming and tiring operation. Also, as the mines have become deeper the compressed air pipe networks have become more extensive and the air pressure available at the stopes has decreased, as discussed in section 2.7, further increasing the instances of drill stalling and generally reducing the performance of the drills.

In order to overcome these design inadequacies in a pneumatic rockdrill a new concept would be required including mechanically separate percussion and rotation mechanisms, with increased percussive power output at the low air pressures available, and increased stall torque. However, to achieve these improvements would in all probability necessitate a heavier, larger drill.

A further limitation of the pneumatic rockdrill is the exhausting of the air directly to atmosphere, this creates a safety hazard because of the 'fogging' produced and a health hazard due to the excessive noise generated. In the latter case, the use of ear protection alone would be insufficient to reduce the operators noise exposure level below the expected legal requirements, while the addition of an exhaust muffler may be effective it would increase the mass of the drill and further reduce its efficiency.

6.5 Electrical Rockdrill Limitations

The Research Organization carried out a feasibility study into the use of electrically powered drills for stope blast hole drilling. Discussions were held with a potential manufacturer but the project was not pursued because of concerns regarding the safety of high voltage in the stope, heat generation at the drill, mechanical design considerations and mass.(13).

7 EMULSION POWERED HYDRAULIC ROCKDRILLS AND SYSTEMS

7.1 Background

In 1975 the Research Organization instituted a programme to develop a hydraulic drilling system that would overcome the inherent problems of the existing pneumatic systems, namely inefficient use of electrical energy and poor performance in fractured rock,[14]. The development programme was undertaken with the aid of two principle collaborators, Ingersoll-Rand Co. and Vickers Systems working on the rockdrill and thrustleg, and the emulsion pump respectively.

The initial specification for the rockdrill was that it must be lighter than the 'normal' pneumatic stoping drill but have same percussive output power, operate with mechanically separate rotation and percussive mechanisms and have a significantly higher torque output. It was intended that the drill be for one-man operation with the same productivity as a two-man pneumatic drill, thus saving direct drilling labour costs. (In addition, the reduced stope labour would ease man transport problems and lower the incidence of accidents at the face).

In 1977/8 the drill specification was changed, the mass was to be similar to that of a 'normal' pneumatic stoping drill, for use by a two-man crew, but with a much greater output power to yield a higher penetration rate. The motivation for changing the output was the realization that by increasing penetration rate in fractured rock the operator's aide would have less idle time between extracting drillsteels from completed holes, further increasing productivity[14].

An intensive engineering research and development programme was carried out in both South Africa and the United States of America,[13,15].

During this period of development, 1975 to 1982, several prototype versions of the rockdrill and thrustleg were extensively evaluated in both the laboratory and underground, and a large quantity of performance and reliability data was produced,[16].

7.2 Performance of Prototype Rockdrills

The prototype rockdrills operating underground in mid 1981 achieved mean drilling rates of 18,5 m/h while about 30% of all holes were drilled at a rate exceeding 20 m/h,(17), this was to be compared with the original expection of 14,5 m/h for the high powered emulsion drill.

In preparing target figures for the production trial a drilling rate of 20,4 m/h at a penetration rate of 0,44 m/min was agreed on, based on the previous experience.

7.3 Running Cost of Prototype Emulsion Drilling Systems

During the evaluation of prototype systems little relevant information was gained regarding running costs, however, target running costs were required for the production trial.

The drill and leg costs were synthesized using the knowledge of component lives gained during underground evaluation of the prototypes, and the cost of components produced for the trial. In this manner expected running costs of 43,93 c/m for the drill and 13,89 c/m for the leg were derived for parts only, while labour was expected to be 0,20 c/m and 0,12 c/m for the drill and leg respectively,(17). Other system running cost targets were prepared, based on information gathered from tests conducted during the development stage(18).

7.4 The Production Trial of Small Hydraulic Rockdrills

Following the successful engineering development of the emulsion rockdrilling system it was deemed necessary to conduct a production trial with the latest equipment, prior to its commercial application. The objectives of the trial were to determine drill performance figures, reliability data and running costs under production conditions and pressures.

To this end, a 10 month production trial was held at the number 6 shaft West Driefontein gold mine from June 1983 to March 1984 inclusive. The trial was conducted on an 830 m longwall at a depth of approximately 2200 m, the production from this area accounting for approximately 15% of the mine's total gold output.

The production trial was organized and funded by the Research Organization and overseen by a special Sub-Committee of the Research Advisory Committee. The Sub-Committee was to be kept appraised of the progress of the trial and give guidance on the programme, with the ultimate aim of their being able to assist with the implementation of the equipment to the production situation.

'Normal' mine labour, supervised by Research Organization personnel, was used to operate and repair the rockdrills and thrustlegs, with Research Organization personnel maintaining the hydraulic system,(19).

During the trial all aspects of performance and cost were monitored,(20), the pertinent results are summarized in Table 2 below with original expectations where applicable:

Table 2 SUMMARY OF PRODUCTION TRIAL RESULTS AND EXPECTATIONS

	AVERAGES		BEST RESULTS ACHIEVED	EXPECTATION
	FULL TRIAL	FINAL 3 MONTHS		
Penetration rate (m/min)	0,53	0,57	1,0	0,44
Drilling rate (m/h/drill)	19,5	22,9	50,2	20,4
Holes per drilling crew per shift	56,5	67,1	200,4	-
Running costs (c/m)*				
Hydraulic power supply	20,0	11,2	-	20,0
Hydraulic fluid	14,0	11,1	-	8,3
Pipes, hoses and fittings	13,2	17,6	-	20,8
Rockdrills	31,5	35,3	-	43,9
Thrustlegs	14,6	10,9	-	13,9
Electric power supply	0,9	0,5	-	5,0
Electric energy**	2,4	2,5	-	1,7
Sub Total	96,6	89,1		113,6
Drillsteel	15,0	15,0		15,0
TOTAL	111,6	104,1		128,6
Reliability (m/breakdown)				
Rockdrill	347	525	1000-1100***	1200
Thrustleg	-	750	2100-2150	-

* Costs are in February 1983 prices, excluding GST and labour.
** Electric energy consumption during the trial was 1,2 kWh/m(18) compared with at least 10 kWh/m for pneumatic drilling.
*** 7 occasions

The 10 month trial period was insufficient to determine accurately the long term running costs for all the equipment since many items had lives well in excess of this.

During the trial many areas requiring further engineering improvements were identified, particularly on the rockdrills, thrustlegs, hoses and stope manifolds(17,18). Many improvements were introduced and successfully implemented, increasing reliability and reducing costs during the latter months of the trial.

A suite of auxiliary equipment was designed to operate on mediums other than compressed air,(21). This equipment was introduced to the stope and the compressed air supply removed, making the area dependent on hydraulic drilling. At this stage of the trial an increased utilization of the hydraulic drills was noted, it can be assumed because no backup was then available.

After some initial resistance, the operators accepted the new system very well(22). It was apparent that they perceived the benefits of the drills having no air exhausted to the atmosphere, less noise and no fogging. The major complaint of the operators regarding the emulsion drill was its increased mass compared with the 'normal' stoping pneumatic drill. However, on occasions when hydraulic drill operators were required to revert to pneumatic machines they were very reticent, implying that the other benefits of the hydraulic drill overshadowed its increased mass.

In the course of the trial it became evident that though drill operator training had been adequate there was a need to train the other personnel involved in hydraulic drilling systems. It also became apparent that because the surface compressors were replaced by underground power units there was a requirement to restructure the management organization and involve the engineering department with the underground equipment[19,22,23].

At the end of the trial a series of recommendations was made regarding the future of hydraulic drilling in the stope[14], and these are summarized below:

(i) Hydraulic drilling be introduced to the mining industry as rapidly as possible, consistent with the retention of proper control, initially by controlled expansion of the production trial equipment at West Driefontein. This should be followed by other mines in the same area introducing the equipment fully, rather than a series of small mine trials widely separated geographically.

(ii) In order to alleviate possible concern regarding the future economic viability of the hydraulic drilling system, the economic implications of the use of the system should be re-appraised very critically to establish beyond doubt the likely benefits that could accrue from their use.

(iii) Ongoing engineering development should be encouraged and the Research Organization should continue to play an intimate role in that development and in the evaluation of improvements arising from it. In addition, as use increases, the Research Organization should take the initiative in formulating and implementing test procedures to enable the claims of manufacturers regarding improved equipment to be evaluated.

(iv) The technical and economic aspects of using hydraulic drilling in areas such as tunnelling or roofbolting should be investigated.

It was concluded that drilling stope blast holes using an emulsion powered rockdrilling system was viable. The majority of the expectations were achieved and in many instances surpassed. The one area where the expectations were not achieved was reliability, principally the rockdrill, and with development it was confidently expected that significant improvements would be achieved. The introduction of the system in a stoping environment had been generally accepted, however, changes to the organization structures and training of all personnel involved with the system would be necessary when large-scale operations commenced.

7.5 Application in Mines

At the conclusion of the production trial at West Driefontein gold mine in March 1984, that mine purchased the equipment and continued to operate it on a trial basis(24,25). Subsequently small mine trials were started at Free State Geduld, Western Holdings, Elandsrand, Unisel and Western Areas gold mines, the first three being narrow stope operations with two-man crews, while Unisel was a wide stope, (2 to 2,5 m), with one-man operation, and Western Areas was a wide reef application where drilling took place off a mobile, adjustable platform.

In all these Industry trials the mining performance was very good, typically 2 to 2½ times the drilling rate of pneumatic drills, see Table 3, but poor organization restricted the drilling shift to only about 2,5 hours. Experience has shown that where operators drilled for shifts of up to 4 hours they experienced no difficulty in handling the equipment. Given the proper organization, it should be possible to achieve productivities of 90 to 100 holes/shift for two-man crews in narrow stopes and about 75 holes/shift for one-man operators in wide stopes.

Table 3 PERFORMANCE OF EMULSION POWERED ROCKDRILLS USED FOR STOPING (Ref.23)

Production Site	No. in Crew	Steel Length m	Bit Diameter mm	Average Penetration Rate m/min	Average Drilling Rate holes/h
A	2	0,9	31 to 38	0,53	24,2
B	2	1,2	38 to 42	0,63	23,0
C	1*	1,2	27 to 38	0,57	19,0

*The large stoping width at this site (2 m to 2,5 m) made it possible to drill with single operators.

Throughout this period, up to the end of 1987, the Research Organization maintained its involvement with the engineering developments required to improve the reliability and running costs, and continued to evaluate the organization structures and training requirements of the system,(13,15,23).

The main engineering effort during this period focused on combating the effects of corrosion and finding a less dirt sensitive pump to replace the axial piston pump originally recommended. The corrosive stope environment, that is particularly bad in the Free State mines, affected all stoping equipment that was not corrosion resistant. In many instances components could be manufactured from corrosion resistant material and so alleviate the problem. However, the most severly affected areas were the rockdrill housings where the use of corrosion resistant material was not acceptable, development in this area is still ongoing,(13), though some design improvements have already been successfully introduced.

After the initial success with the axial piston pumps, poor quality control of critical components led to many premature dirt related breakdowns. The Research Organization subsequently evaluated various makes of less dirt sensitive plunger pumps and now recommend this type of unit(15).

As a result of these changes the rockdrill, thrustleg and pump reliabilities have improved and system running costs have been reduced.

In all the mine trials the organization and underground control have continued to be of concern(24,25). Areas particularly affected by poor control are emulsion use, which is excessive because leaking pipes are not repaired, and the large number of new hoses used, because damaged ones are scrapped and not repaired.

The latest improvements to the hydraulic drilling systems have been evaluated in order to determine the expected running cost for a system making best use of available equipment(25), and this is summarized below in Table 4.

Table 4 RUNNING COST EXPECTATIONS USING BEST AVAILABLE EQUIPMENT

System Component	Running cost (c/m) Refer to August 1987 excluding GST and Labour
Hydraulic power supply (including plunger pumps)	32,4
Hydraulic fluid @ 2 %	7,4
Pipes, hoses and fittings (assuming corrosion resistant end fitting and braid)	35,6
Rockdrills	65,0
Thrustlegs	24,8
Electric power supply	3,4
Total	168,6

In the course of the mine trials Free State Geduld, Western Holdings and Elandsrand experienced high running costs and poor rockdrill availability, a result of the small size of the trial and inappropriate organization. These mines suspended their trials pending improvements in reliability and costs, subsequently Western Holdings have restarted their evaluation on similar scale to the previous one.

The remaining three mines now consider hydraulic drilling in the stope to be a 'normal production tool' and West Driefontein and Western Areas are expanding their operations.

The equipment is now considered commercial and can be purchased from the manufacturers. Several options are available for plunger pumps while the rockdrills and thrustlegs are currently only available from the original collaborator, Ingersoll-Rand. It is of interest to note, however, that now hydraulic drilling has been proven to be viable at least one additional major rockdrill manufacturer is well advanced with the development of an alternative emulsion rockdrill and thrustleg.

8 DRILLING IN OTHER APPLICATIONS

When hydraulic rockdrilling is introduced on a mine wide basis, and compressed air systems are no longer in use, it will be necessary to use hydraulic drills to drill blast holes and roofbolt holes in areas other than the stope. To this end the Research Organization has evaluated the use of the emulsion hydraulic rockdrill in areas outside the stope. Results obtained in these situations[26] indicate that the drilling rates are at lease twice those for pneumatic machines and typical results are shown in Table 5.

Table 5 PERFORMANCE OF EMULSION POWERED ROCKDRILLS USED FOR DEVELOPMENT DRILLING

Type of Development	Holes per Round	Steel Length m	Bit Diameter mm	Penetration Rate m/min	Drilling Rate holes/h	Time to Complete Round
Haulages & Crosscuts 2,5m x 3,0m	62	2,2	36 to 42	0,49	10,3	3 h using 2 drills
Raises and Winzes 2,6m x 1,8m	24	2,2	36 to 42	0,41	9,6	2,5 h using 1 drill
Boxholes 1,5m x 1,5m	22	1,5	36 to 42	0,47	8,8	2,5 h using 1 drill

9 ADVANTAGES OF CLEAR WATER POWERED ROCKDRILLS

The Research Organization has developed the concept of hydro-power as a means of providing power to the workings of deep mines through the use of the hydrostatic head gained by service water as it descends in pipes within the shafts(27). The water is generally chilled in surface refrigeration plants to enable it to act as a cooling medium in the mine. This concept is very attractive because it has the potential for greatly simplifying the services infrastructure on the mines.

One major aspect of any underground powering system is that it must be capable of operating drills. In the ideal situation the rockdrills and thrustlegs should be capable of operating directly off the hydro-power reticulation and many benefits would accrue. For instance, because there is no additive to the powering medium running costs would be lowered, the hosing arrangement to the drill and leg would be simplified, a single high pressure hose would be used and flushing water derived from the drill exhaust water, there would be no return reticulation which would save costs and the electrical energy requirements would be lower than an emulsion electro-hydraulic system. All the benefits derived from emulsion electro-hydraulic drilling would also be available making hydro-powered drilling a very attractive concept.

To develop drills and legs to work in the absence of protection against corrosion and wear in the operating fluid is a very challenging task. Due to the uncertain nature of the development two design concepts have been pursued by the Research Organization(13), one uses the proven emulsion drill cycle and is being developed in collaboration with Ingersoll-Rand, whilst the second uses a novel concept, the direct valve cycle, and is being developed in collaboration with Innovatek Drills. It is envisaged that unless some major engineering problems are encountered, the drills and legs should be available commercially in 1991.

Currently, however, mines installing hydro-power can operate emulsion hydraulic rockdrills and thrustlegs via a water to emulsion hydro-transformer(27). The hydro-transformer is driven from the hydro-power column and pumps emulsion through a closed loop in the stope to power drills. A part of the exhaust water from the hydro-transformer is used as flushing water for the drills, simplifying the services to the stope.

Those mines using hydro-transformers from a hydro-power column to operate rockdrills and thrustlegs will gain some of the benefits available from a fully water-powered system. However, the early introduction of hydraulic drilling and hydro-power to mines will enable them to develop the infrastructure to cope with the new technology at an early stage and equip them for full implementation in the near future.

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CHAPTER 3

MOUNTED ROCKDRILLING

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MOUNTED ROCKDRILLING - SCHEME

- ROCKDRILLING — Mounted Drill Rigs
  - Conveyor mounted — AFCs and RFC
  - Narrow and Wide Reefs — High Stope Drill Rig
  - Industry Drill Rigs

1 INTRODUCTION

For a number of years the gold mining industry in South Africa has made various attempts at the introduction of drill rigs in narrow reef stopes with little success. According to British Standard 3618 a drill rig is 'any means of supporting a rockdrill at its work', and a drill rig is made up of a drill boom which is normally an adjustable arm which carries the rockdrill and holds it in selected positions. The drill boom is normally mounted on a drill carriage, this carriage permits the drills to be positioned at the face and to be removed before blasting.

Initially stope drill rigs have used pneumatic machines but there has been a tendency towards using hydraulic rockdrills on drill rigs. Drill rigs can be a stand alone type rig such as those used at Vaal Reefs and Randfontein Estates or can be a part of a mining system such as those mounted on the Reciprocating Flight Conveyors as used by the Chamber of Mines. The reasons for the change to hydraulic drilling are increased penetration rates, lower noise levels and lower operating costs. Present mining methods use the hand-held rockdrills with air legs, undesirable results of using hand-held rockdrills are:

(i) a greater number of rockdrill operators and machines are required,

(ii) dependence on operator skills and motivation,

(iii) incorrect burdens on the holes lead to bad breaking,

(iv) poor stope width control,

(v) reduced drill steel life,

(vi) hand and finger injuries during collaring the holes. (Immelman 1983).

Therefore work has been carried out in the industry and by the Chamber of Mines to alleviate the abovementioned problems by use of stope drill rigs.

Many advantages and disadvantages have been put forward concerning drill rigs but the disadvantages outweigh the advantages of the use of drill rigs at the present time in narrow reefs.

2 CHAMBER OF MINES DRILL RIGS

2.1 Conveyor Mounted Drill Rigs

During the development of mechanized face cleaning equipment, armoured face conveyors and reciprocating flight conveyor a number of drill rigs were attempted to be integrated with this equipment. On the Dowty-Meco AFC drill rigs as those previously used on Vaal Reefs were fitted to the conveyor.

Comments on the performance and operation of these rigs were 'the height of the conveyor led to problems with the drilling of the bottom holes when the conveyor was close to the face, and the conveyor had to be pulled back approximately 1 m to allow the rig to fit in. This greatly increased the mining cycle time'.

'Due mainly to the confined space, rig drilling was slow, taking at least one full shift when two two-drill rigs were used together with two hand-held machines at the ends of the conveyor'. 'Any drill rig arrangement must be developed specifically for this application if it is to contribute to the stope productivity. It is believed that the height of this conveyor and its plough guide preclude its effective use in stopes of less than 1,1 m'. (Buckmaster 1976).

A drill rig was used for a short period of time on the Westfalia Lunen. AFC results suggest that 'the rig had a number of limitations but the brief test period indicated a hole cycle rate of about 5,5 min. Therefore it was envisaged that 5 such drill rigs, properly developed, could drill the face in under 2,5 hours'. (Walczak and Fibrich 1978).

Drill rigs were not tried on the RFCs until RFC 3, initially a simple drill rig was used which consisted of a pair of pneumatic rockdrills mounted on simplex rigs fitted to the drill rail of the RFC as shown in Figure 1. This rig was designed such that pairs of holes, i.e. a top hole and bottom hole could be drilled simultaneously by the two drills at each drilling position, when not in use they were dismantled and stored in the back area. Also tested on the RFC 3 was an earlier version of the drilling jumbo which was capable of drilling a hole 1 m deep in 3 to 5 min. On RFC 4 and 4A Anderson Mavor pneumatic drill rigs were used these were similar to the type on RFC 3. Average cycle times of 8,8 min per pair of holes was achieved but a number of problems were encountered.

'The overall drilling cycle was significantly affected by the fact that the two drills were paired in the same rig. Studies showed that 15 per cent of the cycle time was attributed to one drill waiting on the other to complete drilling before moving to the next drilling position', and 'the inability to drill all holes on the face to the full depth of the drillsteel' because of bumps in the face. (Viljoen 1984).

Traversing of the rigs was a problem and the mounting of the rigs onto the RFC after the blast was slow and ardeous.

Consequently a more complicated drill rig was tested, this rig was initially tested at Randfontein Estates and then on RFC 4B at Vaal Reefs and is shown in Figure 2. One of the objectives of the RFC 4B trial was to evaluate the compatibility of the drill Jumbo with the RFC. The average penetration rate achieved for the machine was 1,08 min and it drilled over 9 500 holes at both sites. The Jumbo required a specialized parking rig situated behind the conveyor for during the blast, results suggest that the RFC and Jumbo were incompatible and that the Jumbo was still underdeveloped.

'During the trials the reliability of all equipment including the Jumbo, parking rig, conveyor and power packs was relatively poor' and 'tremendous difficulties were experienced in the traversing operation due to misalignment (out of specification) of pans'.

'Experience at this site indicates that the physical size of the parking rig combined with the slipping of the conveyor severely limits the support pattern required by such deep level mining particularly when poor hangingwall conditions are prevalent' (Bell 1981).

The RFC 5A returned to the use of more simpler rigs with paired hydraulic rockdrills, the mining system was hindered by the drill rig dead length commanding the stand off of the RFC and the frequent failure of the rockdrills.

In 1985 a drill rig was developed to work with the Low Profile Reciprocating Flight Conveyor. This rig was a middle level development between the complex drill Jumbo and the simplex drill rigs and is shown in Figure 3. Instead of the drillrig being removed from the RFC it would be attached to the blast barricade and folded away during the blast. From the tests with the LPRFC the following was concluded.

'In spite of all the mechanical problems and operational delays experienced with the drill rigs it has been shown that by using drilling rigs accurate and consistantly parallel holes could be drilled. Whether this translates into better footwall or hangingwall control was not ascertained but measurements indicated a more straighter face resulted from using the drill rigs. The drill rigs did not appreciably improve on results obtained from using hand-held hydraulic rockdrills'. (Critcher 1987).

The summarized problems encountered with conveyor mounted rig drilling are:

(i) The storage of the rigs during the blast.

(ii) Traversing the rig along the conveyor.

(iii) Articulation and misalignment of conveyor pans.

(iv) Only face holes can be drilled with conveyor mounted rigs.

(v) Complexity of the system.

Table 1 shows a calendar of events on the development of Chamber of Mines drill rigs, Table 2 gives a general description of the equipment and the results obtained.

2.2 Narrow and Wide Reef Drill Rigs

The Chamber of Mines has undertaken research in the area of drill rigs in narrow reefs as far back as 1965. A self-propelled rig which mounted four fast percussion drills in a pre-set pattern was developed. Also the Chamber during the early 1980's evaluated various pneumatic stope drill rigs available on the commercial market. The advantages and disadvantages from stope rig drilling which were ascertained were:

Advantages

(i) Holes are all drilled parallel, resulting in efficient breaking, even if the miner does not exert a close supervision.

(ii) Results in straight face facilitating cleaning, in that back lashing is minimized, and thereby better labour productivity is obtained.

(iii) Good stoping width control is achieved, when parting planes exist.

(iv) Good face advances per blast are evident, although the full length of the jumper is not fully utilized, because of the inevitable hole inclination.

(v) Labour rationalization is possible, in as much as one operator assistant could be allocated per 2 rock drill operators.

(vi) Rockdrill crews can easily be trained to install the rig correctly, following the mine standards.

Disadvantages

(i) Rigs require a certain amount of maintenance and have to be brought to surface or to the underground store for repairing. Unless capital outlays are increased, high unavailability shall persist. The expeditious on the spot repairing is time consuming and constitutes the main factor accounting for the dislike of the rig.

(ii) Can very easily drill into sockets, because continuous lines are painted on the face. Problems with the inspectorate can arise with the greatest probability.

(iii) Rockdrill crews tend to remove any temporary support interferring with their drilling. This can be extremely dangerous in block ground and can result in serious accidents.

(iv) Cannot be used in ledging.

(v) Applicable mainly in shallow mining where wide unsupported spans allow the installation of the rig. The use of stope drilling rigs is definitely restricted by the confinement imposed by narrow stoping widths and the maximum spans between the face and the permanent supports. The occurrence of large throw faults and dykes, and steeply inclined stopes add further burdens'. (Dos Santos, Laubscher and Deysel 1984).

It was concluded that the potential of achieving a straighter face by using rigs is diminished by sequential shortcomings and the heterogeneity of the ground. But that 'rigs become however a proposition in shallow mines of medium stoping widths, not requiring much face support and allowing wide spans between the face and the first line of permanent support'. (Dos Santos, Laubscher and Deysel 1984).

During 1980 it was decided that the mining industry would benefit if wide reef stopes were mechanized, by completing the drilling cycle on a more reliable and regular basis and with more effective use of labour. A pre-prototype mobile drilling system was developed to be used in stoping widths of 2,2 m to 3,2 m, negotiate 20° dips and footwall steps of 0,3 m. The machine consisted of a four wheel drive tractor unit carrying two hydraulic drill rigs, as shown in Figure 4 the machine was powered by two 45 kW 120 ℓ/min electro-hydraulic powerpacks operating on 5/95 emulsion. Both drill rigs consisted of a horizontal slide, primary vertical lift, secondary vertical lift, drill swing and a drill feed mounted on a rotary actuator, all these operations were controlled by manually operated valves. The machine was tested underground at Western Areas gold mine in 1984 and results indicated that the machine was highly complex and required a multitude of components to enable it to function correctly. The operator had difficulty in drilling the bottom holes because the body of the machine obscured his view. The average cycle time was 3,09 min with penetration rates of 0,65 m/min. The machine was able to drill 2 shot holes in 2,3 min and 2 roof bolt holes in 3,87 min. It was decided to terminate the work on this equipment in 1985 because of the low level of interest from the industry and 1986 budgetary reasons.

3 INDUSTRY DRILL RIGS

Stope drilling rigs an example is shown in Figure 5 have been tried in various mines in the industry with differing results. The following mines have attempted to instigate the use of drill rigs in narrow stopes.

Mine	Year	Type of Drill Rig
1) Vaal Reefs	1975	Mark IV rig, twin boom rig.
2) Vaal Reefs	1978-1981	Seco rigs and double boom Coalequip rigs.
3) Randfontein Estates	1976	Compair and Seco rigs.

4) Rustenburg Platinum Mine	1976	Demag crawler mounted stope drill rig.
5) Randfontein Estates	1986	Atlas Copco Stomec, tracked drilling rig.

From all these trials with differing drill rigs conclusions have been drawn as to their future in the gold mining industry. A few summarized concepts are:

'The stope drilling rig has a good potential for productivity improvement, it is on its own not necessarily the ultimate answer. The major problems at present are organizational and motivational ones'. (Dicks 1977).

'Without positive re-inforcement and proper back-up maintenance service the drill rig wil die a certain death and all potential productivity gains will be lost'. (van Wyngaard 1984).

'The prospects for success in hydraulic rig drilling may appear to be slim, especially in the near future. However with the increase in acceptance of lightweight hydraulic rockdrills in stoping, some form of rig drilling is necessary in order to utilize the power available in these rockdrills'. (Boshoff and Worsley 1984).

'With existing blasting accessories and mining practices it is unlikely that the use of rig drilling will improve stope control. (Pickering 1984).

Therefore at the present time it would seem that the disadvantages of using drill rigs in narrow stopes outweigh the possible advantages, the main problems being complexity, reliability, assembly, dismantling, cost and industry acceptance.

4 COMMENTS/CONCLUSIONS

Past and present experiences with stope drill rigs have shown that there is still more development required before an acceptable rig is produced. The present flexibility of using a hand-held rockdrill and air leg cannot be matched by a stope drill rig because of its complexity and difficulty in manoeuvering through the stope. The potential advantages of drill rigs is that of improved accuracy and drilling rates coupled with improved quality of life because of ease of operation of rigs using valve banks. These attributes may be the main factors in future motivation for stope drill rigs, but after all the previous attempts no single stope drill rig has come to fruition to lead the way for complete mechanization of drilling in narrow reefs. Attention is presently focused on the underground trials of the Stomec device at Randfontein Estates with its possible use in future gold mining in South Africa.

Use of drill rigs on mechanized mining systems (AFC and RFC's) have always been influenced by the trials of the mining machine. The main emphasis has been on achieving results from the mining machine rather then the drill rigs therefore development work on drill rigs has lagged behind the work carried out on mechanized mining systems. Main problems have been integrating drill rigs onto the mining machine with resulting problems of parking the rig, traversing, support problems, controlling the rig. The need for drill rigs on mechanized mining systems is essential, but they must thought of as a major component of the mining system which must work efficiently for the successful operation of the mining system.

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Table 1 CALENDAR SHOWING DEVELOPMENT OF COMRO DRILL RIGS

No.	Description	Mine	Shaft	Level	Stope	60	66	68	70	72	74	78	80	82	84	86	88
1	Self-propelling with 4 drills	-	-	-	-												
2	RFC3 Anderson-Mavor pneumatic Drill Rig and hydraulic drill rig	Vaal Reefs (West)	4	64	66K6												
3	RFC4 Pneumatic Drill Rig	Vaal Reefs (West)	4	66	66CW6(14)												
4	RFC4A Pneumatic Drill Rig	Vaal Reefs (South)	8	73	45Ex/cut												
5	IR Jumbo Drill Rig	Randfontein Estates	Cooke 2	90	-												
	Tested on RFC4B	Vaal Reefs (South)	8	73	DE53(W1)												
6	RFC5A IR drills on simple rigs	Hartebeestfontein	4	34	34SE41												
7	LPRFC COM Drill Rigs	Hartebeestfontein	6	75	77N25(95)												
8	High Stope Drill Rig	Western Areas	North shaft	45	9ExcutED												
9	Evaluation of pneumatic drill rigs	-	-	-	-												
10	AFC2 Dowty-Meco	Vaal Reefs	-	-	-												
	AFC3 Westfalia-Lunen	Blyvooruitzicht	70	-	-												

Table 2 COMRO DRILL RIGS, DESCRIPTION AND RESULTS

No.	Equipment	Description of Equipment	Results of Tests	Comments/Conclusions
1	Self-propelled rig with 4 percussive drills.	———	———	No information available. Source data Research Review 1965.
2	Anderson-Mavor pneumatic drilling and hydraulic drill rig (Mounted on RFC3).	2 conventional pneumatic jack hammers to drill top and bottom holes. Hydraulic drill jumbo using 5KW drifter, rail mounted on RFC. Placed on parking rigs during blast, 5/95 rock drill.	No results available on pneumatic or hydraulic drill rigs used on RFC3.	Pneumatic Drill Rig 1. Machine v/v sticking caused by access of dirt and rust to R/drill. 2. Repair of rigs involved process. 3. Need for positive indexing. Hydraulic Drill Rig 1. Capable of drilling 1 m hole in 3 to 5 minutes.
3	Anderson-Mavor pneumatic drill rig mounted on RFC4.	2 Seco S215 pneumatic rockdrills 2 slides mounted on manifold frame on drill rail of conveyor. (3 rigs used on the RFC). Each rig manned by 2 operators Simplex Rig. Used forged collar drill steels 1380 o/all length of steel.	1. Pressures of 530 kPa. 2. Average cycle times 8,3 min per pair of holes. 3. Penetration rates of 300 mm/min at 700 kPa. 4. Drilling cycle affected by 2 drills being paired on one rig interference of one drill with other. 5. Inability to drill to full depth of steel.	1. Mounting of Drill Rigs slow and arduous. 2. Traversing of rigs affected by roughness of footwall. 3. R/drills used for top hole were not interchangeable with r/drills to be used on the bottom holes. 4. Piping problems occurred on the manifold.
4	Anderson-Mavor pneumatic drill rig mounted on RFC4A.	As on RFC4.	No results available.	No information available.
5	Ingersoll-Rand Jumbo Drill Rig.	6 kW hydraulic drifter WF080 operating fluid 5/95 emulsion and utilized a parking rig behind the conveyor. 9 cylinders used to operate jumbo. 6 cylinders used on the parking rig. Used single 45 kW p/pack.	Randfontein Estates 1. Total number of holes drilled 6 502. 2. Penetration rate 1,08 m/min. 3. Cycle time 2,31 minutes. 4. Drifter drilled 200 holes between breakdowns. 5. Supply pressure 18,5 MPa. Vaal Reefs on RFC4B 1. Insufficient hydraulic power to operate Jumbo and conveyor. 2. Approximately 3 000 holes drilled. 3. Average hole depth 0,927 m. 4. Cycle time 4,07 min. 5. Penetration rate 0,84 m/min.	1. Traversing of the Jumbo was a problem. 2. Pan-misalignment created problems. 3. Operator inefficiency. 4. Specialized rig required for parking of Jumbo. 5. Parking rig size severely limited support pattern. 6. Fast and accurate drilling of holes. 7. Low noise level. 8. Easy to operate m/c. 9. Jamming of hoses. 10. Cleaning of dirt around parking rig impossible. 11. 'O' ring problems, 37 blown 'O' rings. 12. Valve problems 44 changed. 13. P/pack problems. 14. Lack of spares and quality of repairs. 15. Adverse mining conditions (corrosion). 16. RFC and Jumbo incompatible. 17. Large number of drill steel failures.
6	Anderson-Mavor hydraulic Drill Rig.	2 Ingersoll-Rand WS35 hydraulic rock drills (5/95 emulsion). 2 slides per rig (2 rigs per RFC). 3 cylinders per rig, rig stored behind barricade during blast.	No results available.	1. System hindered by the Drill Rig dead length commanding the stand off of the RFC. 2. Rock drills broke down often.

Table 2 (cont.)

No.	Equipment	Description of Equipment	Results of Tests	Comments/Conclusions
7	COM Drill Rig LPRFC.	Single mount WF035 Ingersoll-Rand 5 kW rockdrill (2 rigs on LPRFC). Drill slide incorporating a positioning cylinder and thrust cylinder. Traversing carried out by a walking cylinder and indexing cylinder. Height adjustment on drill slide via a raise and lower cylinder. Total weight of drill rig 511 kg. Drill steel 1 500 Wimet steel. Operating fluid 5/95 emulsion.	1. Average penetration rate 0,66 m/min. 2. Average cycle time 2,96 min. 3. Emulsion usage 1,2 ℓ/hole. 4. Accurate drilling of holes was accomplished. 5. Total holes drilled 528.	1. Drilled parallel and accurate holes. 2. Traversing of drill rig between pans was arduous and time consuming. 3. Too many cylinders used on rig. 4. Corrosion of rig created problems. 5. Complex back-up services used on rig. 6. Hosing layout was complex. 7. Penetration rates were the same as hand-held rockdrills. 8. Cleaning of each drill rig after the blast was time consuming. 9. Manoeuvering of the rig around face side props was time consuming.
8	Wide Reef Drill Rig (High Stope Drill Rig).	4 wheel-drive tractor unit with 2 hydraulic drill rigs. Machine powered by two 45 kW p/packs (5/95 emulsion). Ingersoll-Rand rockdrills WF035. Steering and movement of the M/C was controlled from a remote control drive box on infra-red transmitter. 18 cylinders used on M/C. Max. length 7,46 m Min. length 5,46 m Width 1,78 m Max. height 3,1 m Min. height 2,1 m Mass 8 000 kg Max. drilled hole length 1 340 mm Drill steel 1,6 m o/all.	1. Average penetration rates 0,65 m/min. 2. Cycle time 3,09 min. 3. Holes drilled by drifters 418. 4. Holes drilled by r/drills 454. 5. Various drilling patterns were tried with varying success. 6. M/C could drill 2 shot holes in 2,3 min. 7. M/C could drill 2 roofbolt holes in 3,87 min.	1. Reliability problems with drill rig. 2. P/pack problems. 3. 2 drills on one chassis tended to limit availability of drills. 4. Valving problems occurred. 5. Hydraulic hoses should have been changed to steel piping. 6. Rig was too large to fit into mine cages. 7. Drill steel failures caused by over thrusting and side loads. 8. Complex system.
9	Evaluation of Pneumatic Drill Rigs.	Various commercially available pneumatic drill rigs. 1. Compair Drill Rig. 2. Coalequip Drill Rig.	General observations made on each Drill Rig. <u>Advantages</u> 1. Accurate drilling of holes. 2. Improved stoping conditions. 3. Greater penetration rates. 4. Labour saving. 5. Reduction in operator fatigue. <u>Disadvantages</u> 1. Complex system. 2. Weight problem. 3. Time consuming for installation. 4. Increased training required. 5. Additional capital and maintenance costs. 6. Prop support hinders traversing.	1. Possible financial and other gains could be substantial. 2. Possible reduction in costs of drilling, explosives, barricades. 3. Improved stope face advance and reduction in required face length. 4. Fewer accidents. 5. Size of rig is a problem. 6. If two r/drills are used on one rig, one drill can hold up the other. 7. Long setting up times, assembly and dismantling times. 8. Difficulties with rigs coping with faults.
10	AFC2 Dowty-Meco AFC3 Westfalia-Lunen	Twin boom rig as developed by Vaal Reefs (Mk IV Rig 1974). No description of drill rig used on the Westfalia Lunen machine.	No results available.	1. Problem of drilling bottom holes with conveyor close to the face. 2. Drill rigs could not drill delivery and tail-end ends. 3. Rig drilling was slow mainly due to the confined space. 4. The height of the conveyor excluded its effective use in stopes of less than 1

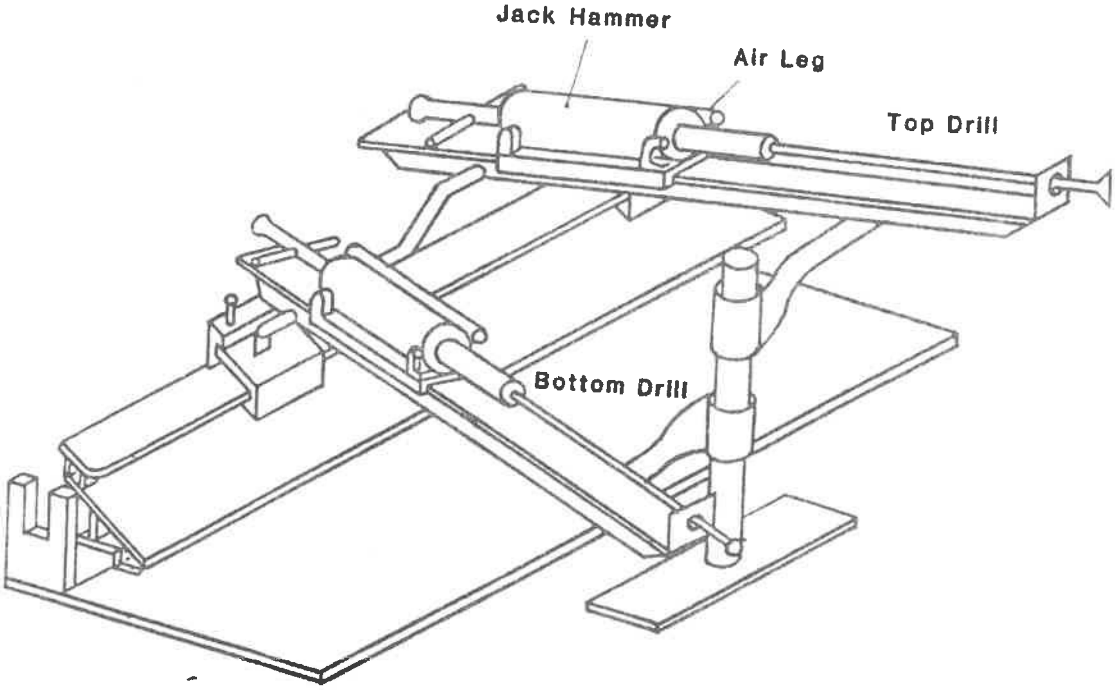

FIGURE 1: Pneumatic Rockdrills on Simple Drill Rig for use with Reciprocating Flight Conveyor

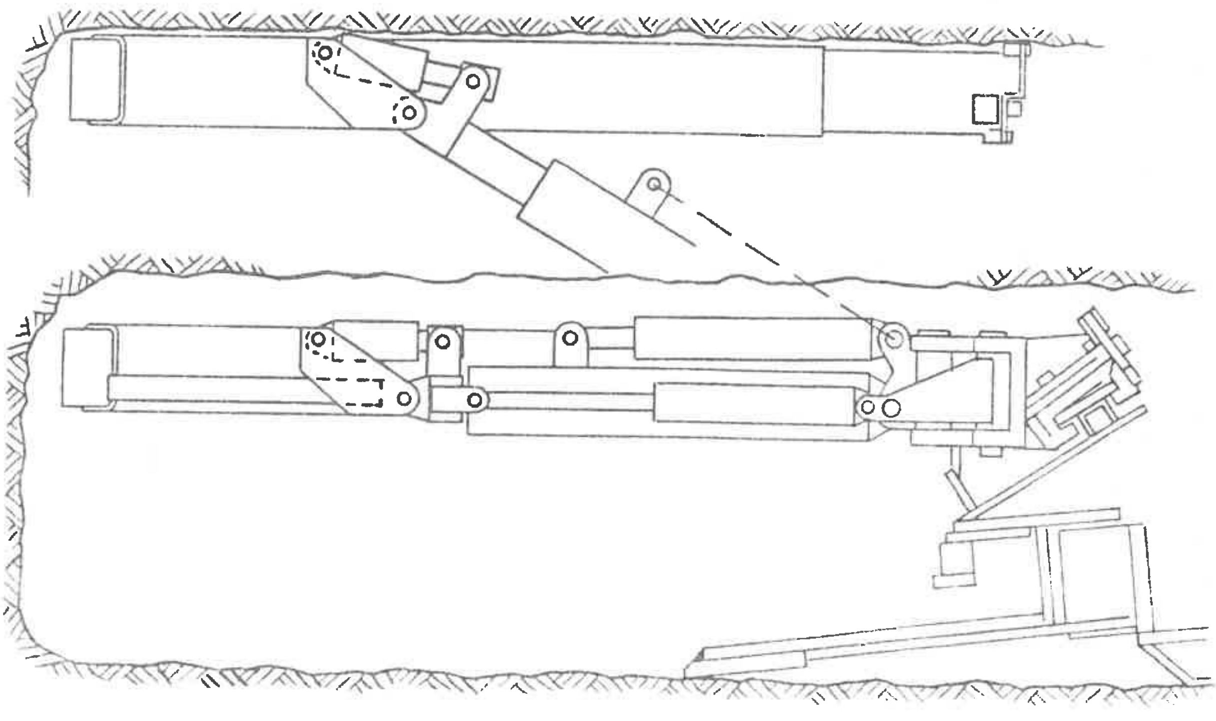

FIGURE 2: Drill Rig used on RFC 4B

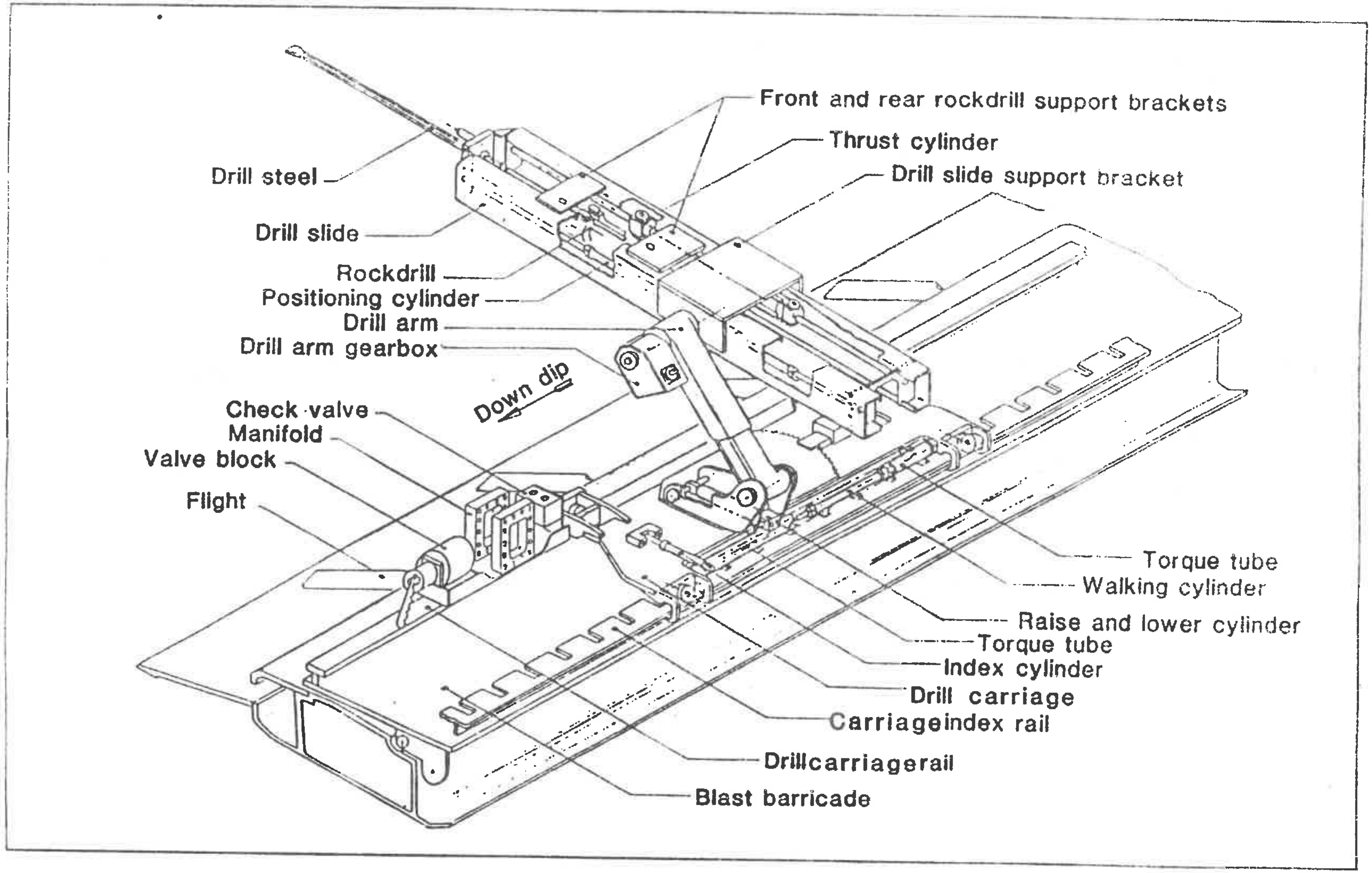

Figure 3 GENERAL ARRANGEMENT OF DRILL RIG ON LPRFC.

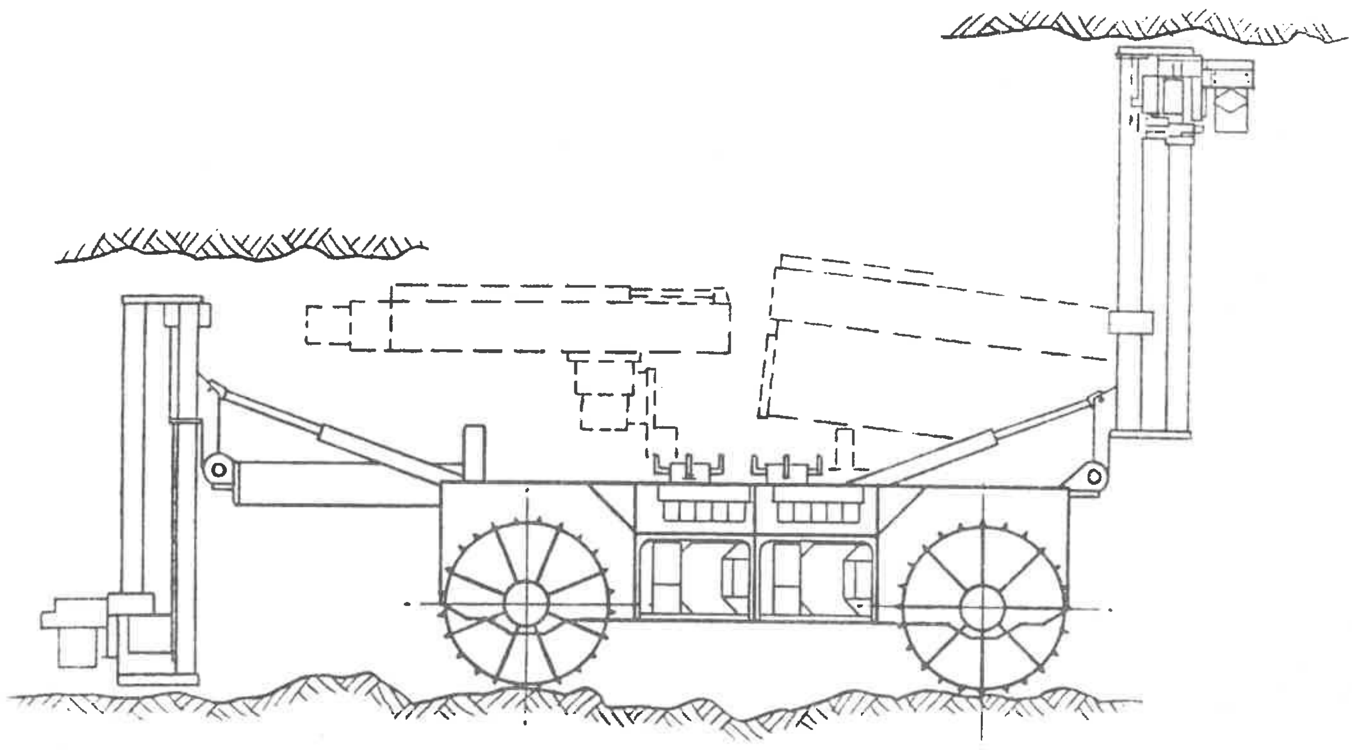

FIGURE 4: High Stope Drill Rig

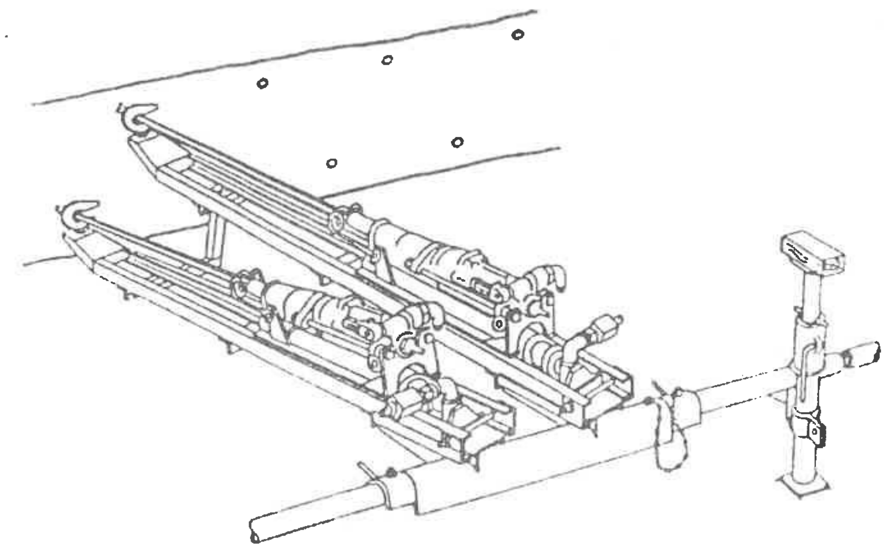

FIGURE 5 Stope-drilling rig

CHAPTER 4

NON-EXPLOSIVE ROCKBREAKING

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NON-EXPLOSIVE ROCKBREAKING - SCHEME

- NON-EXPLOSIVE ROCKBREAKING
  - Fundamentals
    - Rock Condition
    - Rockbreaking
      - Physical Properties
      - Location Characteristics
      - Fracture Modes
      - Principal Methods
      - Combination Methods
    - Performance Criteria
    - Environmental Compatibility
      - Dimensional Limitations
      - System Requirements
      - Pollution
    - Economics
  - Methods for Quartzite
    - Breaking from Drilled Holes
      - Operating Principles
      - High Pressure Gas Devices
      - Fluid Pressure Devices
      - Expanding Solids
    - Breaking from Slots
      - Mechanical
      - Thermal
    - Thermal
      - External Heat
      - Internal Heat
    - Fluid Dynamic
      - Continuous Jets
      - High Pressure Abrasive Jets
      - Low Pressure Abrasive Jets
      - Improved Abrasive Jet Cutting
      - Non-Abrasive
    - Mechanical
      - Drag Bit Cutting
      - Roller Cutting
      - Diamond Cutting
      - Impact Breaking

INTRODUCTION

Non-explosive methods of rockbreaking offer significant potential for improving the stoping operation. The continuous generation of relatively small amounts of rock facilitate mechanization, in particular the crucial rockhandling function. This in turn results in improved labour productivity. The elimination of the cyclic opertion of stoping, which in conventional mining is enforced by breaking the rock with blasting, enables continuous operation, which again promotes the more effective use of labour. It also gives scope for higher face advance rates and concentrated mining, which justifies the investment of more capital for equipment. Also non-explosive methods of mining can either mine selectively or permit the sorting of waste rock in the face for use as backfill, if conditions are favourable. This can result in a considerable reduction of dilution, besides the advantages which are typically derived from backfill. The enconomic benefits which can be gained from non-explosive methods of rockbreaking are therefore of sufficient magnitude to justify the extensive research and development efforts which have taken place since 1965

This report gives an overview of all the rockbreaking methods which were investigated by the Chamber of Mines Research Organization and others for the particularly arduous conditions of deep level gold mining in hard rock. It summarizes the fundamental research work, the results of the experimentation and the present state of knowledge. Finally, it explains the direction present investigations are taking and comments on the scope of the development potential of some methods in the light of recent technological advances. The report is structured in such a way as to facilitate the understanding of the fundamentals, which are involved for the different methods.

NON-EXPLOSIVE ROCKBREAKING FOR SOUTH AFRICAN GOLD MINES

1. FUNDAMENTALS.

1.1 Type and conditions of rock encountered on South African gold mines.

The potential viability of non-exploive rockbreaking techniques is critically dependent on the physical properties and characteristics of the rock. This sub-section summarizes the relevant findings of many years of research into these aspects of the gold-bearing rocks.

1.1.1. Type of rock and important physical properties for rockbreaking.

The rocks encountered in and around the gold bearing reefs are quartzitic except for intermittent layers of shale [1]. The composition and the physical properties of quartzite can vary significantly from area to area and even in different layers in one location.

As seen in Table 1 and 2, uniaxial compressive strength can vary from 146 MPa to 466 MPa. Typically the uniaxial tensile strength is between 6% to 15% of compressive strength and the shear strength is from 8% to 30% of compressive strength.

Variations of indentation strength as a function of rock composition, in this case quartz contents, are shown in Figure 1 [2].

TABLE 1. ELASTIC AND STRENGTH PROPERTIES OF WITWATERSRAND SEDIMENTS

Horizon	σ_t	σ_c	S_o	E	ν	k
A. *Evander Goldfield*						
Drab quartzite		257	78			
Intermediate quartzite		269(71)	64	77(3)	0,17	6,9
HW1 quartzite		158		71		
HW2 quartzite		205		76		
MK1 quartzite		263		80		
MK3 quartite		249(11)	73	78(7)	0,16	3,1
LK1 quartzite		466	58	87	0,11	8,6
MBO1 quartzite		455	46	83	0,10	16,6
B. *Central Rand Goldfield*						
HW quartzite — Main Reef		231(47)	—	84(6)	0,21(,01)	—
FW quartzite — Main Reef		257(51)	—	87(8)	0,19(,02)	—
HW quartzite — Composite Reef		285(65)	—	83(3)	0,14(,01)	—
Composite Reef		292(22)	—	87(3)	0,16(0,05)	—
FW quartzite — Main Reef leader		235(33)	—	77(4)	0,19(0,03)	—
C. *Carletonville Goldfield*						
Transition zone:						
Booysens shale to Cobble Reef quartzite		263	0,17	76	3,8	
Quartzites:						
Cobble to Livingstone reefs		167		87		
Quartzites:						
Johnstone reefs	15(1)	142(37)	22(6)	73(5)	0,20(0,02)	
HW of Green Bar	19(4)	218(58)	35(9)	87(15)	0,20(0,03)	
Green Bar quartzites	16(2)	222(85)	18(9)	87(5)	0,24(0,05)	9,9(3,5)
HW of Carbon leader	16(4)	250(105)	49(21)	87(5)	0,17(0,01)	9,1(2,4)
FW quartzite of Carbon leader	19(6)	202(45)	38(10)	74(10)	0,19(0,02)	7,5(1,7)
Upper square pebble quartzite	22(1)	255(107)	49(25)	78(11)	0,20(0,04)	8,5(2,8)
Middle square pebble quartzite	21(1)	219(25)	35(8)	71(7)	0,23(0,04)	9,0(2,8)
Top Jeppestown quartzite		190(65)	30	85(17)	0,22	14,9
Top Jeppestown shales		260(8)	70(5)	71(2)	0,23(0,06)	3,3(1)
D. *Klerksdorp Goldfield*						
GE7 — Dennys quartzite	22	308(66)	40(19)	76(12)	0,12(0,04)	17,2(9)
GE9 — Conglomerates		187	24	87	0,09	15,9
MBA, MB1 — argillaceous quartzites and shales		218(34)	46(7)	73(6)	0,23(,03)	4,7(1,5)
MB2, MB3 — argillaceous quartzites		209(39)	45(9)	74(4)	0,17(0,03)	6,2(1,2)
MB4 — argillaceous quartzites	17(4)	189(44)	35(14)	70(7)	0,20(0,04)	8,0(3,0)
Vaal Reef quartzites	18(1)	203(60)	36(14)	78(8)	0,19(0,05)	7,9(3,2)
MB5 — siliceous quartzites		268(55)	44(9)	81(8)	0,19(0,05)	12,8(6,3)
MB5 — argillaceous quartzite	27(14)	180(53)	40(18)	74(5)	0,18(0,03)	8,9(5,5)
MB6 — argillaceous quartzites		177(46)	33(15)	73(11)	0,19(0,04)	6,4(2,2)
MB7 — siliceous quartzites		254(50)	31(4)	90(15)	0,19(0,07)	11,0(2,3)
E. *Welkom Goldfield*						
Aandenk quartzites		192(27)	35(10)	66(3)	0,16(0,01)	8,7(3,7)
Dagbreek quartzites		159(47)	34(6)	67(12)	0,18(0,02)	6,7(2)
Leader Reef zone		146(13)	29	60	0,10	9,6
Leader Reef quartzite		271(85)	45	83(14)	0,12	5,2
Basal quartzite		195(8)	38	74	0,19	6,7
UF1 — Zone I siliceous quartzites	17	154(78)	40(7)	76(9)	0,16(0,04)	9,0(5,7)

σ_c, σ_t — uniaxial compressive and tensile strengths (MPa)
S_o — computed shear strength from triaxial test data (MPa)
k — factor indicating increase in failure strength, σ_1, during triaxial compression; according to equation: $\sigma_1 = \sigma_c + k\sigma_3$, where σ_3 is confining pressure.
E — Young's modulus (GPa).
ν — Poisson's ratio.
Standard deviations in parentheses.

TABLE 2. AVERAGE UNIAXIAL COMPRESSIVE STRENGTHS (IN MPa) OF HANGING WALL AND AND FOOTWALL STRATA IN THE CARLETONVILLE GOLDFIELD. STANDARD DEVIATIONS GIVEN IN PARENTHESES

Rock type	Doornfontein	Western Deep Levels	West Driefontein	East Driefontein
HW of Green Bar	208(61)	239	—	165
Green Bar	277(90)	—	266(94)	296
HW of Carbon Leader	237(105)	—	124(18)	—
FW of Carbon Leader	199(37)	—	222(109)	—
Upper Sq. Pebble Marker	228(56)	399	—	—
Middle Sq. Pebble Marker	215(38)	—	199	253
Lower Sq. Pebble Marker	197	—	—	223(8)
Top Jeppestown Quartzite	167	245	—	173(18)
Top Jeppestown Shale	152	251	—	243(39)

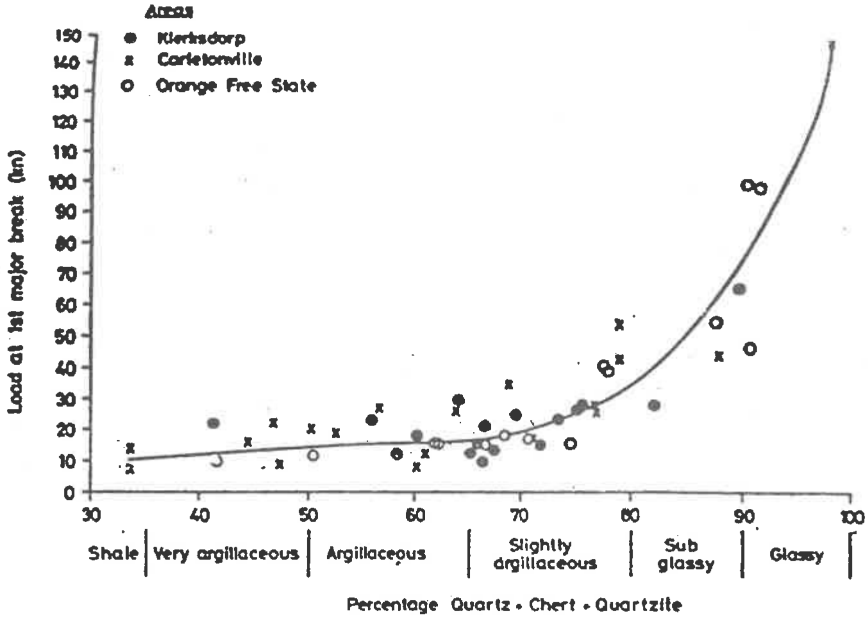


Figure 1. INDENTATION STRENGTH OF QUARTZITE AS A FUNCTION OF THE QUARTZ CONTENT

TABLE 3. PROPERTIES OF ROCKS TESTED

Rock	σ_c MN/m²	$\sigma_{ST(9)}$ MN/m²	$\sigma_{ST(15)}$ MN/m²	σ_{ST}/σ_c (9)	σ_{ST}/σ_c (15)
ERPM Quartzite	180	2 430	1 870	13,5	10
Free State Quartzite	260	1 940	1 470	7,5	5,6
Marievale Quartzite	117	1 200	960	10,3	8,2
Buffelsfontein Quartzite	55	950	740	17,2	13,4

σ_c = uniaxial compressive strength
$\sigma_{ST(9)}$ = stamp load mean strength (stamp diameter 9 mm)
$\sigma_{ST(15)}$ = stamp load mean strength (stamp diameter 15 mm)

Unfortunately there is no single physical property which can be directly related to rockbreaking performance as is evident from the example of Table 3 [3]. Further complications arise from the effect of stress gradients on strength [4] and the effect of loading rates on strength [5].

Other physical properties which are relevant mainly for thermal methods of breaking quartzite are:

Low water content - relevant for poor absorption of microwaves [6].

Low electrical conductivity - unsuitable for resistance heating [7].

Marginally suitable values for relative dielectric constant and dissipation factor - potentially suitable for dielectric heating [8].

Phase changes from α - β at 573°^{C} with a volume increase of approximately 1% - useful for thermal spalling [9].

1.1.2 Special rock characteristics due to location.

The presently mined reefs are located at an average depth of approximately 2000 m.

Vertical stresses of approximately 60 MPa are typically measured at that depth and horizontal stresses of 25-55 MPa [1]. Confining pressure can have a significant effect on strength properties as shown in Figure 2 [10].

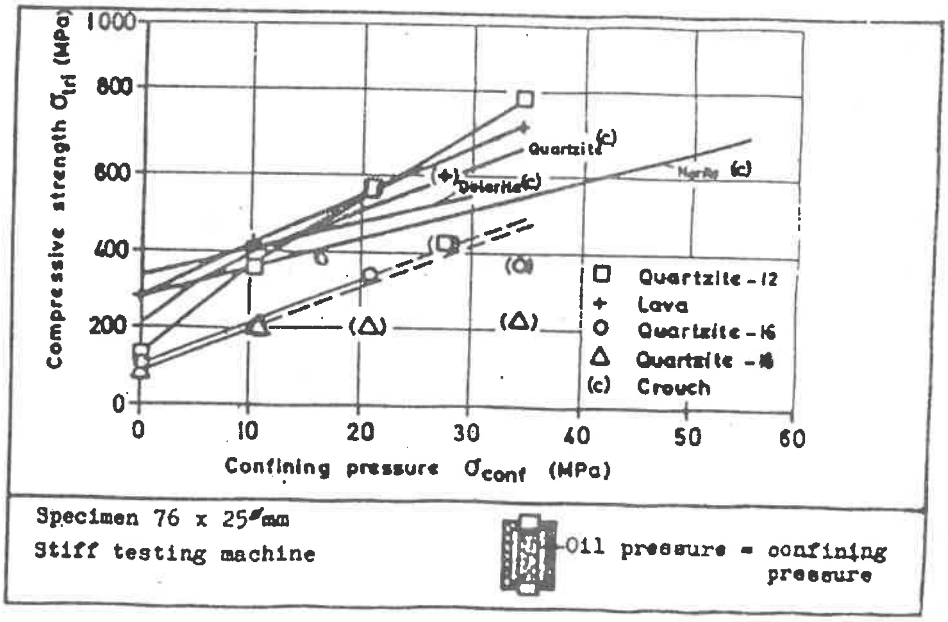


Figure 2. STRENGTH TESTS - TRIAXIAL COMPRESSIVE STRENTH AS A FUNCTION OF CONFINING PRESSURE

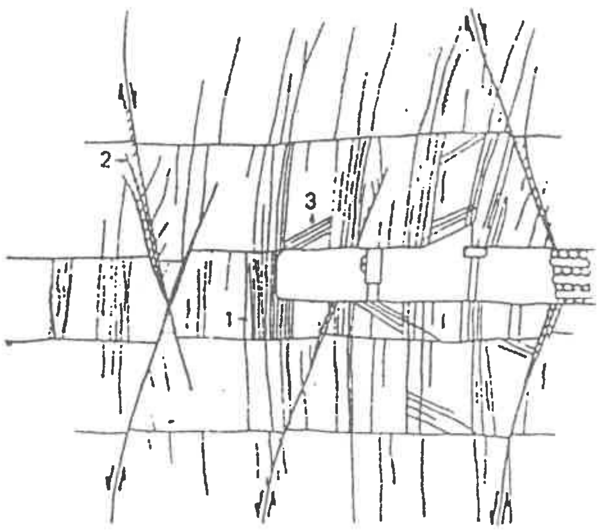


Figure 3. PATTERN OF THE THREE TYPES OF FRACTURES THAT OCCUR AROUND A STOPE FACE

If an excavation is made under the above conditions the stress field is disturbed, resulting in high stress concentrations close to the edge of the excavation. This can raise the stress above the strength of the rock, resulting in fracturing. This again shifts the peak stress further into the rock mass. At a certain energy release rate face parallel extension fractures are first formed, the depth of which are related to the energy release rate [11]. If the energy release rate is further increased shear fractures are formed several metres into the stope face. This combined with geological discontinuities results in very uneven stress distribution for the rock which is subjected to face parallel extension fracturing close to the stope face. This again causes uneven fracturing ranging from totally unfractured - so called hard patches - to totally crushed rock, which virtually falls out by itself.

Therefore the typical fracture conditions are as shown in Figure 3. These different conditions can all be found over the length of a stope panel. Unfractured rock of a few metres in length can be right next to a highly fractured area. The same applies in terms of depth except that rapid changes can occur over distances of less than 1 metre. New fractures form mainly as a result of advancing the face. A good indication of this is given in Figure 4 for a stope advanced in steps by blasting [47].

An impression of the typical distribution of the fracture conditions can be gained from Figure 5.

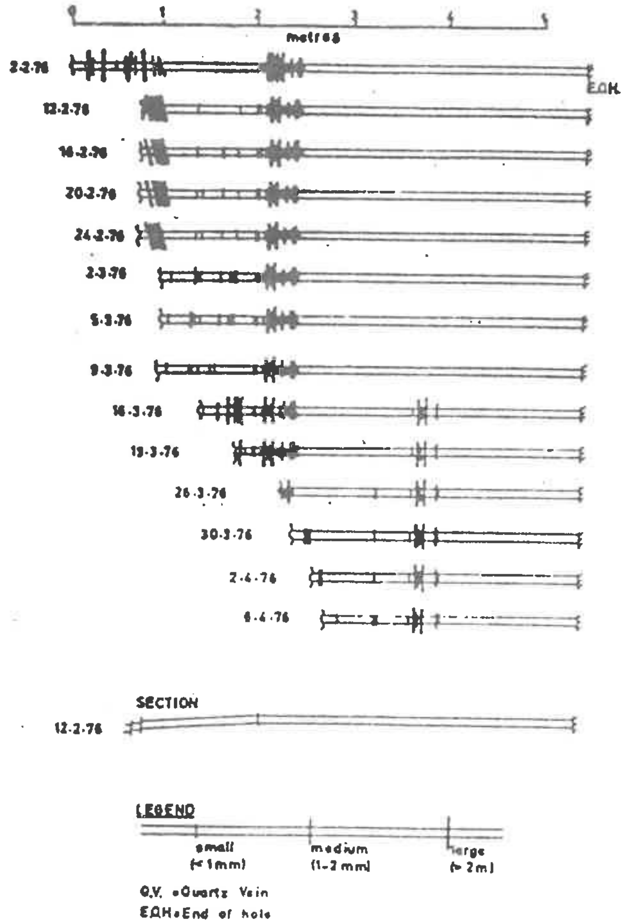


Figure 4. DIAGRAM SHOWING PLAN VIEWS OF A PETROSCOPE HOLE WITH FRACTURE OBSERVATIONS ON DATES SHOWN. (FOR ENERGY RELEASE RATE OF 19 MJ/m^2)

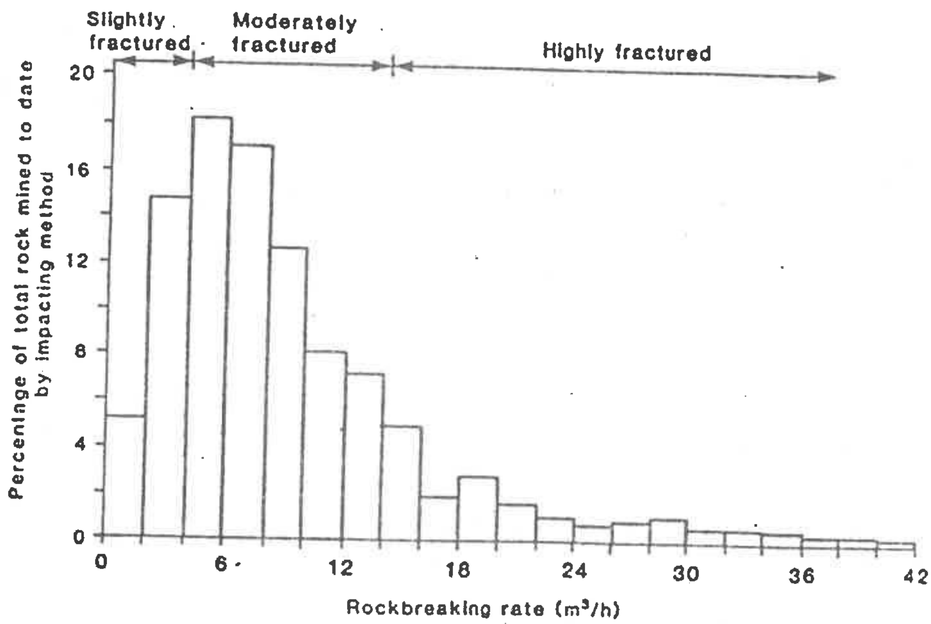


Figure 5. THE DISTRIBUTION OF ROCK MINED AT VARIOUS ROCKBREAKING RATES BY HYDRAULIC IMPACT HAMMER WITH BLOW ENERGIES BETWEEN 2 400 AND 2 700 J AND AT A FREQUENCY OF 3 Hz

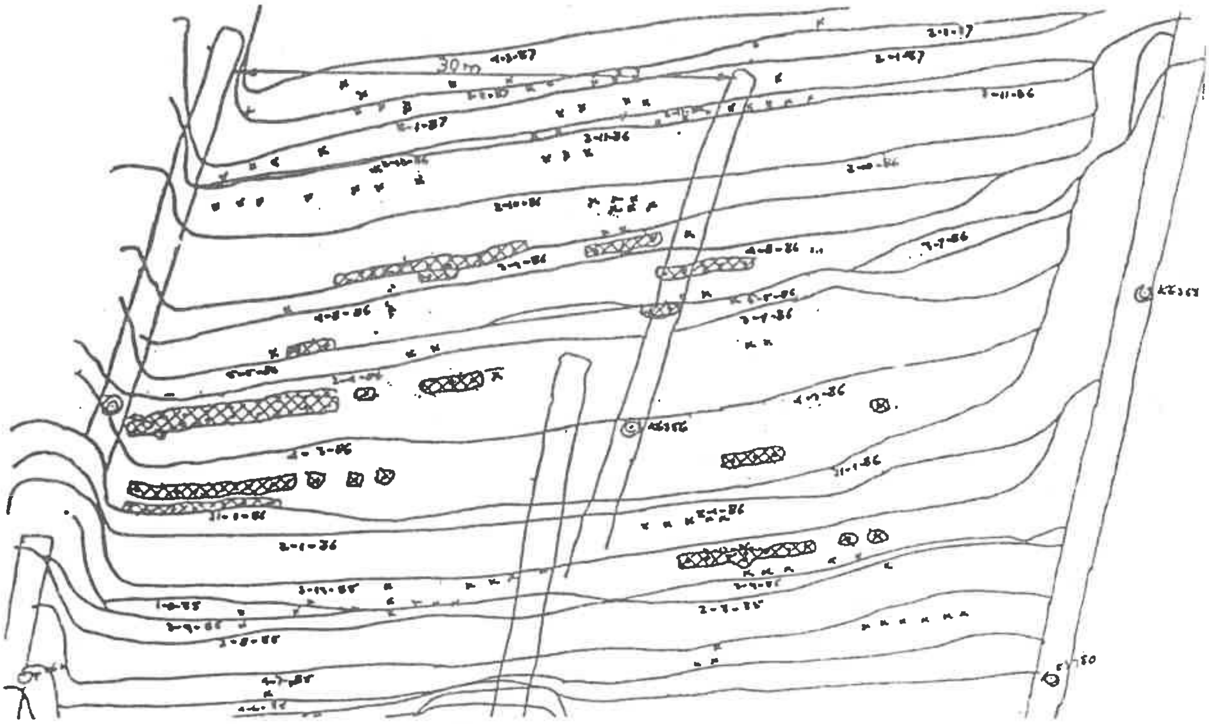
Although it indicates rockbreaking rates for impacting [12], it is a direct measure of rock fracture conditions, because all other parameters were kept constant. Present knowledge of hard patches does not yet permit the prediction of their likely locations or their likely extent. However, their very uneven distribution suggests a correlation to geological disturbances, and possibly excavation geometry. Figure 6 gives a recent example of hard patch distributions. Methods suitable to determine the existence of hard patches ahead of the stope face have not yet been found.

As evident from Figure 5, the fracture conditions have an overriding effect on rockbreaking effectiveness. Methods which can exploit the fractures have a significant advantage. On the other hand methods which are ineffective over part of the range of conditions, particularly for little or no fracturing, suffer the complication of requiring assistance by other methods which are effective.

The fractured state of the foot wall and hanging wall also has a bearing on the suitability of some rockbreaking methods, since it imposes limits on the staking forces, which may be required to absorb reaction forces from the rockbreaking device.

1.2 Rockbreaking

Rockbreaking methods are characterized by the techniques used to apply energy to the rock and the resulting fragmentation. This sub-section



[XXX] HARDPATCHES

×× Areas with mining rates<2 m²/h

Figure 6. AREAS MINED BY SECOND LAST IMPACT RIPPING SYSTEM FROM JULY 1985 TO MARCH 1987 WITH INDICATION OF HARD PATCHES

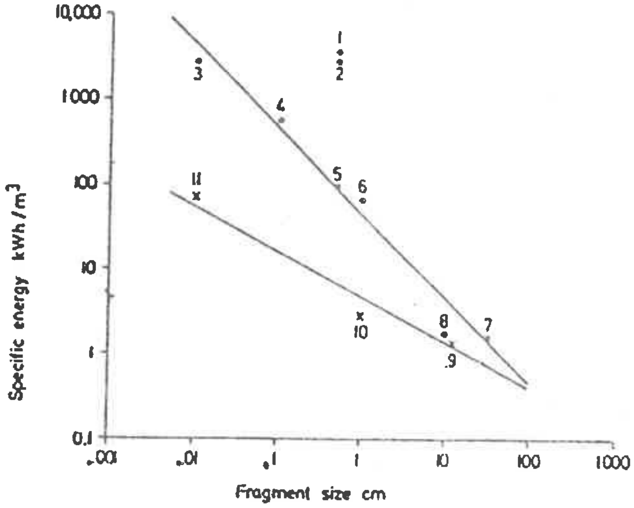


Figure 7. SPECIFIC ENERGY AS A FUNCTION OF NOMINAL FRAGMENT SIZE FOR PRIMARY (•) AND SECONDARY (X) ROCKBREAKING METHODS IN QUARTZITE

1. Jet-piercing. — 2. Erosion drilling (granite). — 3. Diamond cutting or drilling. — 4. Percussion drilling. — 5. Drag bit cutting or drilling. — 6. Roller bit drilling. — 7. Impact-driven wedge. — 8. Explosives. — 9. Jaw crusher. — 10. Giratory crusher. — 11. Milling.

summarizes what is known about the basic means by which rock may be broken non-explosively and the characteristics of the classes of rockbreaking methods.

1.2.1 Rock fracture modes and specific energy requirements.

The various rockbreaking methods fragment rock in different ways and produce fragments of particular size ranges. It has been shown, that the specific energy requirements for rockbreaking are inversely proportional to the fragment size [13]. Figure 7 indicates also that more energy is required for primary rockbreaking than for comminution. It is therefore more energy efficient to aim for the largest possible fragment size in primary breaking which is compatible with rockhandling, and to use comminution to reduce the fragments to the finally required size. This approach also leads to faster breaking rates.

Most rockbreaking methods rely on mechanical stress with the exception of chemical methods and thermal methods of high energy flux which cause melting and evaporation.

Ideally rock should be subjected to pure tension to exploit its low tensile strength. Unfortunately there is no practical way of achieving this. It can only be partially achieved for a continuous breaking process through a combination of methods as described under 1.2.2. For direct methods of continuous breaking large compressive forces are applied in the first place to induce a complex stress field which eventually results in chipping and splitting through induced tensile and shear stresses. Due to the uneveness of the stress field a wide range of fragment sizes are produced and.the large compressive stresses which are required invariably result in crushing of rock under the tool and the generation of dust. One can therefore only attempt to generate a large percentage of the maximum fragment sizes.

Since quartzite rocks are brittle, failure is by crack propagation. The physics of crack propagation have been the subject of intensive investigation (14) (15) (16) (17) and are not yet fully understood. Major cracks mostly curve to a free surface and produce chips. The size of the chips is related to the tool size. If specific forces are large enough a crack develops under the tool in the direction of loading which can split large pieces of rock. Theories to determine the speed of fracture growth have been proposed [18]. Generally the speed is assumed to be 1/3 to 1/2 of the speed of translational waves.

1.2.2 Principal methods.

(i) Mechanical methods.

Rock can be broken by pulling, bending, shearing, cleaving and indentation. Pulling is only possible by first drilling a hole, bending and shearing can only be achieved by first cutting slots and cleaving is achieved by first drilling holes which are then pressurized to cause breakage. The only direct method for continuous breaking is indentation. This can be in the form of roller cutting, drag bit cutting and impacting. In all these cases, the tool forces are in a direction different to the direction of motion.

The tool forces can be applied quazi-statically as in drag bit cutting or vibrating as in activated roller cutting or intermittent as in impacting. The forces can attack the rock at single points as in drag bit cutting or in multiple points as in sawing or diamond core drilling. Basic tool dimensions like the blade width determine the maximum force which can be transmitted to prevent tool damage for given materials and to maintain acceptable wear rates. Since the maximum fragment size of rock generated by a tool is a function of the major tool dimension, the specific energy requirements for a wider tool are lower (see Figure 7), provided sufficient energy is delivered to the tool to generate large fragments. Therefore better performance and lower energy consumption can normally be achieved with big and powerful devices.

(ii) Fluid dynamic methods.

High pressure water jets can excavate rock by erosion, cavitation and impact. If abrasive material is entrained in the jet, they can cause abrasion. Specific energy requirements for all these methods are high because of poor energy transmission and very small fragments.

Fluid dynamic pulses in drilled holes can be used to cleave rock.

(iii) Thermal methods.

Thermal methods can be used to melt, evaporate, spall or split rock. Melting can be used under high heat flux conditions in rocks which have low inclination for spalling. Spalling is the more efficient method as it operates at lower temperatures and produces big fragments. Spalling is induced by differential thermal expansion from external heat sources. Quartz is particularly suitable because of the to matrix transition at $573^{\circ C}$ which is accompanied by an expansion of 0,8%.

Methods which work with external heat sources are sensitive to the efficiency of heat transfer to the rock. To enable the method to be continuous the released fragments must be removed. If no gas burners are used auxiliary gas jets, water jets or mechanical devices would be required.

Large scale splitting can be generated by applying heat in drilled holes or by generating heat inside the rock through the absorption of electromagnetic energy. In this case substantial mechanical forces are still required to dislodge the rock.

(iv) Chemical methods.

Quartz can be dissolved chemically. But there are insurmountable practical difficulties in using it as a continuous mining method. The process is extremely slow and would result in face advance rates of only a few millimeters per month if only the advancing face was in contact with the fluid. The process is also extremely dangerous, requiring extradordinary precautions even under laboratory conditions.

Chemical leaching requires homogeneous fragmentation to a sufficiently small degree for adequate gold recovery. There is no practical way to achieve this, particularly since open fractures would be required for fluids to reach all particles. This would necessitate an expansion of the in situ rock into cavities.

For the above reason no work has been conducted using chemical methods, and it is unlikely that it will be considered in the near future.

1.2.3 Combination of methods.

A large variety of combinations of different methods are possible. Some can be used simultaneously like water jets with mechanical cutters, but most others can only be used sequentially.

The reason for using combinations can be improved performance, reduced costs, or both. Combinations also allow methods to be used which would be unsuitable on their own. Typical examples are:

(i) Drilling of holes followed by pressurization of that hole to achieve tensile and shear breakage to a free surface.

(ii) Cutting of slots to generate additional free faces which can significantly reduce the specific energy requirements for subsequent breaking methods. In an extreme case a block of rock could be removed by cutting it out completely. Because of the small volume removed with slotting, high specific energies can be tolerated.

(iii) Thermal fracturing followed by other methods.

(iv) Impact ripping using auxiliary devices when hard patches are encountered.

1.3 Performance criteria

For a rockbreaking method to be considered as the basis of a practical mining system, it must have the potential, at least, to break rock at a rate that can allow a mining rate to be achieved that is comparable to the industry's present performance, averaging 6 m per month face advance on single shift operation.

A further requirement is that this face advance must be achieved without resorting to a high number of breaking mechanisms per face length. Firstly, this is to avoid detrimental effects on labour productivity and, secondly, to avoid an unnecessary density of expensive strike gullies, assuming that interference problems make it impractical to operate more than one machine per panel.

1.4 Environmental compatibility

1.4.1 Dimensional limitations

The stoping width used in conventional gold mining of approximately 1 m must not be exceeded, as the economic penalties would be considerable. At the same time economic benefits would be derived if the stoping width could be reduced in those cases where the reef band is narrower than the existing stoping width.

A further limitation is the free space available between the face and the first row of supports. Although 2,5 m distance is allowed by law, support requirements may dictate a reduced distance or even temporary support along the face.

1.4.2 System requirements

Usable rockbreaking methods must be compatible with other essential mining functions, such as the clearing of rock from the working area and the conveying of the rock out of the panel into the gully.

1.4.3 Pollution

1.4.3.1 Waste heat.

All rockbreaking methods are extremely inefficient. Typically less than 1% of the supplied output energy is used to create new fracture surfaces. The rest is converted to heat. Therefore substantial amounts of heat can be generated by methods which require high specific energies. This can be further aggrevated if the in stope energy conversion in the rockbreaking device is inefficient.

If this heat generation is close to or exceeds the already high geothermal heat flow into the stope, then costly additional cooling would be required. For instance slotting with abrasive water jets produces heat in the order of geothermal heat flow (19) and any flame piercing methods generate waste heat far in excess of this figure.

1.4.3.2 Noxious fumes.

The existing ventilation systems in mines does not permit the generation of any noxious fumes while the work force is underground.

1.5 Economic considerations

The economics of a method is mainly a function of performance and operating cost. Obviously the running costs of a rockbreaking method can only be allowed to be a relatively small percentage of present stoping costs to achieve a cost advantage for the total system. The only exception which can be made is when the rockbreaking method permits additional savings to be made when compared to conventional mining. One such case would be a method with a very low stoping width, which would require the workers to be in the gully rather than in the stope. In this case the criteria applies that total mining costs must not be higher than present total mining costs or in.any case must not be higher than the value of the gold which is recovered.

In connection with cost considerations one must also address those cost penalties which are a result of multistep sequential breaking methods. Firstly, each separate operation is likely to require a worker, which reduces labour productivity. Secondly, system availability is likely to be effected. Thirdly, sequential methods suffer from end effect problems under the given operating conditions. Therefore additional devices may be required which further reduce availability and productivity.

Experience has shown that system simplicity is of very high importance for the achievement of good mining rates.

2. DESCRIPTION OF METHODS SUITABLE FOR QUARTZITE

2.1 Methods to break rock from a drilled hole to a free surface which is parallel to the hole.

Methods based on this principle were investigated partly as possible direct alternatives to the use of conventional chemical explosives (1963 to 1976), and partly to assist other non-explosive methods of mining when unfractured rock is encountered (1984 to now).

2.1.1 Operating principles

The attractiveness of this method lies in its potential for low specific energy, because it can break rock in tension. Since this is a two step method, the energy for drilling the hole must be added to the energy for breaking. The total energy for drilling a hole of 42 mm diameter and 1 m length in quartzite is approximately 0,4 MJ, which is higher than the energy requirement of the subsequent breaking methods. This drilling energy can further increase and the drilling performance decrease if a bigger than standard hole is required for the breaking method. However, drilling is a well established and simple technology with adequate performance under normal conditions. Performance can be increased if required by using multiple drills and drill rigs.

Practical problems can arise when the breaking method requires the insertion of a tube into the straight as a result of rock conditions and their shape is spiral triangular, when drilled with the customary chisel bit. This can, however, be improved with modern technology like drill rigs and hydraulic rockdrills with button bits.

The effectiveness of breaking from holes can be improved if stress raisers are provided in the hole in the form of longitudinal notches as shown in Figure 8 [20]. Fairly shallow notches can reduce the threshhold pressure for crack initiation and can induce fracturing in the preferred direction.

Another effective method is circumferential notching at the bottom of a hole. The major drawback is that these notches can presently only be generated with high pressure water jets. This would increase the required hardware by another and fairly sophisticated piece of equipment.

Rock fracture conditions have an obvious influence on breaking performance. Although fractured rock will generally require lower energy input, some methods can actually be less effective in fractured rock, as it permits the dissipation of the supplied energy through open cracks. Therefore a clear understanding of the intended application of the method is essential. For instance, it may be required for the full range of encountered fracture conditions or for unfractured rock only.

Three important parameters have been identified which critically influence rockbreaking performance

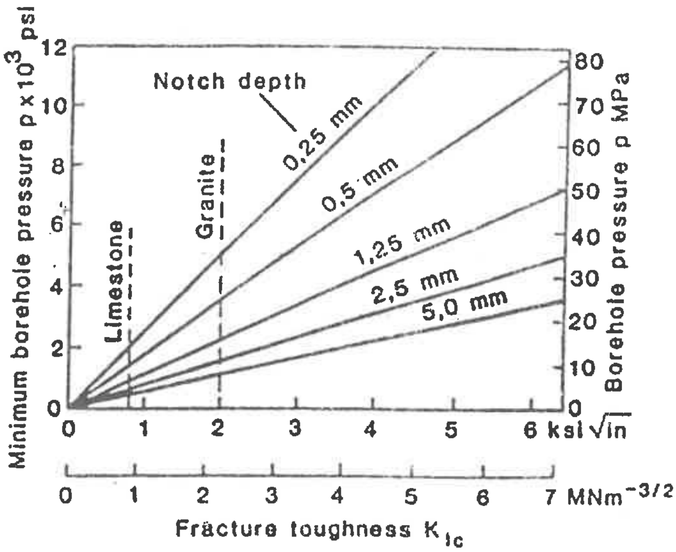


Figure 8. THRESHOLD PRESSURE FOR CRACK INITIATION FOR VARIOUS NOTCH DEPTHS (Ø = 50 mm)

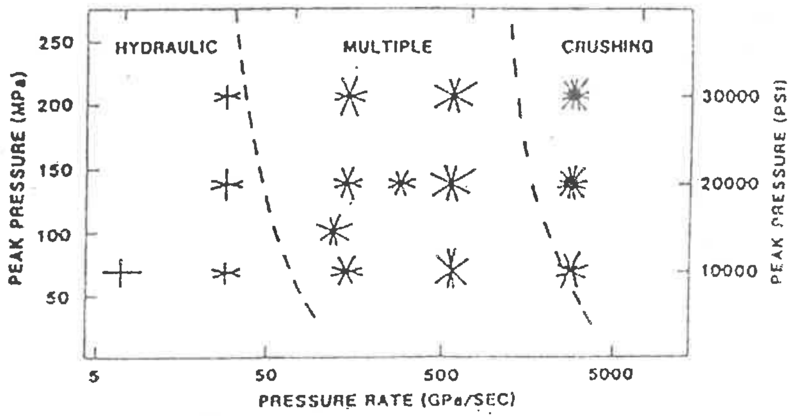


Figure 9. FRACTURE INTENSITY AS A FUNCTION OF PEAK PRESSURE AND LOADING RATE

[21]. There is firstly the peak pressure which must at least exceed the tensile strength around the hole. If the peak pressure exceeds the compressive strength of the rock, crushing will be induced around the hole as a first step in the disintegration process. This is not desirable as it produces dust and blocks the driving medium from entering the developing cracks, and assisting in the disintegration process. Non-explosive methods do not generally reach pressures which crush the rock, particularly since the underground quartzite is still subjected to some overburden stress, which increases its compressive strength [22]. As mentioned before, fracture initiation pressure can be reduced by notching.

The second important parameter is the pressurization time. For quasistatic pressurization rates typically two fractures develop, which does not necessarily mean, that the rock will be dislodged. If pressurization rates are increased more fractures will develop because of inertia effects in the rock mass. It has also been shown [23] that short pressurization rates have an advantage over static pressurization because of up to 60% higher hoop stresses around the hole for the same driving pressures. This happens when pressurization rates are fast in relation to the translational wave velocity in the rock. The effect of the first two parameters is illustrated in Figure 9 [21]. The additional influence of hole diameter is shown in Figure 10 [22].

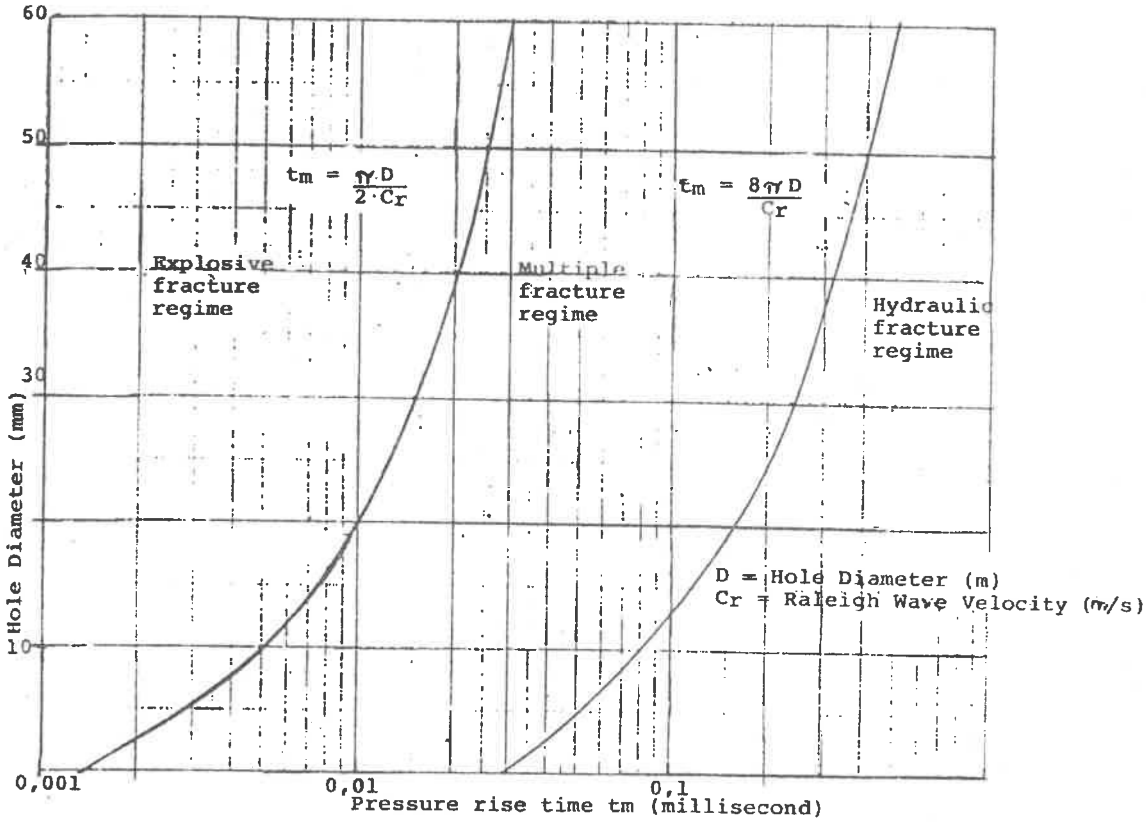


Figure 10. FRACTURE INTENSITY FOR LOCAL QUARTZITE AS A FUNCTION OF HOLE DIAMETER AND PRESSURE RISE TIME ACCORDING TO THE MODEL PROPOSED BY CUDERMAN

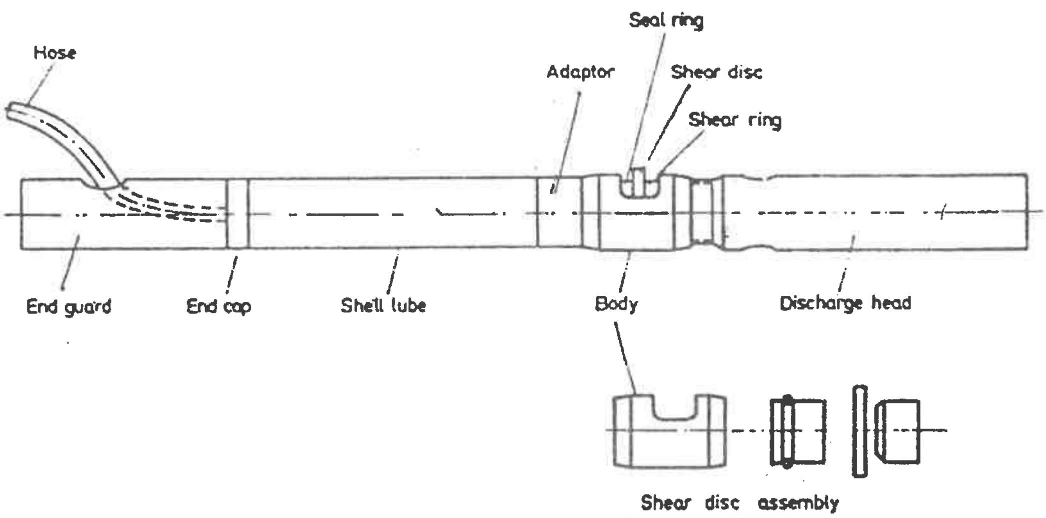


Figure 11. THE ARMSTRONG AIRBREAKER SHELL

The third important parameter is the length of the pulse, which is relevant for the disintegration of rock, since cracks propagate with speeds lower than those of translational waves. Suitable ranges of values for the three critical paramenters have been calculated for quartzite [22] and multiple fracturing. From that a peak pressure from 75 MPa to 500 MPa, a rise time from 0,25 ms to 0,02 ms and a pulse length from approximately 0,05 ms to 0,1 ms appear desirable.

Before the results of the various rockbreaking devices which were tested in South African gold mines are discussed, a note must be made on the difficulties of comparing the recorded results. This can be explained with the large and unpredictable variation of the fracture density in the rock face, which has only been determined recently by mining extensive areas without the use of explosives and under constant operating conditions. Since the exact degree of fracturing was not known for the recorded tests, comparisons between different methods and often also different tests for the same method cannot be made unless large differences were recorded.

2.1.2 Rockbreaking devices based on the discharge of high pressure gas.

(i) Armstrong Airbreaker

Between 1963 and 1965 extensive tests were carried out on this device which consisted of a tube with an outside diameter from 38 to 57 mm and a length from 1,5 m to 3,6 m as shown in Figure 11. Sometimes an accumulator was added at the supply end of the tube to increase its capacity. At the discharge end the tube was sealed off with a rupture disc which would break at pressures from 68 to 103 MPa. Stored energy was up to 0.23 KJ/cm^3 which means between 150-850 KJ depending on the dimensions used [24]. The tube was inserted into holes, which were approximately 6 mm larger than the outside diameter of the tube. Compressed air was supplied from a multi-stage compressor via a hose. Peak pressures in the hole varied from 83% to 47% of rupture disc pressures. The higher percentages were achieved with the larger diameter tubes.

This device was tested as a mining method to replace blasting. Therefore constant burdens were used and multiple blasts were released or relieving holes were drilled if the hole would not break. In different locations burdens of 150 mm, 200 mm and 300 mm were broken as long as the rock was reasonably fractured. Great difficulties were encountered if the rock was less fractured. In one case out of 260 holes drilled only 72% were straight enough to insert the tube. Tests with smaller diameter tubes to save on drilling costs were less effective compared to the larger diameter tube, even if the charge pressure was increased to the maximum the device was capable of. The final conclusion was that the Armstrong Airbreaker was not powerful enough to break an acceptable burden under all fracture conditions which, combined with the operating difficulties, did not permit economic application.

(ii) Cardox

This commercially available device, shown in Figure 12, was experimented with in 1965. Its advantage is that the energy was contained in a cartridge, which only had to be inserted into the firing tube. The energy in the form of compressed CO_2 is released when the cartridge is ruptured by a small electrically ignited detonator. The pressure in the tube is then released into the hole after breakage of a rupture disc. The tube is held in the hole with a mechanical wedging device. Peak operating pressure according to the manufacture was 260 MPa with a stored energy of 250 KJ. Tubes with 54 mm outside diameter and 950 mm and 1250 mm length were tested.

Ignition reliability was poor and tubes were ejected from the hole despite the anchoring device if the holes were not broken. Breakage was also poor. Burdens of 150 m could not be broken consistently [25].

(iii) Shockwave or Combustion Breaker

This device, shown in Figure 13, was tested between 1967 and 1971. The applied combustion process in the tube was considered a more economical process than the previously investigated method to achieve high driving pressures and high energy

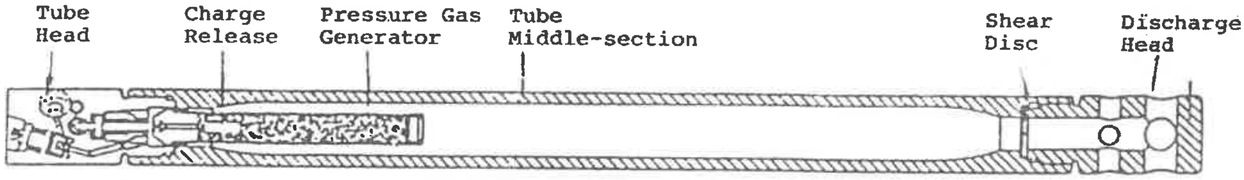


Figure 12. <u>CARDOX GAS BREAKER</u>

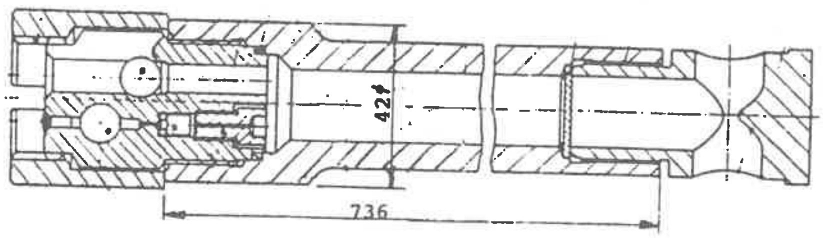


Figure 13. <u>SECTION OF SHOCK-WAVE BREAKER</u>

concentrations. The device consisted of a tube of 42 mm outside diameter and 850 mm length, which was inserted into a hole of 45 mm diameter. The tube was charged with a 10% acetylene in air mixture at 20 MPa. On initiation a shock wave was produced with pressures up to 1500 MPa and rise times of 10 ms, which would cause peak hole pressures of up to 500 MPa. The stored energy was 0,55 KJ/cm^3 or 100 KJ [25].

This device was extensively researched [23], which resulted in several important findings. The high energy storage was largely a result of the high temperatures generated during the combustion and the high temperature also allowed a 7 times higher escape speed from the tube than would be possible at ambient temperatures. At the same time it would allow faster gas penetration into the developing cracks. Speeds of 500 m/s were determined in 0,25 mm wide simulated cracks. This could be exploited since crushing of the hole was avoided. Suitable steels for the tubes and methods of manufacture were identified. Unfortunately, comparative testing was very difficult and even dangerous because of inconsistencies in the combustion process. These difficulties could not be satisfactorily resolved, so that further work was eventually discontinued in favour of the simpler and more reliable mechanical methods.

2.1.3 Rockbreaking devices based on fluid pressures

Fluid driven methods have the potential for requiring less energy than gas driven methods since they do not operate with large volumetric expansions, which in some applications has the additional benefit of reduced fly rock.

Rock can be broken either with static pressure or with a dynamic pulse of short duration. Static pressure tests were conducted in 1966, whereas dynamic pulse methods are a recent development which only begun in 1985.

Dynamic methods can operate in a variety of ways. They can pressurise a water filled hole by releasing a pulse from the bottom of an inserted tube or by generating a pulse at the entry of the hole. They can also inject a high speed fluid jet into a dry hole and produce a pulse on impact at the bottom of the hole. Pulses can be generated by impact, by explosives, by discharge of electric energy in the fluid or by discharge from accumulators.

The entering of fluids into developing cracks has been investigated by several researchers [26] [27]. Important parameters are crack width, surface tension of the fluid, pressure and flow resistance. No quantitative data is as yet available to indicate whether this assists in the propagation of cracks and thertefore the effectiveness of breaking.

2.1.3.1 Static methods

Static pressurization has the potential to generate sufficient pressure for tensile breakage of rock.

It requires, however, adequate sealing in the holes.

(i) Expanding Rubber Tube

This was a commercially available device tested in 1966. It consisted of a rubber sausage of heavily reinforced material [(28)]. It was inserted into a 45 mm hole and then pressurized. Due to the heavy reinforcement, which was necessary to prevent bursting after the hole had been broken, the full internal pressure could not be transmitted to the rock. The major operational problem with this device was that rock would not break evenly, so that the device would become partially exposed. This would concentrate all further expansion in the exposed section and not break the remaining rock.

(ii) Plunger Buster

This device was also investigated in 1966. It consisted of a tube with a multitude of radial plungers [(28)]. It was designed for a hole diameter of 50 mm. Due to space limitations forces were limited. The retaining of the plungers at the end of their stroke was inadequate and the stroke of the plungers was insufficient to dislodge the broken rock from the face, therefore this concept was abandoned.

2.1.3.2 Dynamic methods

Tests have been conducted by the Chamber since 1985 with water guns driven by propellant charges with the purpose of developing a method for the breaking of hard patches. Therefore unfractured rock was required and Norite was chosen for being readily accessible and adequately homogeneous. To correlate breaking behaviour with the shape of the pressure pulse, a test cell was constructed to record the pulse in a simulated hole. This eventually should lead to a specification for a pulse generator based on an accumulator discharge from hydro-power instead of using propellants.

(i) Flowex

This device consists of an accumulator of 6 litres water capacity which is connected with a NW 38 hose via a guick release valve to a blast tube with a bore of 11 mm. This tube is inserted to a length of 200 mm into a 40 mm diameter water-filled hole, where it is mechanically sealed and locked. The pulse from the accumulator, which is charged to 30 MPa, is released through two holes in the wall of the tube. The details of the tube are shown in Figure 14.

The results in Norite were that a burden of 100 mm could not be broken. The peak pressure was recorded in the test cell at 35 MPa with a rise times of 12 ms and a pressurization rate of 4 GPa/s. The pulse length was infinite because of the sealing in the hole.

The poor breaking results are confirmed by the manufacturers recommendations which are to use the

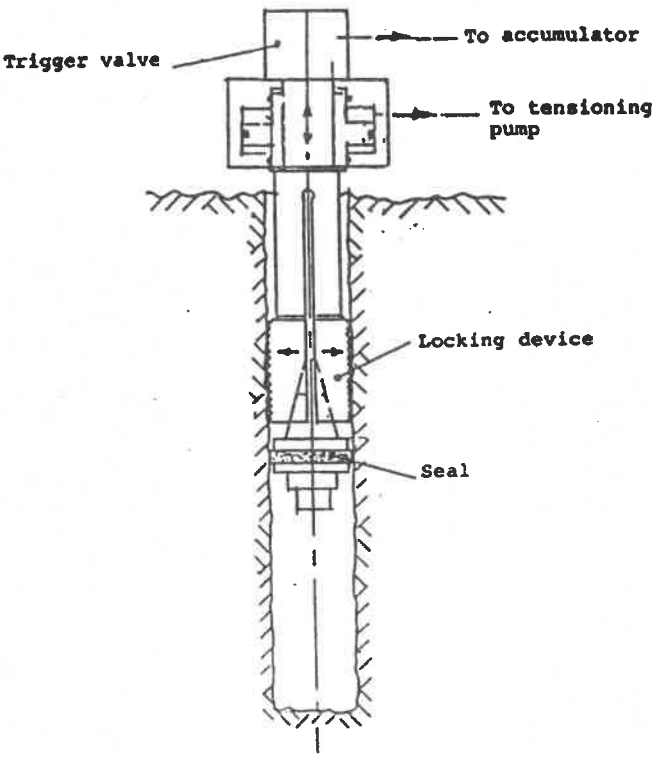


Figure 14. FLOWEX

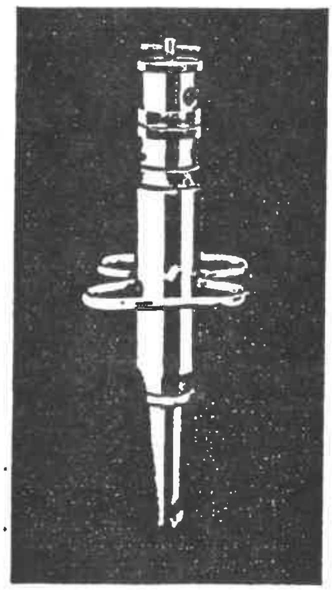


Figure 15. BOULDER BUSTER

device only with longitudinal and circumferential notching. This is unattractive because of the time required and the complexity of the equipment.

(ii) Boulder Buster

This device consists of a body which contains a shotgun cartridge with a charge of 10 g Ballastite with an energy content of 40 KJ. The discharge end has a conical section, which is inserted into a water-filled hole of 40 mm diameter. On insertion water rises into the body to the level of the cart-ridge. Firing is mechanical by a pull wire. On firing water emanates from two holes in the side of the conical insertion section. The device is shown in Figure 15.

The results in Norite were that it could not break a burden of 100 mm for a 400 mm deep hole and it is therefore inadequate.

(iii) Essig Rockbreaker

This device, shown in Figure 16, is similar to the Boulder Buster and uses the same charge. But it has to be inserted fully into the hole, where it is locked with a plastic sleeve. Water is poured in after installation.

This device was almost impossible to remove, if the rock was not broken. The breaking ability was as low as that of the Boulder Buster and because of the operating difficulties this device was less suitable.

(iv) CSIR Water Gun

The principle of this device is shown in Figure 17. Essentially the cartridge of 9 g Ballastite of 36 KJ energy is combined with a 50 ml water slug.

Only a few tests were conducted with the gun, which showed that the energy was too low for effective rockbreaking. Therefore a new gun was designed by the Research Organization to work on a similar principle, but with considerably increased energy.

(v) Chamber of Mines Water Gun

Based on the experience with the CSIR gun, a new gun which was adapted for standard size holes was designed as shown in Figure 18. 250 ml of water is contained in a rubber sleeve and the front end of the gun is detachable to allow different nozzles to be used. The charge is 45 g of Ballastite with an energy of 250 KJ.

Four different tests were conducted in Norite.

a) Water-filled hole: 250 mm burden broken.
b) Dry hole: 100 mm burden not broken.
c) Gun and hole water-filled: 150 mm burden broken.
d) Gun and hole water-filled, cartridge inserted without a piston: 100 mm burden not broken.

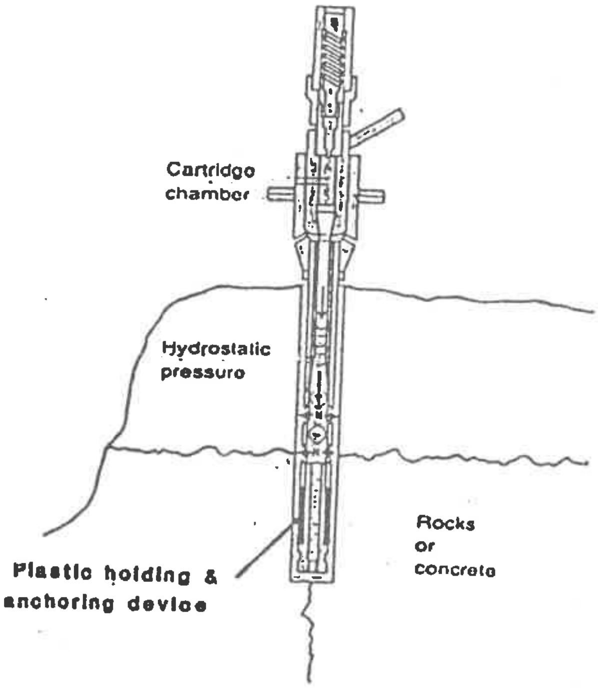


Figure 16. ESSIG ROCK BREAKER

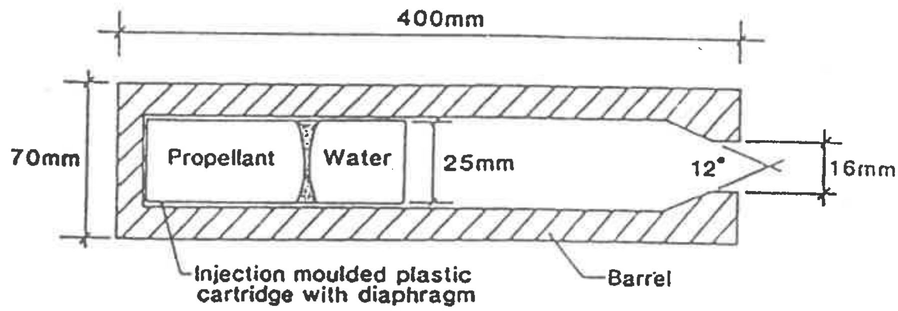


Figure 17. THE CSIR WATER CANNON

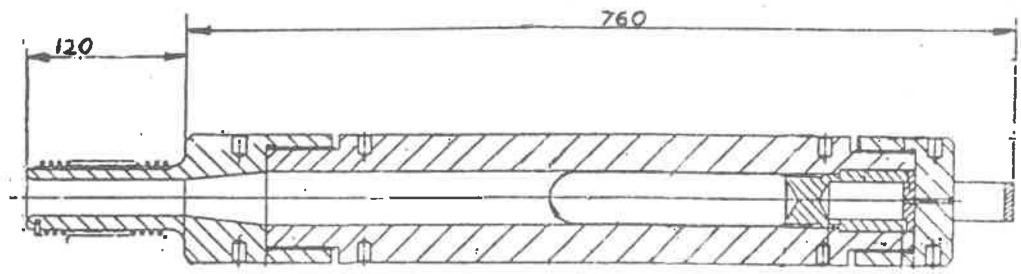


Figure 18. CHAMBER OF MINES WATER GUN

These large differences still have to be explained. Only for the case of a dry hole is some information available [29]. According to this, the normally to be expected jet stagnation pressure as per Figure 19 is not achieved if the jet is injected into a deep hole as shown in Figure 20. Therefore a further possible test variation is a deep hole with the jet nozzle extending to the bottom of the hole.

Pulse measurements still have to be conducted.

(vi) Hydrex Device from Flow Industries

This prototype device consists of a high pressure accumulator with a discharge nozzle, as shown in Figure 21, which is inserted to the full length of the hole. When the accumulator is pressurized to 400 MPa, by which time 60 KJ of energy will have been stored through the compression of the water, the rupture disc at the discharge end of the tube will release the pulse into the water-filled hole.

Measurements conducted by Flow Industries in a test cell have given peak hole pressures of 60-70% of the burst pressure of the rupture disc [30]. A typical pulse shape is given in Figure 22, giving a rise times of 1,5 ms, an average pressurization rate of 190 GPa/s and an extended pulse duration. The rise times changed considerably from 0,5 ms to 1,5 ms during the few tests. The pulse shape was such that

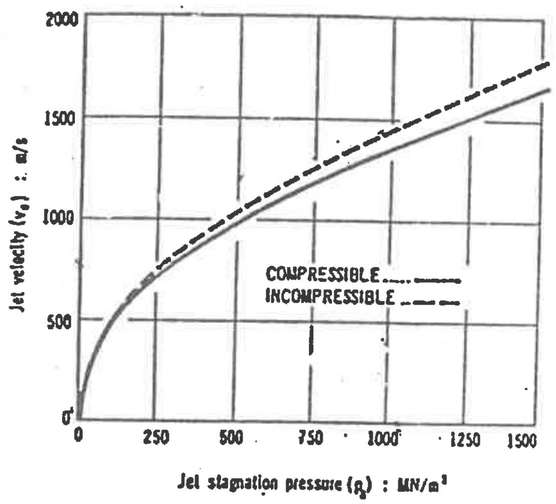
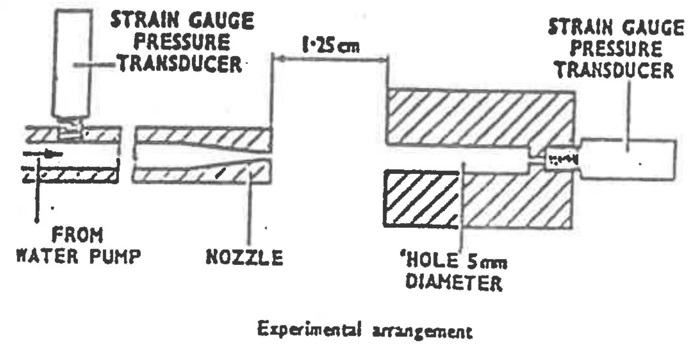


Figure 19. THE THEORETICAL VARIATION OF JET VELOCITY WITH STAGNATION PRESSURE



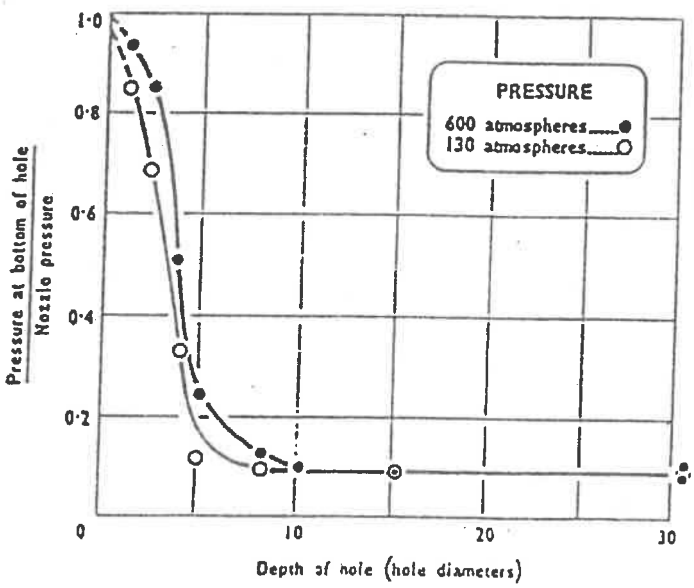


Figure 20. PRESSURE AT THE BOTTOM OF A SIMULATED HOLE (AS PER EXPERIMENTAL ARRANGEMENT)

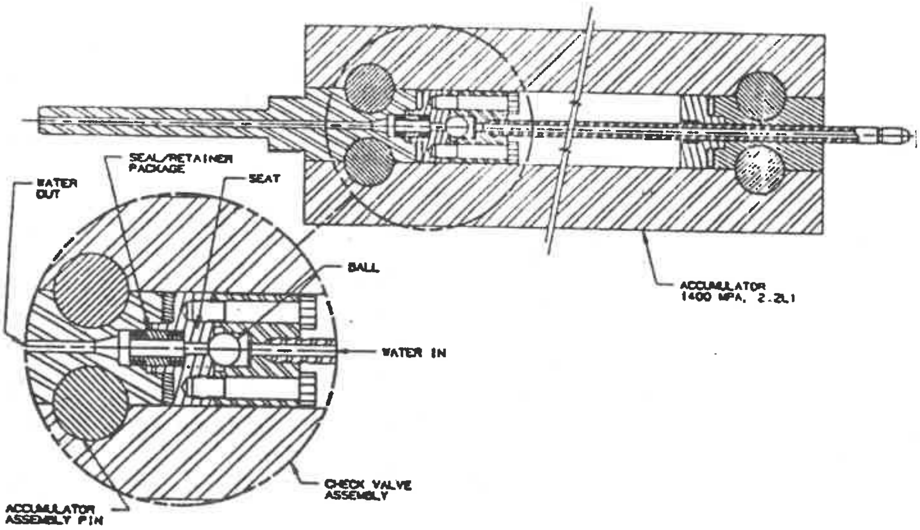


Figure 21 Design for Fast-Opening Discharge Valve

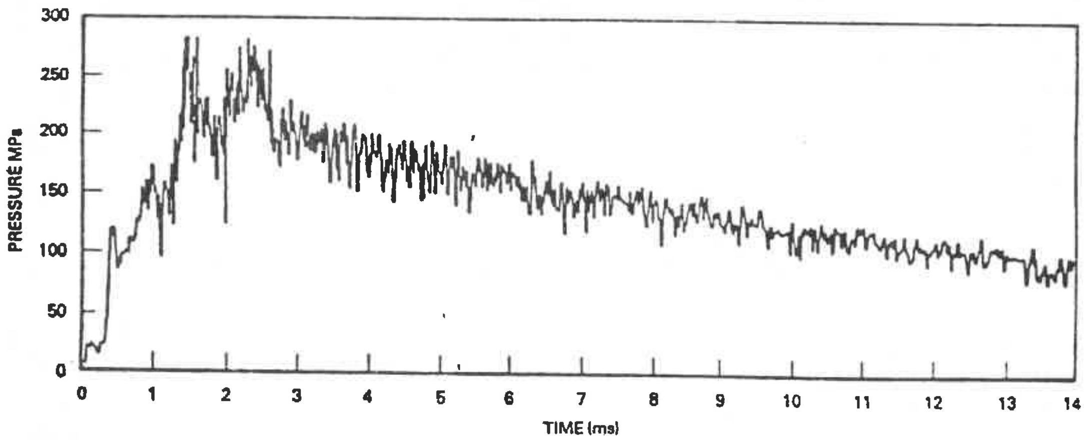


Figure 22 HYDREX Pressure Curve upon Discharge into 26,2 mm Diameter Test Chamber

it caused crushing in the bottom of the hole in the rockbreaking experiments. It is intended to obtain this device for experimentation.

(vii) Water Cannon from Briggs Technology

This cannon, shown in Figure 23 [31], consists of an accumulator charged to 170 MPa. The accumulator has a quick release valve, which allows rapid firing at 5Hz. Water discharge is through a cumulation nozzle, which results in narrow supersonic water jets. The benefits of cumulation nozzles in connection with breaking from drilled holes has still to be established. Tests have been planned to investigate this.

(viii) Mechanical Impact on Water

This method could potentially be applied by using the impact hammer on a mining machine to provide the impact energy. The principle was tested by impacting a drop weight onto a piston, which was inserted into the water-filled test cell, produced a pulse with a peak pressure of 180 MPa, a rise time of 7 ms, a pressurization rate of 35 GPa/s and a pulse decay time of 7 ms. This applied for an input pulse of 2,3 KJ. This method would be amenable to repeated pulsing. More tests need to be conducted to establish the suitability of the method.

(ix) Electrohydrodynamic Method

With this method electric energy is stored in

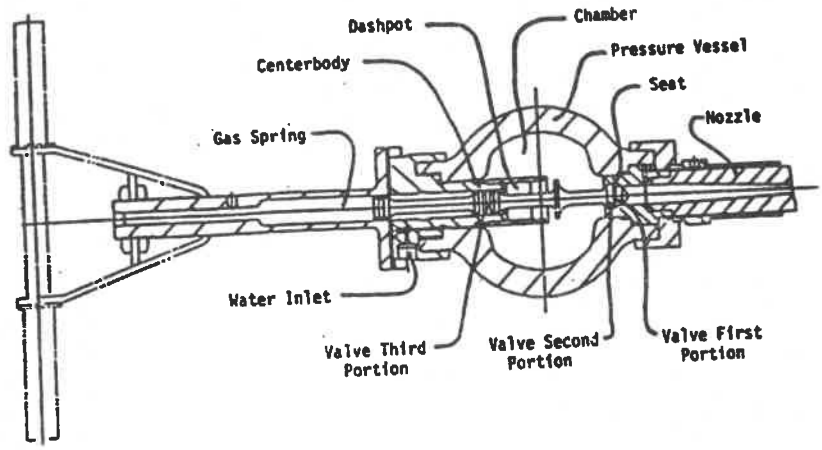
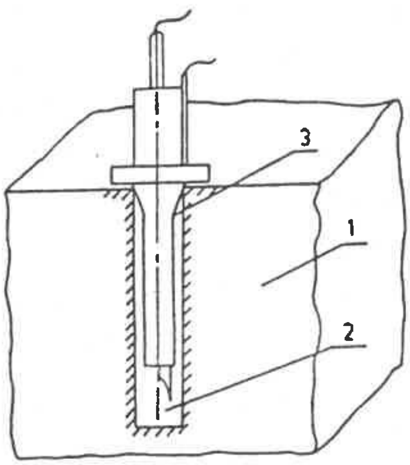


Figure 23. BLOWDOWN WATER CANNON



- Electrohydrodynamic transducer placed in hard rock
  1. block of hard rock
  2. hole drilled in the block and filled with water
  3. electrohydrodynamic transducer

Figure 24. ELECTROHYDRODYNAMIC ROCKBREAKING METHOD

capacitors at a voltage of several KV. This energy is discharged between two electrodes inserted into a water-filled hole, as shown in Figure 24 [32].

The pulse is claimed to reach a peak pressure in excess of 1000 MPa [33] at a pressurization rate in excess of 200 000 GPa/s. Generators up to 80 KJ have been built [34]. Total pulse duration is only 0,01 ms. Pulses have been fired up to a frequency of 5 Hz to compensate for the short duration of the pulses. More information is at the moment being gathered on this method to assess the merit of testing by the Chamber of Mines.

2.1.4 Expanding solids

This is a static method because of the long time involved. Special cement mixtures have been developed which generate tensile stresses up to 45 MPa when expanding in a hole while in the process of hydration and crystallization. Application potential exists for the prebreaking of hard patches, which was investigated in 1987.

The result in Norite was a pattern of open cracks after a period of 24 hours. Because of confinement on two sides, the rock could not be readily dislodged. It may nevertheless still be suitable as a prefracturing method in unfractured rock, but only if setting times can be significantly reduced. This appears to have happened with a new product which sets after only 15 minutes, as shown in Figure 25. Further tests will be conducted underground.

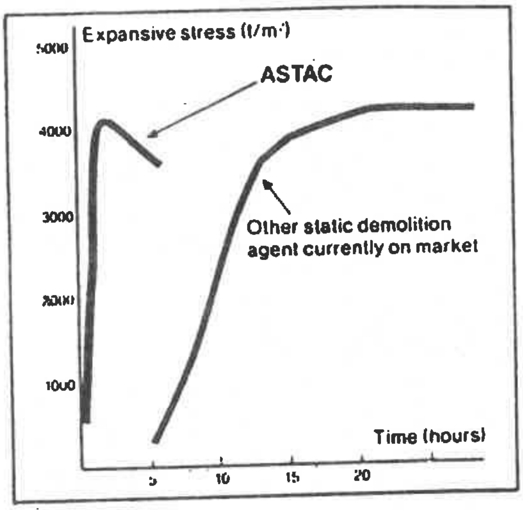


Figure 25. SETTING BEHAVIOUR OF EXPANDING CEMENT

Figure 26. ARMSTRONG ROCKBREAKER IN OPERATION

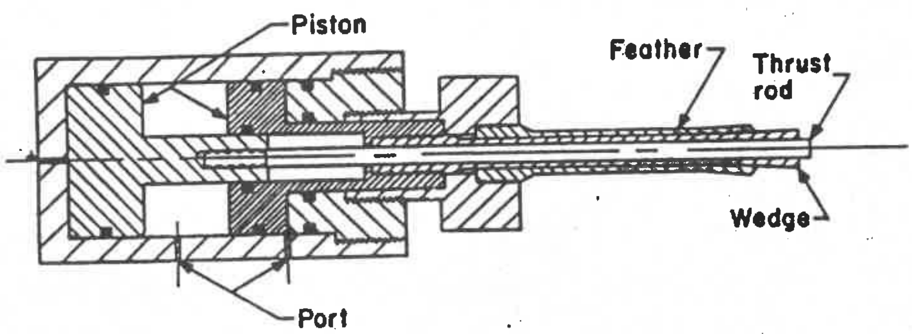


Figure 27. SECTION THROUGH AXIAL - RADIAL ROCKSPLITTER

2.1.5 Mechanical methods

Mechanical methods can operate with static force generation in the hole or with intermittent pulsed force generation.

2.1.5.1 Static forces

(i) Darda Rocksplitter

This device tested from 1970 to 1972 to assist in the breaking of unfractured rock consists of a feather and wedge assembly, which is inserted into the hole and tensioned by a built on hydraulic cylinder, as shown in Figure 26 [35]. Operating pressure is up to 50 MPa, which results in a radial splitting force of 1 MN for an assumed coefficient of friction of 0,1. The expansion range is either from 40-50 mm or from 43-48 mm.

The breaking effect was adequate but many practical problems eventually terminated its further use. These were: A straight hole was required to avoid bending forces in the feather and wedge assembly, which would cause damage. A fairly round hole was required, as otherwise rock would first have to be crushed until forces were evenly distributed. This caused high forces and wear on the outer parts and used up the useful expansion range. The feather and wedge assembly had to be greased which caused heavy wear with the unavoidable quartzite dust. Mechanical failures occurred when the rock was not broken and the device had to be retracted. For all these reasons the life of the unit was short and operating costs were high. It was therefore superseded by the development of bull wedges.

(ii) Radial-Axial Splitter

This device developed and tested in 1986 by the U S Bureau of Mines, is similar to the Darda Rock Splitter but incorporates an additional function, which allows it to also induce a co-axial pulling force on the rock by reacting against the bottom of the hole [36]. This enabled the device to pull a piece of rock off a rock face without the need of an additional free face to break to. The device is shown in Figure 27 and the basic action in Figure 28. The reasons, which lead to the rejection of the Darda Rock Splitter should even more apply to this device. Also the benefits of the pulling action in the corners of a narrowly confined stope is uncertain. It has therefore been decided to await further developments and representative operational trials by the U S Bureau of Mines.

2.1.5.2 Dynamic forces

(i) Bull Wedges

Wedge rockbreaking was originally tried as a mining method on its own [37]. The tests were conducted from 1963 to 1969 by Anglo Transvaal Consolidated Investment Co. A conical tool of 12° angle was used at an angle of 6° to the hole axis. The tool was

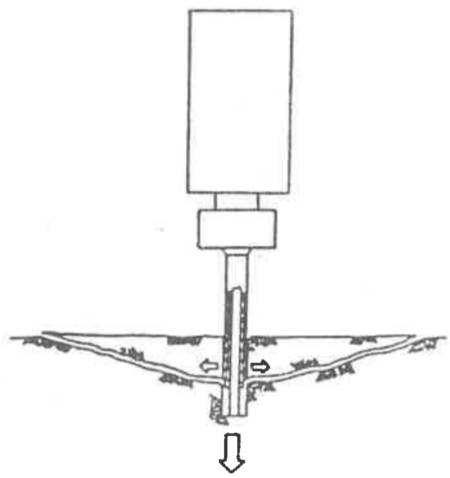


Figure 28. OPERATION OF AXIAL-RADIAL SPLITTER

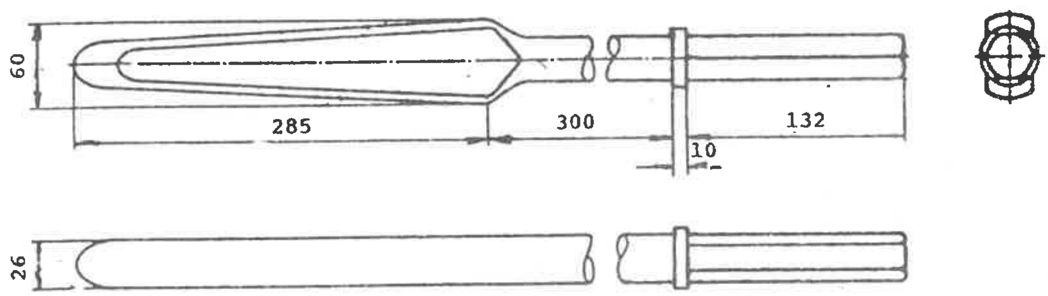


Figure 29. BULL WEDGE FOR 40 mm DIAMETER HOLE

first pushed into the hole with a cylinder until a preset force was reached, after which a pile driver hammer would start impacting. The hammer had a blow energy of 2100 J and cylinder forces varied between 130 to 250 KN. Under these conditions burdens of 300 mm could be broken. Feed rates dropped to 40 mm/min in unfractured rock. The presence of parting planes improved breaking rates. In one set of tests a force of 900 KN was determined to break holes without impacting, however, reaction forces of this magnitutde were very difficult to absorb. Wedge breaking never advanced from the experimental stage to a prototype mining system.

Bull wedges were investigated by the Chamber of Mines in 1972 as an ancilliary device to break unfractured rock which could not be removed with hand-held paving breakers during the rock cutter trials. They were originally commercially supplied in a conical shape for use in a paving breaker for breaking from predrilled holes. Tests conducted by the Chamber determined that a two sided wedge action was far more effective. Experiments with wedge angles arrived at a compromise of 5,5° included angle. Bigger angles caused too much spalling near the hole entry and narrower angles reduced the range of hole diameters for which it could be used. The final shape is shown in Figure 29 [35].

The operational experience was as follows: Bigger hole diameters were more effective. Straight holes were beneficial to prevent uneven contact and to prevent jamming. Increased blow energy significantly improved performance. 70 J blow energy was considered marginal and above 100 J very satisfactory. Wedges would free themselves if they contacted the bottom of the hole. It was preferable to impact wedges only until they stopped moving and then to rather impact the neighbouring wedge. By changing the wedges, which were impacted, better performance could be achieved. Bull wedging was considered slightly less effective than the Darda Rock Splitter, but far more practical in underground operations.

It was recognised that higher blow energies would be required to increase the performance substantially, which would make a hand-held device impossible with the existing technology. Therefore concepts of counterbalance machines operating at higher blow energy but lower frequency were considered or devices which exploited the friction forces in the wedge to absorb the reaction forces. Further work was discontinued when the rock cutter trials were stopped.

Bull Wedging was re-evaluated recently (1987) for hard patch breaking to get a better indication of the effect of increased blow energy and to get a rating against the other methods under investigation. In Norite a burden of 150 mm at a hole distance of 200 mm could be broken with a blow energy of 500 J. The specific energy was 16 MJ/m^3 which was 60% higher than for the Chamber water gun in a water-filled hole. Specific energy increased rapidly if a critical burden was exceeded. For instance a 30% increase of the above burden doubled the specific energy. A comparative test with a 90 J hammer at a burden of 100 mm resulted in a 6 times higher specific energy and required a considerable longer time. It was further noted that bull wedges driven in at the higher blow energy were more difficult to remove and typically 50% of the time was spent impacting the last 25% of the hole. Further tests will be conducted underground.

2.1.6 Thermal methods

Thermal methods could have potential for prebreaking of hard patches. One of the advantages is the simple and available technology. Thermal methods in holes have the benefit compared to surface methods of reducing heat loss to the environment and of producing larger fractures at reduced specific energy.

Tests conducted elsewhere with electric arcs of 10 to 20 kW power in quartzite resulted in specific energies of 25 MJ/m^3 [38]. Environmental effects were reported to be minimal. It is intended to evaluate the fundamentals of the method before a test programme will be defined.

2.2 Secondary breaking from slots

One theoretically promising method of breaking unfractured rock is to create parallel slots in the rock face and to apply bending forces at 90° to the slots to break out the rock.

Laboratory tests were conducted in 1966 to simulate the breaking out of reef which had been slotted horizontally above and below the reef plane.

In that experiment hydraulic hoses were squeezed into slotted rock. A burden of approximately 200 mm could easily be broken in bending by pressurizing the hoses [39].

Although breaking off rock in bending appears promising, the difficulty is in generating the required slots.

2.3 Thermal methods

2.3.1 Thermal methods based on external heat sources.

Quartzite is suitable for spalling from thermal expansion as a full face operation. However, the high specific energy of spalling would require substantial power levels to achieve acceptable mining rates. These power levels could be provided by large gas torches and oxygen burners, but their heat imput into the stope would be approximately 44 times the geothermal heat flow [19] and would therefore be unacceptable. However, these methods could potentially be acceptable if they were restricted to the cutting of narrow slots only.

Tests were conducted in 1973 with a high temperature plasma jet, a commercial device for slot cutting in quarries. During experimentation in quartzite the slot could not be reduced to less than 150 to 200 mm and the depth which was spalled off per pass was only about 1 mm. The cutting rate was therefore too low to be of practical use and the method resulted in a high amount of fumes and too much heat to be environmentally acceptable.

Better suited for the thermal cutting of narrow slots are methods operating with radiation for heat transfer because of their ability for directional concentration of their heat output. Furthermore radiant energy has the advantage over convection heating of greater depth penetration. Tests were conducted on surface in 1973 with a 2½ kW projector lamp combined with a Freznel lense. Although this arrangement did spall off rock, the cutting rate was too low to be of practical use. This can however be explained with the low power output of the then available light source and the low efficiency of the test arrangement. Therefore a re-evaluation with modern technology may be justified.

An attractive method using radiation and high concentration of energy is based on lasers. Lasers can achieve adequate cutting rates in narrow slots [(40)] but they have not been experimented with by the Chamber Research Organization because they are still excessively expensive for high power levels and are bulky in size. Also, the cutting of very narrow slots of only 2 mm may be impractical in the mining environment. However, since local expertise in laser manufacture is improving and power outputs are continually increasing and, furthermore, as lasers can be made with frequencies better suited toefficient absorptions in quartzite, this situation may have to be reviewed.

2.3.2 Thermal Methods based on internal heat generation

A great number of methods exist like electron beams, microwaves, resistance heating, dielectric heating, magnetic fields etc. most of which work only for specific types of rock with suitable physical properties [41]. Only two of these methods are potentially suitable for quartzite.

(i) Electron beams

Electron beams are a surface effect method due to the low penetration of only 1/10 of a mm. This combined with high energy levels results in melting of the rock surface. Practical applications are prevented because of the release of X-rays in the process, which would require excessive shielding.

(ii) Dielectric heating followed by dielectric breakdown and high current discharge.

This method has potential to break certain types of rock including quartzite. It operates by first heating the rock in a high frequency electro-magnetic field between two electrodes, until a conducting path is generated between the electrodes. How this path is generated is not fully understood. This path is then used for a high current discharge which shatters the rock.

This method was researched by Battelle Institute on behalf of the Chamber in 1987. The first phase of the project was to determine if South African quartzite is suitable for dielectric heating. The reported results [8] are that it appears to be marginally suitable, although some anomalies were discovered which could not be explained. This would require further analysis and could still render the method unsuitable.

No information is as yet available if quartzite would develop a conducting path and if so at which voltage and frequency levels and further if the required hardware and methodology would be suitable for underground operation [42]. A proposal for the continuation of the project has been made by Battelle, but has been shelved due to the high costs and low probability of success.

2.4 Fluid dynamic methods

Rock can be excavated by high speed water jets in a variety of ways. Known methods are continuous jets, continuous oscillating or rotating jets, cavitating jets, pulsating jets, supersonic interrupted jets and abrasive jets with high and low pressures [43]. All these methods operate with high specific energy due to the generation of very small particles. They are therefore only suitable for rock slotting, particularly in quartzite which is one of the most difficult rocks for water jet cutting. The fracture process is a combination of several effects such as damage from stress waves caused by impact, erosion, particularly of a soft matrix, enlargement of microcracks and hydraulic wedging of cracks. Of the various methods listed only continuous jets and abrasive jets have been tested in South African quartzite so far.

2.4.1 Continuous jets

Underground tests were conducted in 1973 with a machine supplied by Atlas Copco. It had a pressure intensifier mounted on a fast moving reciprocating transversing mechanism. This arrangement required high staking forces to absorb the acceleration and breaking forces at the ends of the stroke. The generated pressure of 100 MPa was the minimum required to induce damage to quartzite, and therefore cutting rates were unacceptably low. The groove was of very irregular depth with a maximum of 150 mm. In an effort to improve the cutting depth a two-nozzle machine was designed by Atlas Copco to cut 2 groves 20 m apart with the nozzles pointed to the outside corners, so that after the disintegration of the centre web the nozzles could enter the slot to increase its depth. This machine was never supplied for testing.

2.4.2 High pressure abrasive water jets.

High pressure abrasive water jets have been extensively researched [44] and have been tested by the Technical Development Services of Anglo American

(TDS) in underground quartzite [45]. Typical operating parameters are as shown in Table 4. Chromite fines were used as an abrasive because it is readily available in South Africa as a cheap waste product from Chrome mining. Nevertheless the total quantities of abrasive used are considerable. Figures of up to 430 kg of abrasives per ton of rock were quoted. This causes logistic, cost and pollution problems. Costs of R45/m^2 mined for the abrasive only were determined even under favourable conditions. Therefore alternatives have been investigated by TDS, such as non-polluting in situ crushed quartzite. This reduced, however, the performance to 65% as compared to Chromite. Adding steel shot to the quartzite resulted in the same performance as Chromite. Steel shot has the advantage of being easy to separate.

TDS report that the performance underground in quartzite was 1,2 m^2/h in rock with a low fracture intensity. Methods of mining investigated were block cutting as shown in Figure 30, which was however considered uneconomic in terms of cutting time and costs. Alternatively methods of cutting one slot in the hangingwall and footwall each were investigated, as per Figure 31. This required a secondary breaking method, based on the drilling of holes, which were subsequently disced and notched and broken with a Flowex device. Sometimes this would only result in cracks and not in the dis-lodging of the rock so that a further operation

Table 4. TYPICAL OPERATING PARAMETERS OF THE TDS ABRASIVE WATER JET CUTTING SYSTEM

WATER - Pressure	240 MPa
- Flow Rate	13 l/min
- Orifice diameter	0,81mm
- Focus tube I.D.	2,4mm (discard at 3,5mm)
ABRASIVE - Type	Chromite
- Size	0,20 - 0,85mm
- Flow Rate	3,2 kg/min
NOZZLE TRAVERSE RATE	100mm/min
DEPTH OF CUT	250mm

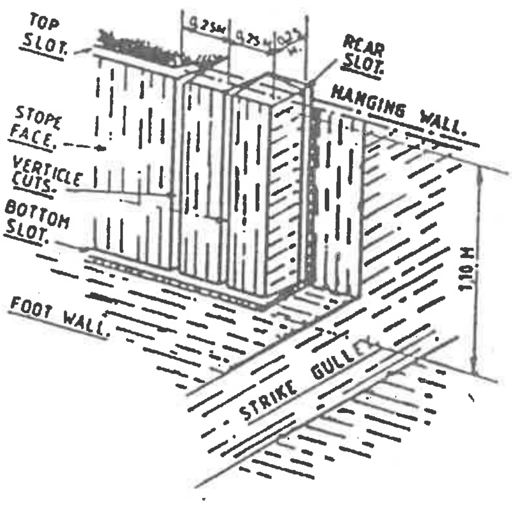


Figure 30. THE CONFIGURATION OF SLOTS IN A STOPE FACE FOR THE BLOCK CUTTING MINING METHOD

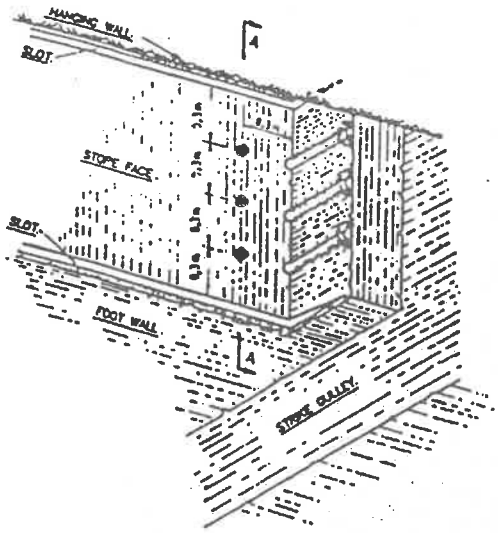


Figure 31. GENERAL ARRANGEMENT FOR STOPING USING SLOTS AND WATER PULSE BREAKER

using paving breakers had to be introduced. It is significant to note that the secondary breaking method could not keep up with the relatively slow cutting rate. The effect of rock with a higher fracturing intensity on the effectiveness of this mining method is still to be investigated by TDS.

A further problem of using high pressure abrasive water jets is the high heat generation, which is equal to about twice the geothermal heat input [19]. Also the high specific power consumption makes this method unsuitable for hydro-power operation since it would require approximately 18 tons of water per ton of rock.

2.4.3 Low pressure abrasive water jet (Diajet)

This method mixes the abrasive with the water in a high pressure tank at 20 MPa. Similar cutting rates can be achieved as for high pressure operation if bigger flows are used.

This approach is presently being tested by Technical Development Services [46]. Water flow rates have been doubled and abrasive usage has been increased to 2-3 times for equal depth of cut and performance if compared to the high pressure jet. The bigger jet diameter lends itself to the use of bigger and more effective steel shot and possibly greater depth of cutting. The steel shot can in this case be reclaimed and be reused up to 3 times before it disintegrates. Specific power consumption is 1/5 of the high pressure method. Considering that this method can also be directly powered by hydro-power with a minimum of hardware, it appears to be a much more promising approach.

2.4.4 Potential for improvements in abrasive water jet cutting.

One of the limitations of abrasive water jetting is the limited depth of cut, which is detrimental to good labour productivity and efficient secondary breaking. This cannot be improved with present designs because of the narrowness of the slots which prevent the nozzle from entering these slots. Several methods have been proposed to overcome this [44] which are mostly based on widening the slot. This may not necessarily increase the specific energy per area cut, as the nozzle can now operate at ideal stand off distances. An alternative approach to this is the use of long collinating tubes, which fit into the slot.

2.4.5 Potential for non-abrasive methods

Cavitating and pulsating jets produce high and pulsating impact loads, which are effective in rock fracturing [43]. A similar effect can be achieved with oscillating and rotating jets because of the of the repeated loading and unloading process. A literature survey has indicated claims for performance figures in hard rock which are similar to abrasive cutting (Table 5). It appears that a potential for ingenious combinations of methods of improved overall effectiveness may well exist.

Table 5. COMPARISON OF SPECIFIC ENERGIES FOR DIFFERENT TYPES OF WATER JETS

WATER JET TYPE	ROCK TYPE	PRESSURE (Mpa)	POWER (KW)	DEPTH OF CUT (mm)	VOLUME PER HOUR (m^3/hr)	SPECIFIC ENERGY (MJ/m^3)
ABRASIVE	QUARTZITE/ GRANITE	240	150	200	0,0130	41,500
ROTATING	PINK GRANITE	69	58	610	0,0210	20,000
PERCUSSIVE	CALIFORNIA BLACK GRANITE	50	42	2,5	0,0004	400,000
CENTRE BODY CAVITATING JET	CALIFORNIA BLACK GRANITE	69	45	10	-	65,000

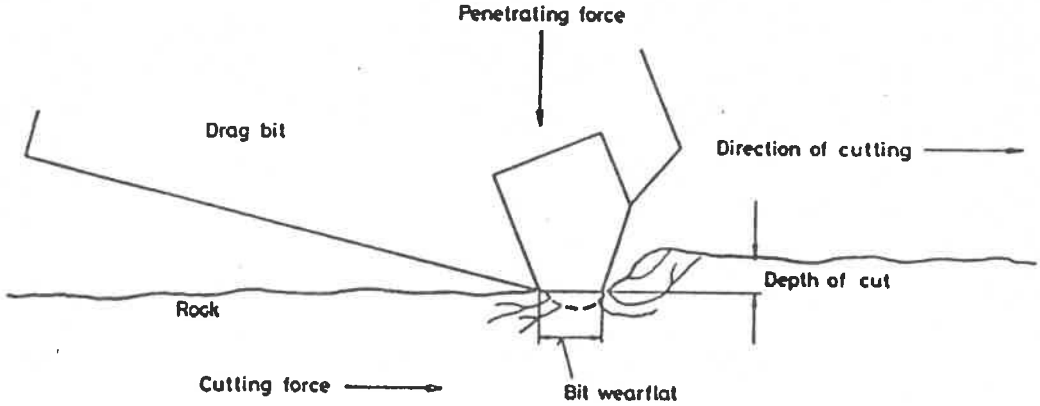


Figure 32. A DRAG BIT CUTTING IN HARD ROCK

2.5 Mechanical methods

2.5.1 Drag bit cutting

Drag bit cutting was developed by the Chamber of Mines in a period from 1965 to 1978 into a complete non-explosive mining system. The major motivation was the potential for selective mining for reefs with narrow channel width, which has a significant impact on the profitabiliy of mining.

2.5.1.1 Operating principles of drag bit cutting in hard rock.

During research conducted at the Chamber covering a timespan from 1967 to 1975, several basic characteristics of hard rock cutting were determined:

(i) The cutting action in hard rock is fundamentally different from cutting in softer rock [10] [2]. Whereas in softer rock a wedge action takes place, which is influenced by the rake and clearance angle of the tool tip, this does not apply in hard rock. There a high penetration force is generated, which is higher than the cutting force, so that a tool is quickly worn to produce a wearflat which is in contact with the rock and which has a zero clearance angle. The forces under the wear flat produce a spalling action ahead of the leading face, which therefore plays no role in the cutting action. A negative rake angle can therefore be used to enhance the tool strength as shown in Figure 32. Essentially the cutting action is a sliding indentation process and resistance to indentation determines cutting forces.

(ii) Wear is independent of cutting speed below a critical value. Above this value rapid deterioration occurs due to thermal overloading. This results in plastic deformation of the tungsten carbide and in failure of the braze joint. The temperature can be significantly reduced if water cooling is used, which allows operation at higher speed.

(iii) Tool forces increase less than proportional to the tool width. This is probably due to the edge effects of the cutting operation.

(iv) Penetration and cutting forces increase with tool penetration as shown in figure 33 and 34.

(v) Tool forces increase substantially with the length of the wear flat as shown in Figure 35 [48]. However there was little scope to reduce the length due to insufficient strength in the carbide.

(vi) During cutting a slip-stick effect occurs with rapid change of tool forces. This results in intermittent tool speeds which are very much higher than the average. Therefore a stiffer drive in the cutting direction reduces the peak speeds and permits an increase of performance. At the same time a decrease of stiffness in the penetration direction reduces penetration forces due to better avoidance of localized hard spots. Table 6 shows

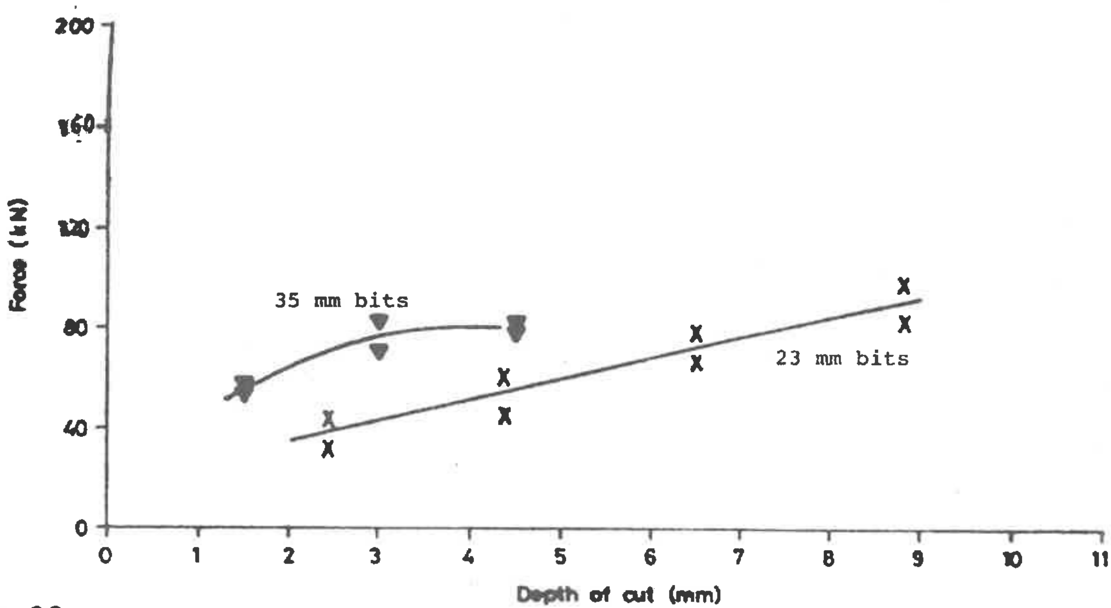


Figure 33 . MEAN BIT CUTTING FORCE PLOTTED AGAINST DEPTH OF CUT FOR 35 mm THICK BITS AND 23 mm THICK BITS

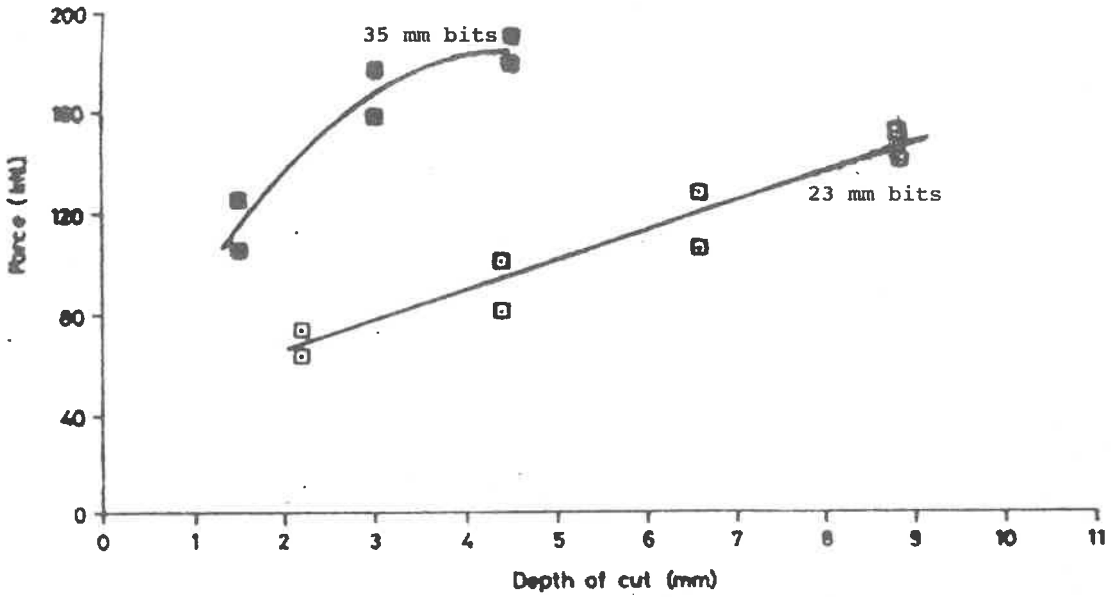


Figure 34 . MEAN BIT PENETRATING FORCE PLOTTED AGAINST DEPTH OF CUT FOR 35 mm THICK BITS AND 23 mm THICK BITS

Table 6. PERFORMANCE OF A SWINGING-ARM TYPE ROCKCUTTING MACHINE : EFFECTS OF VARIOUS CHANGES ON THE CUTTING LOADS WHEN TAKING A CUT 6 mm DEEP IN NORITE

Machine features	Maximum cutting load (kN)	Average dynamic cutting load (kN)	Average penetrating load (kN)
Original design	100	66	200
With a blade having radial compliance .	82	48	145
With a kinetically neutral blade	77	49	160
With a large moving mass	73	45	140
With a high stiffness of hydraulic drive .	66	40	110
With a choke in return line	66	37	110

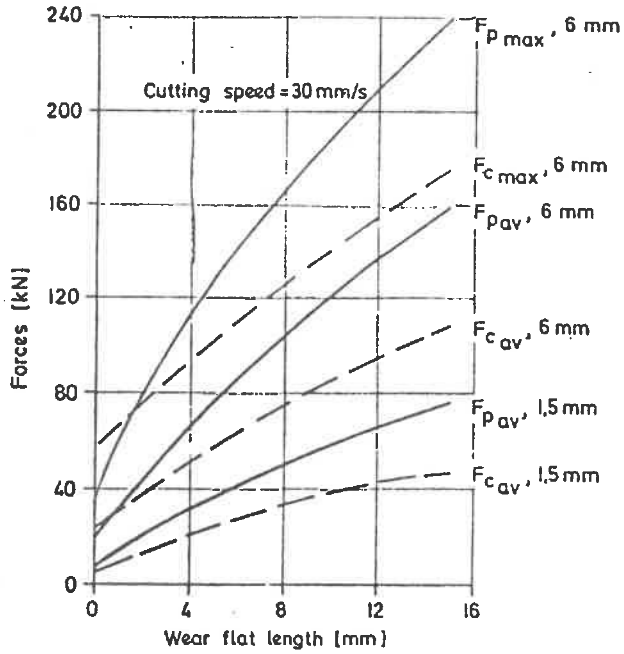


Figure 35. DIAGRAM SHOWING THE INFLUENCE OF WEAR-FLAT LENGTH UPON CUTTING FORCE Fc AND PENETRATING FORCE Fp AT CUTTING DEPTHS OF 1,5 AND 6,0 mm (MARIEVALE QUARTZITE, SLOT WIDTH - 30 mm)

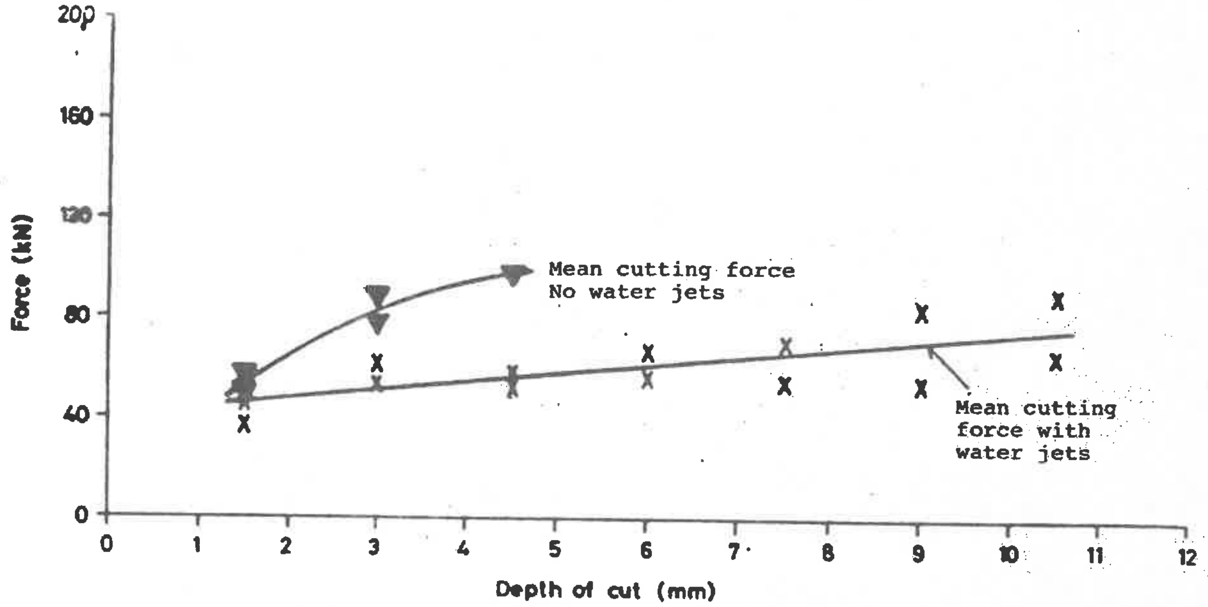


Figure 36. MEAN BIT CUTTING FORCE PLOTTED AGAINST DEPTH OF CUT ILLUSTRATING THE REDUCTION OF THE FORCE REQUIRED TO CUT THE ROCK WITH 50 MPa WATER JETS DIRECTED AHEAD OF THE BITS

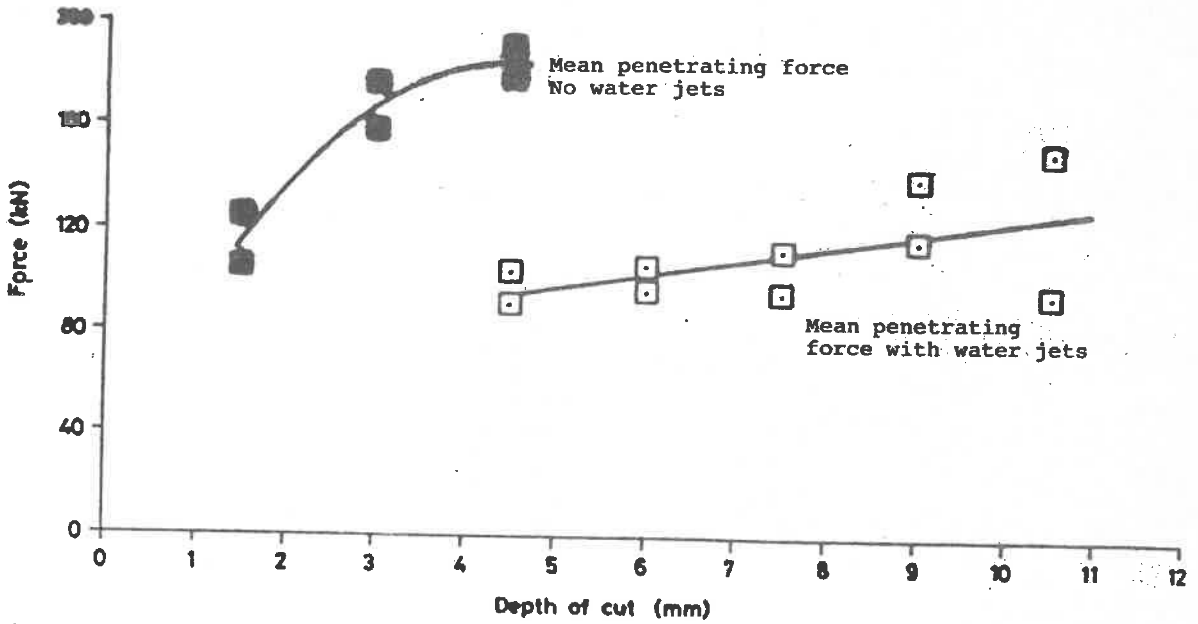


Figure 37 MEAN BIT PENETRATING FORCE PLOTTED AGAINST DEPTH OF CUT ILLUSTRATING THE REDUCTION OF THE FORCE REQUIRED TO CUT THE ROCK WITH 50 MPa WATER JETS DIRECTED AHEAD OF THE BIT

the improvements which were achieved as a results of better tool and machine design (10).

(vii) High pressure water jetting to the inside corners 2 mm ahead of the leading edge makes a substantial difference to the tool forces as shown in Figure 36 and 37 [2]. Whereas the level of water pressure had a noticable effect on penetration forces it made very little difference to the cutting forces.

High pressure water jetting increases the possible cutting rate of the tool significantly. This was applied underground in 1975.

2.5.1.2 Application to stoping and performance results

In underground testing several more site related influences were determined:

(i) In densely fractured rock penetration could be increased to 4 to 5 times the value in unfractured rock with a tool life up to 10 times higher.

(ii) Rock strength had a significant influence on possible penetration. If cuts were made at an interface of rock layers of different hardness, the tool tended to deflect into the softer strata. Cuts against the hanging also induced side forces into the tool.

Typical performance data for the cutting of slots in carbon leader reef on Doornfontein at a depth of

2500 m were as follows:

Depth of penetration in unfractured rock without water jet assistance : 2-3 mm.
Depth of penetration in fractured rock without water jet assistance : 8-10 mm.
Depth of penetration in unfractured rock with water jet assistance : 10-15 mm.
Depth of penetration in fractured rock with water jet assistance : 40 mm.

The water jet details were a pressure of 40 MPa at a flow of 0,67ℓ/s. Under these conditions cutting speeds were 150 mm/sec, slot width was 35 mm, mechanical power input was 30 kW and maximum total depth of cut was 600 mm. An average cutting rate of 4 m^2/h was achieved with a tool insert life of 6,4 m^2 when cutting against harder rock in the hangingwall or 11,6 m^2 when cutting in softer rock. Cutting along the footwall proved impractical because the equipment was covered by the loose rock. This led to the use of impact hammers to mine hard footwall.

The specific energy for drag-bit cutting of 500 MJ/m^2 is high, indicating that drag bit cutting is only suitable for slotting and therefore requires secondary means of rockbreaking, if considered as a mining method. One of the practical disadvantages of drag bit slotting are the high operating forces, which require heavy machinery and staking to absorb the reaction forces.

Work on drag-bit cutting as a basis for a stoping system was concluded in 1978, when it was realized that such a system could not achieve the targets for labour productivity, which only became an issue, when due to changing circumstances in labour supply in 1975 the target was raised. Also by that time impact ripping had been investigated sufficiently to let it appear as a more promising solution.

Further details of the development of rock cutter systems are given in the section on Integrated Mechanized Systems.

2.5.2 Roller Cutting

Roller cutting is better suited for very hard unfractured rock like quartzite than any other mechanical method. This is largely due to the very strong tool shape and the large wearing surfaces. This method has found widespread use in South African gold mines for raise and boxhole boring, despite the fact that operating costs are approximatly 4 times higher than for drill and blast techniques. In this case roller cutting is justified with the increased speed of operation and improved safety, both of which resulted in secondary savings.

Typical performance data for raiseboring in quartzite of high strength are average penetration rates of 1 m/h and an instantaneous rock removal rate of 2,3 m^3/h for a diameter of 2,1 m at a power input of 150 KW. This results in a specific energy of approximately 200 MJ/m^3.

Roller cutting has also been tried for tunnelling. However, this has not been successful. This can be explained with the difference in operating conditions. Whereas raiseboring typically takes place in a steeply inclined position in areas which are overstoped and therefore of low stress, tunnelling often takes place in highly stressed areas. This causes spalling along the sides of horizontal excavations and also in the cut face. Consequently large pieces of rock dislodge, causing jamming and rapid cutter wear. This necessitates frequent cutter replacement and altogether uneconomical operation.

Attempts were also made to investigate the potential for roller cutting to bore out the reef with a series of overlapping holes. The justification was again that considerable secondary savings could be made from the reduction of stoping width and the fully mechanized mining operation. This will be discussed under 2.5.2.2.

2.5.2.1 Operating principle of roller cutting

Roller cutting is an indentation process which requires high forces under the tool to crush and spall the rock. Resistance to indentation is heavily dependent on quartz content as shown in Figure 1, therefore performances will depend on rock strength. The influence of the important variables on forces and performance are given in Figure 38

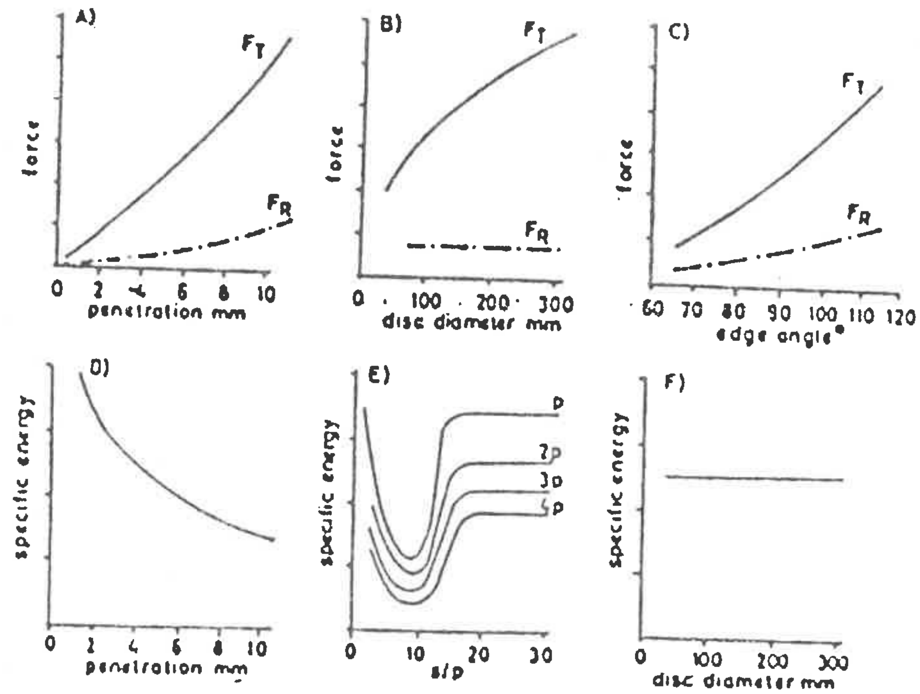


Figure 38. SOME BASIC RELATIONSHIPS FOR DISC CUTTERS

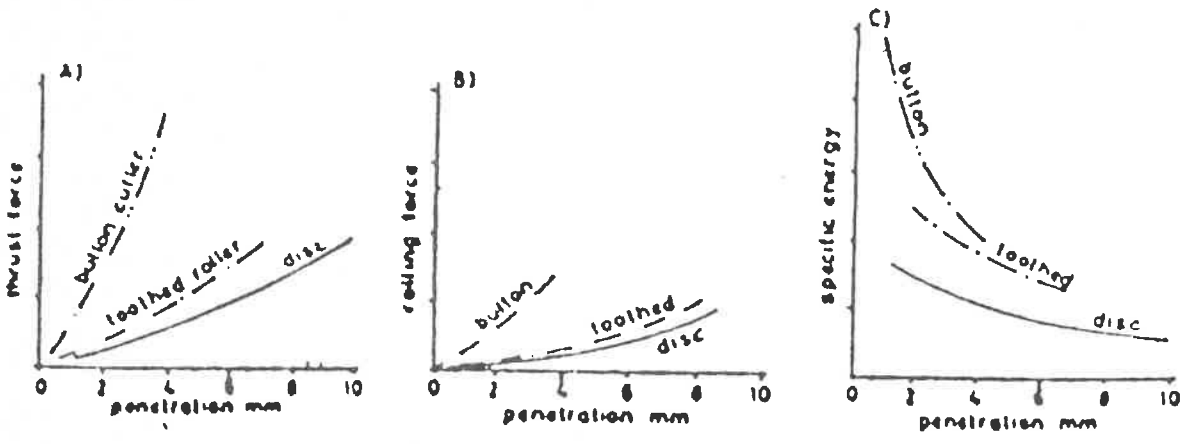


Figure 39. RELATIVE PERFORMANCES FOR DISC, TOOTHED AND BUTTON ROLLERS

(49). The forces acting on the rollers vary considerably during cutting under constant operating conditions. Figures up to 10 times higher than the average values have been recorded. Two different types of cutters are in use, namely disc and toothhead types with or without buttons. The differences in forces and performance are shown in Figure 39. In South African mining generally the toothhead cutter with buttons also referred to as kerf cutter is used because only with tungsten carbide buttons can a reasonable wear life be achieved.

The high wear and operating costs of roller cutters in South African quartzite, which amounts to 40% of the total cost of boring, prompted extensive investigations on a linear roller cutter test rig to alleviate the problems. Methods of reducing costs were to be found by optimizing operating parameters and by studying the benefits of high pressure water jets. The results showed that up to 40% reductions in the thrust and rolling forces were possible with 4 jets of of 40 MPa using 0,3ℓ/s each [50]. Details of the improvements are shown in Figure 40. It must be noted that the graphs for specific energy do not include the energy requirements for the water jets which were higher than the energy delivered to the roller cutters.

The influence of water pressure on performance is shown in Figure 41. Important is the improvement of worn disc cutters to the level of unassisted new disc cutters as shown in Figure 42. This would

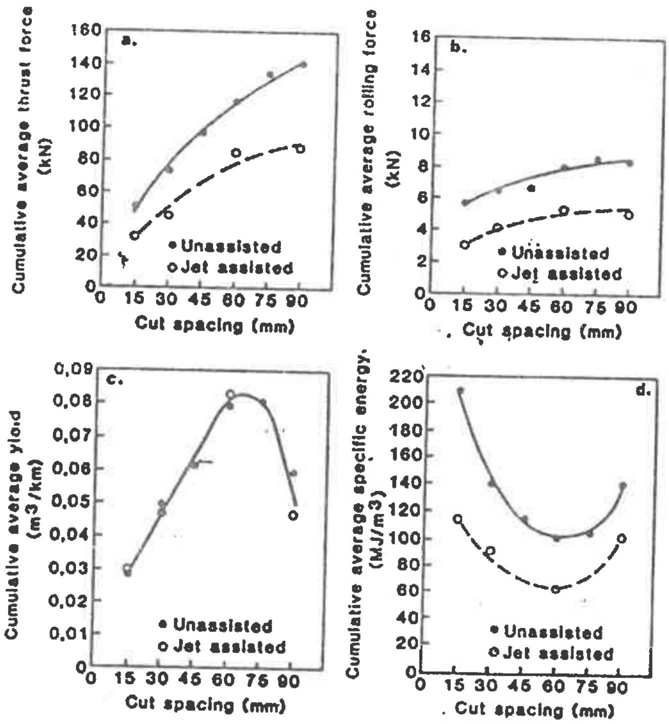


Figure 40. THE EFFECTS OF 40 MPa WATER JETS ON THE DISC CUTTING PARAMETERS FOR A 2 mm DEPTH OF CUT AND FOR VARYING CUT SPACINGS

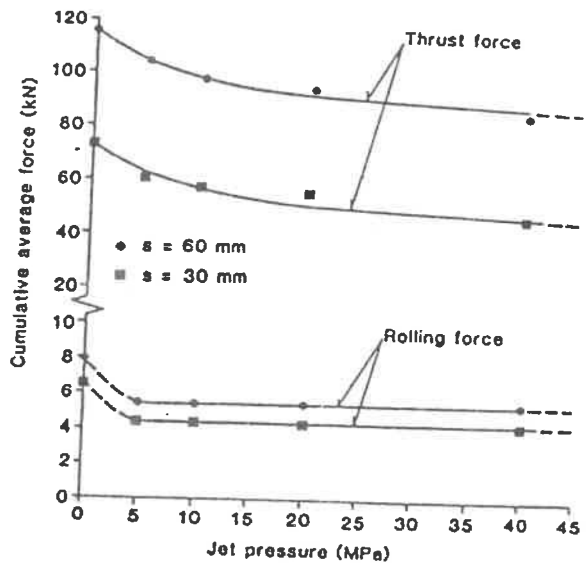


Figure 41. THE EFFECTS OF JET PRESSURE ON THE CUTTING FORCES OF A DISC TYPE CUTTER FOR A 2 mm DEPTH OF CUT AND FOR 30 mm AND 60 mm CUT SPACINGS

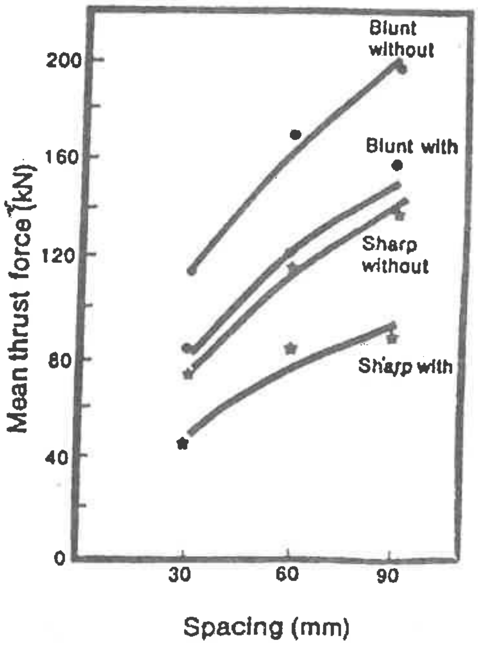


Figure 42. EFFECT OF WATER JETTING ON NEW AND WORN DISC CUTTER

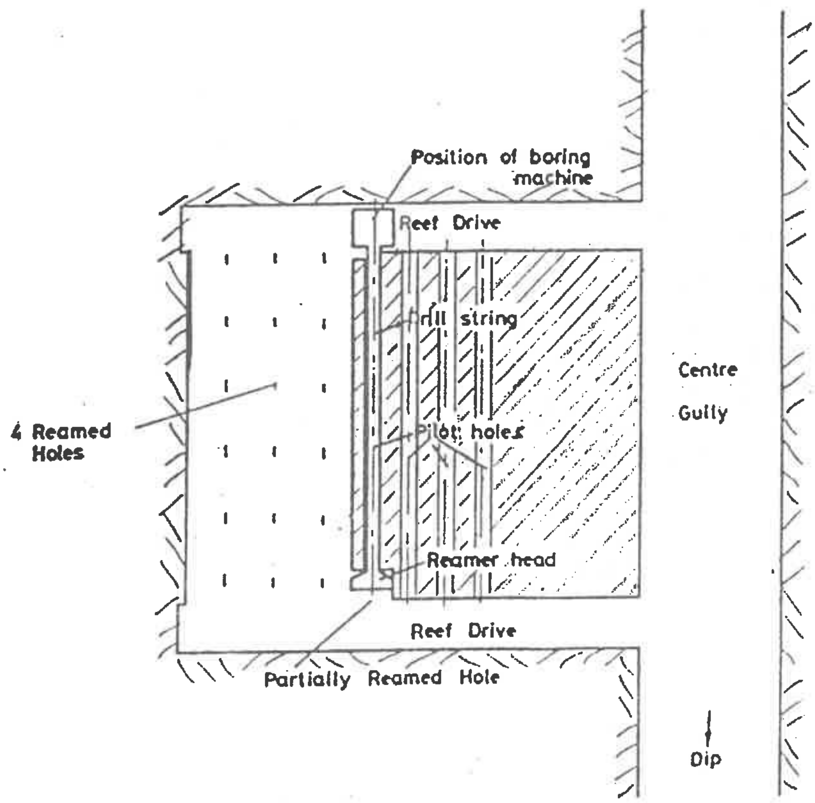


Figure 43. LAYOUT FOR REEF BORING

enable the more efficient disc cutter to be used in quartzite. Improvements could also be determined for disc cutters with buttons, but to a lesser degree than for cutters without buttons.

The application of these findings in field trials is to commence soon. The chances of disc cutting being sucessful has been increased recently by the manufacture of one piece tungsten carbide discs. Attempts to increase the performance of roller cutters by generating vibrating thrust forces on them has not been successful. In 1975 a proposal was made by DEMAG for an activated cutter based on rotating excentric masses in large diameter cutters. This was not pursued because the forces would only have increased by 20 to 30%. Further tests conducted by Bergbauforschung in Germany confirmed that the marginal improvements in cutting rates did not justify the additional mechanical complexity.

2.5.2.2 Application to stoping and performance results

Reef boring was considered as a possible stoping method for narrow reefs of reasonably even horizontal extension. By boring overlapping holes of larger diameter than the width of the reef, it was thought that all the reef could be recovered at a considerably reduced stoping width compared to conventional mining. Holes had to be reasonably short so that demands on drilling accuracy were not too high and that excessive loss of reef due to minor faults and rolls could be avoided. The method was only considered a possibility for unfractured quartzite.

In 1975 a site was chosen on West Driefontein with a very low energy release rate of only 2 MJ/m^2. Pilot holes of 228 mm diameter were drilled on 550 mm centres between two reef drives 20 m apart. These holes were then reamed out to 560 mm diameter, using the pilot holes as a guide to prevent run out into the reamed out hole next to it. The general layout is shown in Figure 43.

The results were an increasing rate of spalling in the pilot holes next to the reamed out holes as the excavation increased [51]. This could be shown to be a result of the stress concentration caused by the advancing excavation. This made the pilot hole eventually unsuitable to serve as guides for the reamer. Also the operating conditions for the reamer were now in highly stressed and fractured rock. The cutters were pinched and damaged by hole closure and the reaming head ran out of the hole being reamed into the one previously completed. Various attempts were made to overcome these problems, such as grouting the finished holes and changing the separation and drilling order of hole. However, these were all unsuccessful. Consequently the life of the cutters was severely reduced to 30 m on average. It was also shown that maintaining the required drilling accuracy for the pilot holes was exceedingly difficult even with special care.

The technical difficulties and operating costs associated with the method led to it being abandoned.

An alternative approach was investigated from 1975 to 1977 to use roller cutting as a face mining machine. The concept was to cut a web parallel with the face by moving two large diameter cutters in an arc back and forth. These rollers were pointed into the corners of the hanging and foot walls. Laboratory experiments were conducted which proved the feasibility of the concept. However, the manufacture of an underground mining machine was subsequently abandoned when it was realized that the technology of roller cutting had not advanced sufficiently, as claimed by the manufacturer, to enable such a macnine to operate at acceptable costs.

2.5.2.3 New developments

A recent development of a new technique could significantly widen the application of roller cutting [52]. With this method roller cutters break towards a free surface and can therefore break large pieces of rock with a claimed reduction of specific energy to 1/10 of conventional values. This would be better than any other full face non-explosive rockbreaking method in unfractured rock. The principle of operation is shown in Figure 44. Interesting features are the different way in which reaction forces are handled and the potential to produce other than circular holes. Once these claims have been substantiated, the potential for adapting the principle of operation to stoping has to be evaluated.

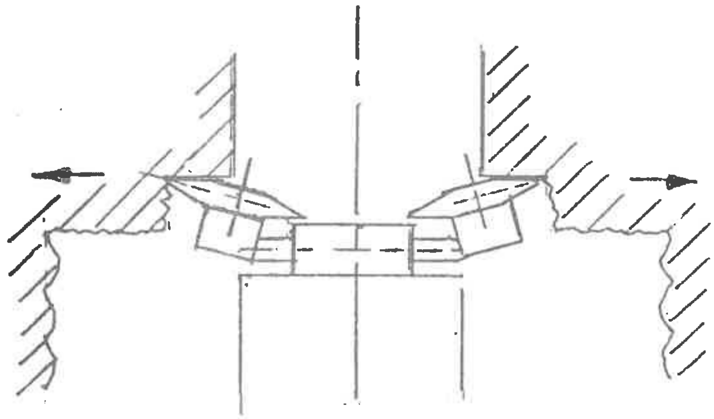


Figure 44. OPERATING PRINCIPLE OF RADIAL ROLLER CUTTING TECHNIQUE

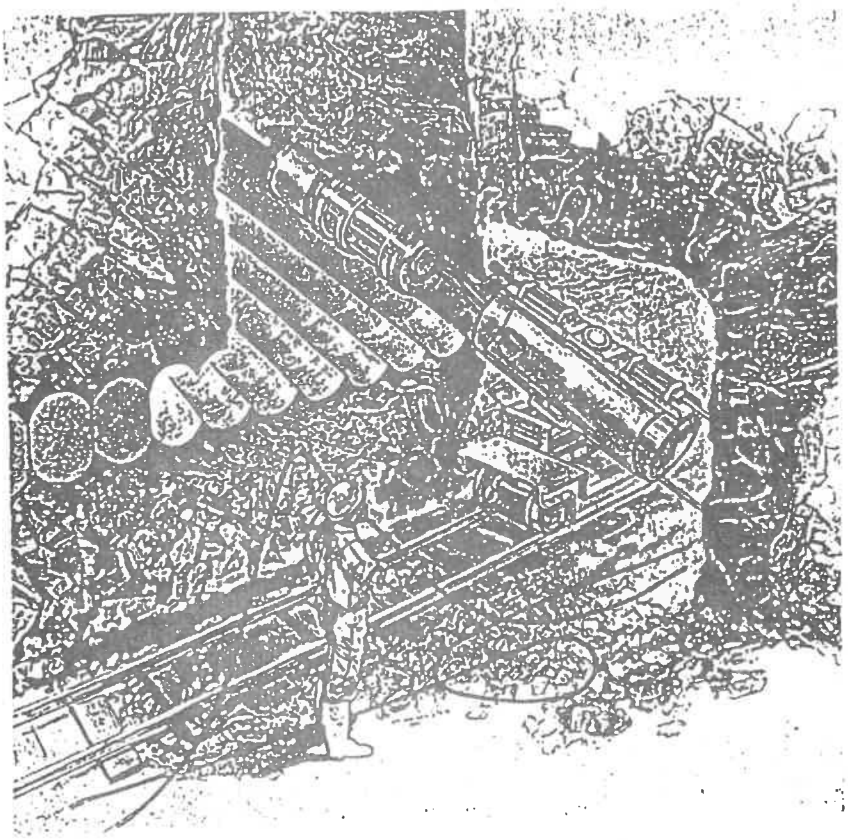


Figure 45. THE STOPECORER, A HYDRAULICALLY POWERED DIAMOND CORE DRILL THAT CAN PROPEL ITSELF UP OR DOWN THE HOLE IT IS DRILLING

2.5.3 Diamond cutting

2.5.3.1 Operating Principle and state of diamond technology

Diamond cutting also operates by indentation, but mostly on a microscopic scale. Multiple indentation is used which can have random distribution or designed distribution for larger diamonds. Specific energy is very high particularly for the smaller diamonds, so that only the removal of small volumes can be economically justified. Diamonds have been used in quartzite for core drilling, slot cutting with saws and large diameter ring slotting for boring.

The state of diamond technology for rockbreaking can be seen from applications for core drilling:

When impregnated synthetic diamonds are used, a BX drill bit of 60 mm O.D, and 42 mm I.D can typically achieve the following performance in quartzite: penetration 0,12 m/min, life 8 m, bit cost R60/m. Drilling would occur with a thrust force of 6 KN at 800 RPM. This method is slow and expensive.

By using natural diamonds the penetration rate and the life can be improved at approximately similar costs.

In a very recent improvement which uses Syndax-3 diamonds in a specific pattern on the surface of the bit only, penetration rates of 0,74 m/min have been achieved with a life of 20 m at a cost of R40/m [53].

Whereas the specific energy for the impregnated bit was 1900 MJ/m³, it is only 320 MJ/m³ for the new bit.

2.5.3.1 Application to stoping and performance results

(i) Saw cutting

Saw cutting was investigated by Goldfields in 1968 for slotting of rock with a blade of 1,2 m diameter. Difficulties were encountered in fractured rock which pinched and damaged the blade, resulting in a premature loss of diamonds. The operating costs were therefore far too high for practical consideration. Slot cutting with diamonds saws is being re-evaluated by TDS [46], because of progress in diamond technology.

(ii) Large diameter core boring

This method was used as a mining method by STOPECORER from 1974 to 1981 on Vaal Reefs and Sallies to bore out stressed and fractured rock. Overlapping holes of 660 mm diameter were generated by cutting a ring slot of 13 mm width and relying on stress fracturing in the rock to disintegrate the core. Figures 45 and 46 indicate the process.

Performance results were as follows [53]. Penetration rates were 0,12 m/min with a tool life of 20 m. Power input was 100 kW at 200 RPM. The best monthly performance was 246 m² and in total 10 000 m² were mined. Operating costs for the tool

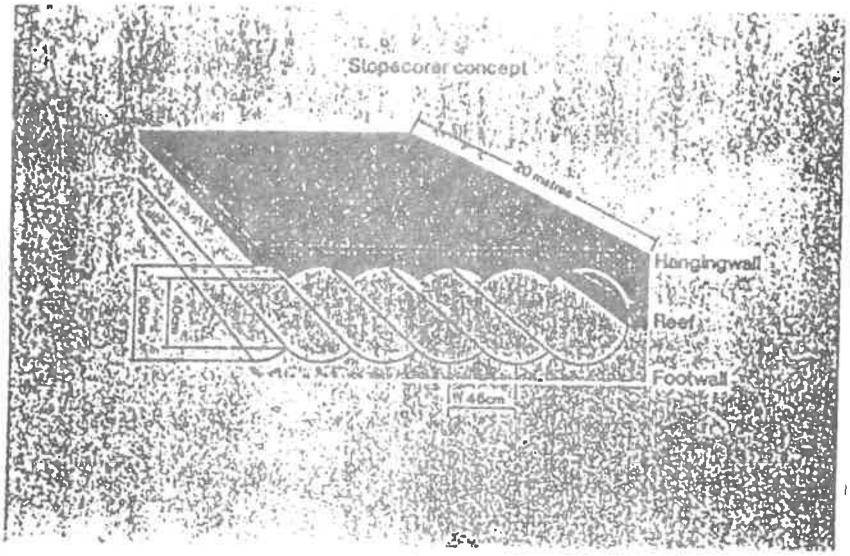


Figure 46. THE MINING PRINCIPLE OF STOPE CORING, DRILLING OVERLAPPING HOLES

Figure 47. PROPOSED NEW CORING PRINCIPLE

alone were approximately 2-2,5 times conventional stoping costs. Practical problems were encountered when the core did not disintegrate, as the machine configuration required the core to be ejected to the side. In this case the drill had to be retracted to remove the core. Besides the time loss this caused problems with stope closure, so that the hole had in fact to be reamed out again. Also spalling in the hole interfered with the clamping action of the drill. This would result in loss of directional control and vibrations. This experiment was discontinued in 1981.

However interest has been expresssed by AMTEC in reviving this method [53]. The reasons given are:

(i) The new diamond technology would substantially improve performance and reduce operating costs.

(ii) The non-disintegrating core problem may possibly be overcome by a system similar to Figure 47.

The revised system may have potential for deep level narrow reef mining, and needs to be reassessed.

(iii) Other methods

Diamond rope cutting is being used extensively for quarrying. It has the advantage of requiring a minimum of hardware in the cutting area. For narrow reef mining it has potential for a man-free stope operation, if the reef only is slotted out by equipment installed in advance gullies. Therefore an extremely low stoping width with very low energy release rates may be possible [54].

Discussions are in progress with equipment suppliers to obtain performance data and information on principles of operation. If suitable concepts for application in gold mining can be devised, then experiments would be conducted to determine if rope cutting is feasible in fractured quartzite and if operating costs and performance justify development efforts.

2.5.4 Impact breaking

Impact breaking was first used in the South African gold mining industry for the drilling of blast holes. The application is described in the section on Rock Drilling. Hand-held paving breakers were used during the rock cutter trials to remove the stress-fractured but still interlocked rock after the slot had been machined into the rock.

Impacting was also considered for rock slotting, which is explained under 2.5.4.2. The application of impacting in conjunction with bull wedges which were inserted into drilled holes for use in areas where paving breakers were inadequate, is discussed under 2.1.5.2. More poweful oil hydraulic hammers, which had been developed for quarrying and demolition work, had first been considered in 1967 to assist the rock cutter to remove partially fractured footwall. The promising results of this equipment led to further trials on Doornfontein and Westdriefontein in 1968 when impact hammers were used by themselves as a rockbreaking method for fractured rock conditions. The subsequent development of complete mining systems based on impact ripping, which is now the most advanced non-explosive mining technique, is described in the section on Integrated Mechanized Systems.

2.5.4.1 Fundamentals of impact breaking

In impacting a constant power supply is converted to intermittent short duration high level force pulses. These high forces can produce large rock fragments at low specific energy levels. An important practical benefit of the method is the low average reaction forces produced; this means that at low power levels machines can still be hand-held, and at high power levels where instantaneous forces of several hundred tons can be generated, staking is not required.

The damage caused to rock by impacting occurs firstly by crushing and spalling, and when forces are sufficiently large, also by splitting. For spalling and splitting to occur, cracks must be able to propagate to a free surface. Free surfaces are provided in different ways for different applic-ations of impacting. For instance in drilling, free surfaces are maintained by rotating the tool between impacts, whereas in fullface ripping, the tool is moved parallel to a free surface as shown in Figure 56. Another difference between drilling and ripping is that for drilling the main concern is for maximum tool penetration per blow, which must be accompanied by rock removal by spalling or crack propagation, essentially at 90° to the direction of impact. Also only one blow is imparted per position. In ripping the main purpose is to split the rock by propagating cracks over long distances ahead of the tool and in line with the direction of impact. Usually several blows at each position are required to propagate the cracks.

Impacting for the purpose of ripping is discussed below, although most of the fundamentals apply equally well to drilling or slotting.

The energy content of the pulse or the blow energy is determined by the energy in the piston on impact with the tool. The piston energy is determined by the mass of the piston and its speed on impact. Dimensional restrictions in an underground stope and the need to manipulate the impact device limit the maximum dimensions of the impact device and therefore its piston mass (Figure 48). Since the piston speed is limited by the strength of materials on impact with the tool, the maximum blow energy is limited to approximately 4000 to 6000 J for the given dimensions. This can only be increased by using ballistic hammers, which can operate at higher impact speed by avoiding the piston to tool impact, because the tool is connected to the piston. This requires, however, that the tool tip is kept a constant distance away from the rock at the beginning of each impacting stroke, which is difficult to achieve.

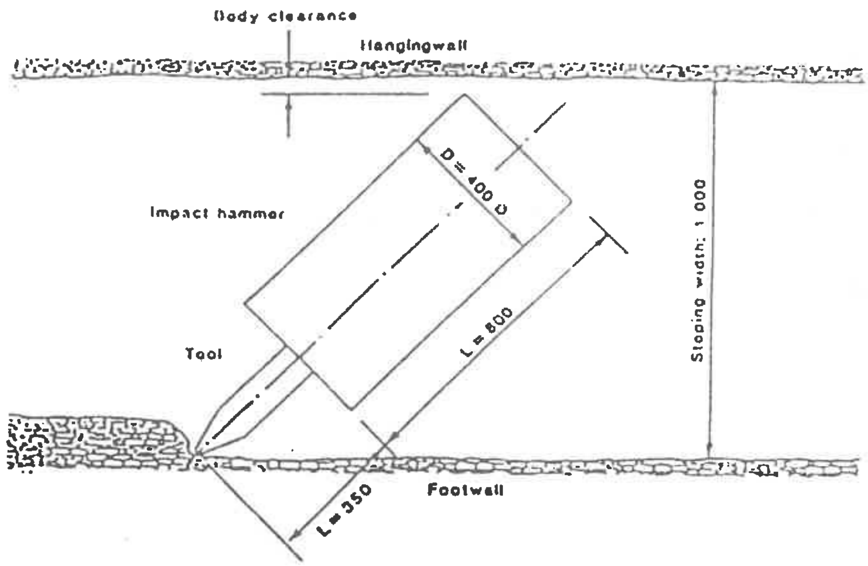


Figure 48. CRITICAL DIMENSIONS FOR IMPACT HAMMERS IN NARROW STOPES (QUOTED FIGURES APPLY TO PRESENTLY USED CONFIGURATION)

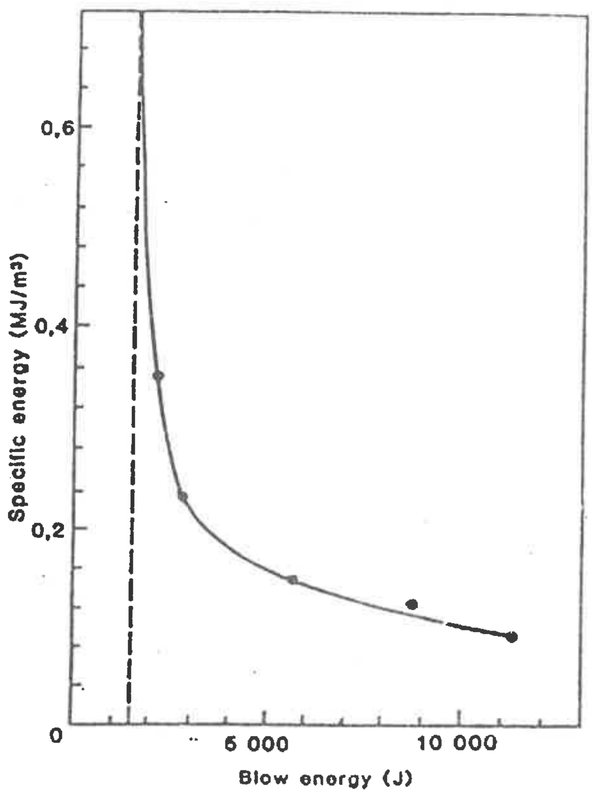


Figure 49.

LABORATORY DROP TEST RIG RESULTS SHOWING THE EFFECT OF BLOW ENERGY ON THE SPECIFIC ENERGY OF SPLITTING 0,5 m CUBIC BLOCKS OF NORITE WITH A WEDGE SHAPED CHISEL TOOL

The effect of blow energy on driving a crack for a given distance was established during research conducted by the Chamber on a drop tower test rig in 1977. The result is shown in Figure 49.

The blow energy which is available for rockbreaking is also influenced by the efficiency of energy transfer between piston and tool. This is determined by the relationship of piston to tool mass and length [58] and by the concentricity of the impact. Here again a ballistic hammer has an advantage.

Besides the energy content of the pulse, rockbreaking effectiveness is also influenced by the shape of the pulse. In this regard a distinction must be made between the pulse generated in the tool by the impact of the piston on the tool and the pulse induced in the rock. This tool pulse cannot be fully transmitted to the rock when if arrives at the tool tip. Consequently it is reflected several times before all the energy is transmitted to the rock [58]. A typical pulse shape as induced in the rock is shown in Figure 50. The oscillations of the tool pulse are superimposed on the rock pulse. It has been demonstrated that the pulse shape of the tool affects rockbreaking [61]. It has also been shown that the pulse shape can be changed by a number of parameters such as the piston shape as shown in Figure 51 [62], the shape of the interface between piston and tool, as shown in Figure 52 [62] and by the shape of the tool. Because of the reflections within the tool the pulse shape becomes

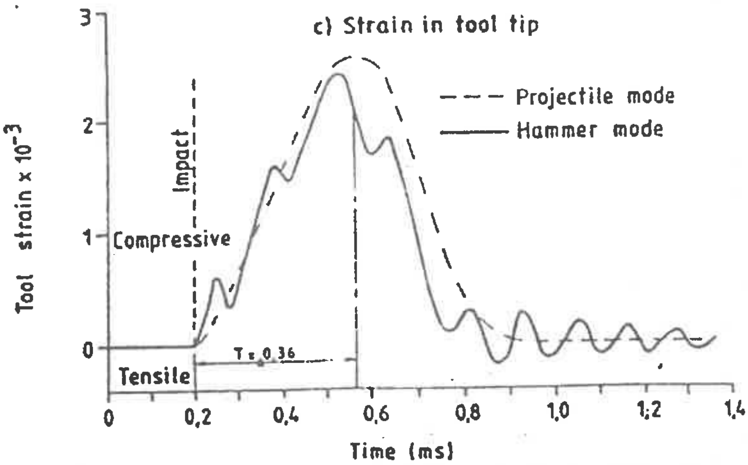


Figure 50 THE PISTON MOTION, PENETRATION OF THE TOOL INTO THE ROCK, AND STRAIN MEASURED IN THE TIP OF A WEDGE SHAPED CHISEL TOOL WHEN IMPACTING NORITE IN THE DROP TEST RIG AT A BLOW ENERGY OF 2 800J

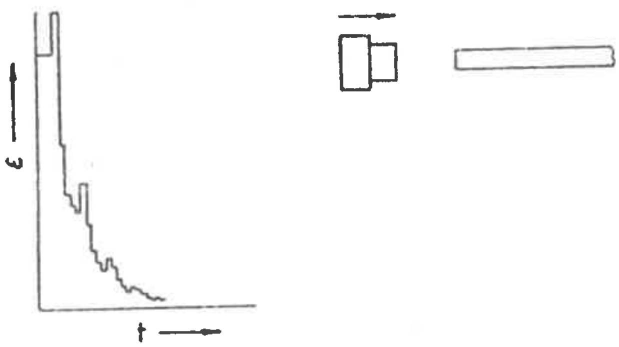


Figure 51. COMPUTED PULSE SHAPES IN A STEEL BAR CAUSED BY IMPACT OF VARIOUS SHAPED PISTONS ON THE BAR

progressively more distorted as a function of the tool shape. Research by the Chamber [60] has shown that when a pulse is induced in the rock, a short pulse of high amplitude (Figure 53) as generated by piston to tool impact is far more effective for rockbreaking than a long flat pulse as is generated when the tool is impacted via a fluid tappet (high velocity hammer).

Furthermore the shape of the tool tip has a significant influence on rockbreaking effectiveness [60] (Figure 54) and can be ascribed to the stress field in the rock and the stress concentrations in the rock. The application of this finding is made difficult by the limited strength of materials for tools. Therefore research is in progress to investigate alternatives such as tungsten carbide.

Impact breaking of rock invariably results in the crushing of rock under the tool tip due to the high stresses. Since multiple blows are often required to drive a crack for a sufficient distance, this crushed rock has a dampening effect on energy transmission. Therefore water flushing through the tool as used for drilling would be desirable, also because of improved dust control and improved cooling of the tool tip.

Generally the theory of rockbreaking by impacting is far better understood for tool penetration as applicable to rock drilling, than for large scale rock splitting as required for impact ripping. More fundamental research is therefore required to enable

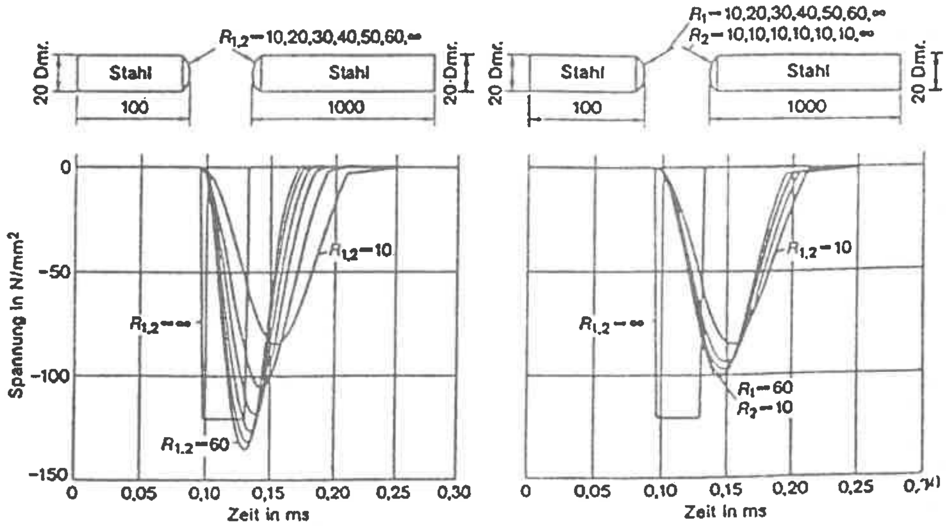


Figure 52. <u>STRAIN WAVE IN THE TOOL AS A FUNCTION OF THE RADII OF THE PISTON AND TOOL</u>

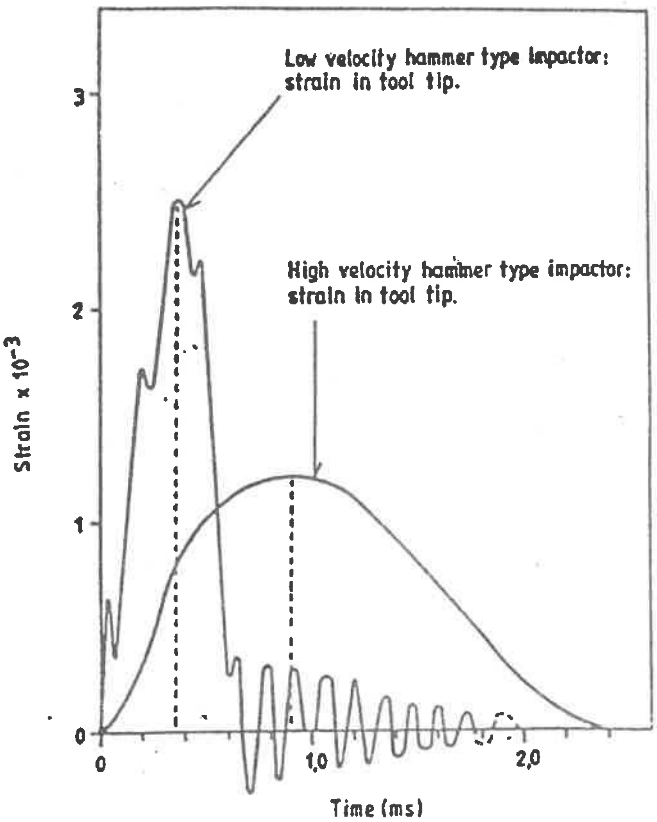


Figure 53.

<u>A COMPARISON OF THE STRAIN PULSES IN THE TOOL TIP FOR THE HIGH AND LOW VELOCITY HAMMER TYPE IMPACTORS WHEN IMPACTING ON ROCK</u>

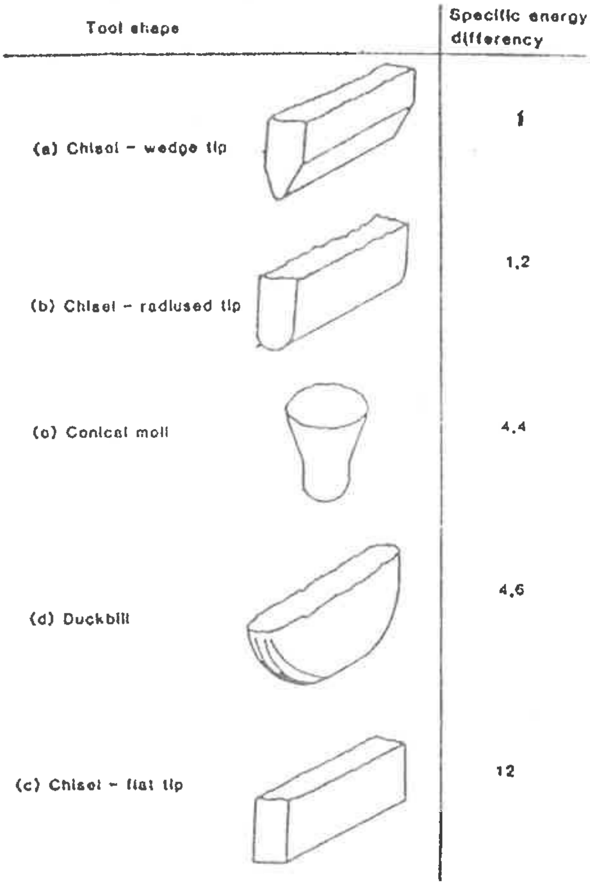


Figure 54. SOME OF THE MORE IMPORTANT GEOMETRIES OF TOOLS USED IN THE DROP TEST RIG

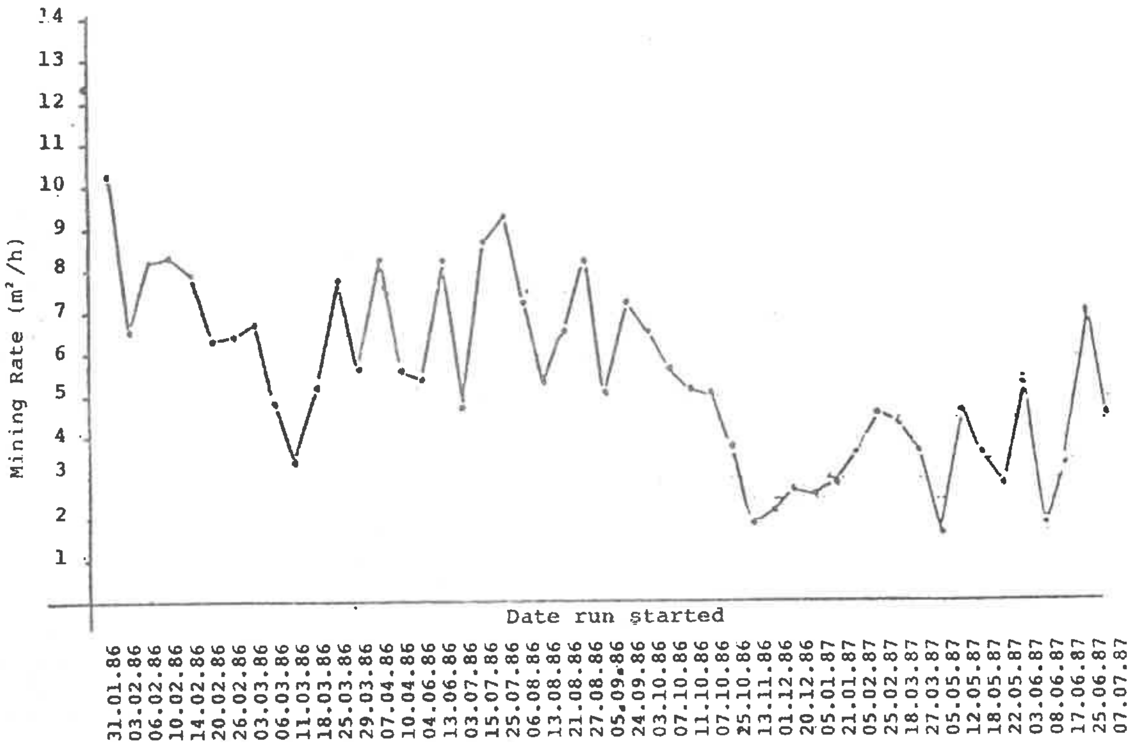


Figure 55. CHANGE OF ROCKBREAKING RATES FOR SECOND LAST IMPACT RIPPING SYSTEM FROM JANUARY 1986 TO JULY 1987 FOR EACH FACE ADVANCE

the optimization of rock splitting by impacting.

Additional influences on rockbreaking effectiveness are brought about by the special conditions encountered in underground impact breaking. One major influence is the unevenness of the fracture conditions. This causes the rockbreaking rates to vary widely, not only as the machine mines along the face, but also for the average of one complete run to the next, as is evident from Figure 55. Even adjacent areas of 1 000 m^2 can have average rock-breaking rates differing up to 30%. This makes comparative underground testing very difficult and time consuming. The breaking conditions underground are typically much more difficult than for instance those encountered for boulder breaking on surface. Underground, the rock fragments are still keyed together with fractures sometimes opened or closed. The rock is also confined in the hanging and foot walls and can still be subjected to varying degrees of confining stress. Some of the fracture spacings are so far apart in the so called hard patches that hammers with up to 6000 J blow energy are not able to break them, necessitating the development of auxiliary devices, as described in previous sections. The histogram in Figure 5 indicates that for the areas which were mined, most of the time was spent breaking relatively unfractured rock. Therefore the maximum possible blow energy should be used to improve rockbreaking rates.

A further influence of the underground conditions on rockbreaking is the fact that the rock fractures are mainly parallel to the rock face. Therefore the direction in which the rock is attacked becomes significant. Three different possibilities are shown in Figure 56. The choice is limited for underground operation because of rockhandling constraints which is explained under mining systems. Case A shows the mining of a narrow web all the way down the face. Rockbreaking could be very difficult if the fracture spacing was larger than the web depth. In case B down-dip mining with a large web depth is shown. In this case the angle of attack in relation to the fractures changes from a situation similar to case A, to a direction in line with the cracks and eventually with a force component which points away from the face. Therefore a wedging action and inducement of very beneficial bending loading should be possible. In case C up-dip mining with rock attack perpendicular to the fractures is shown. Smaller rock particles are likely to be produced with this method. Again rock cannot be loaded in bending if fractures are closed.

Common to all of these methods is that rock is broken to a free face from a preset depth of cut. The tool is angled to the direction of cut to maintain depth of cut without excessive side loads and to clear the hammer body. The influence of both the attack angle and of the depth of cut on performance have not yet been investigated.

Of the three described mining methods cases B and C have been tested with impact rippers. No significant performance differences could be

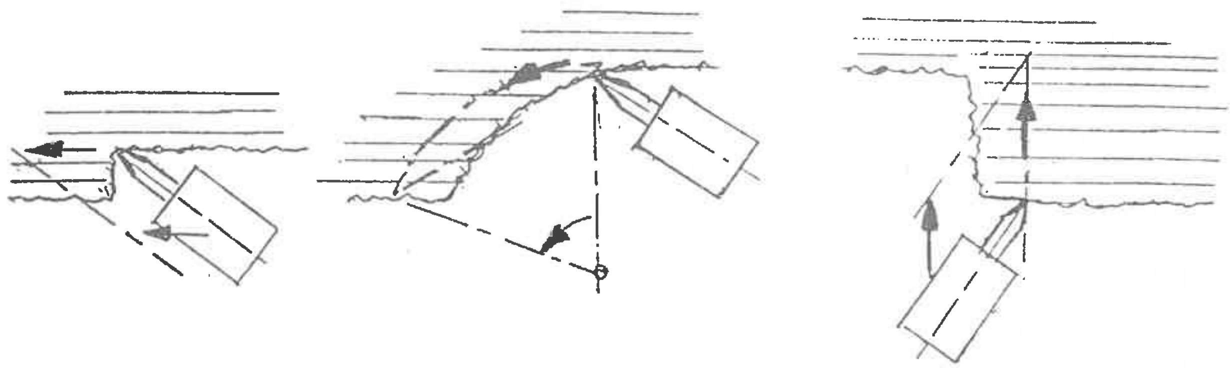


Figure 56. METHODS OF MINING

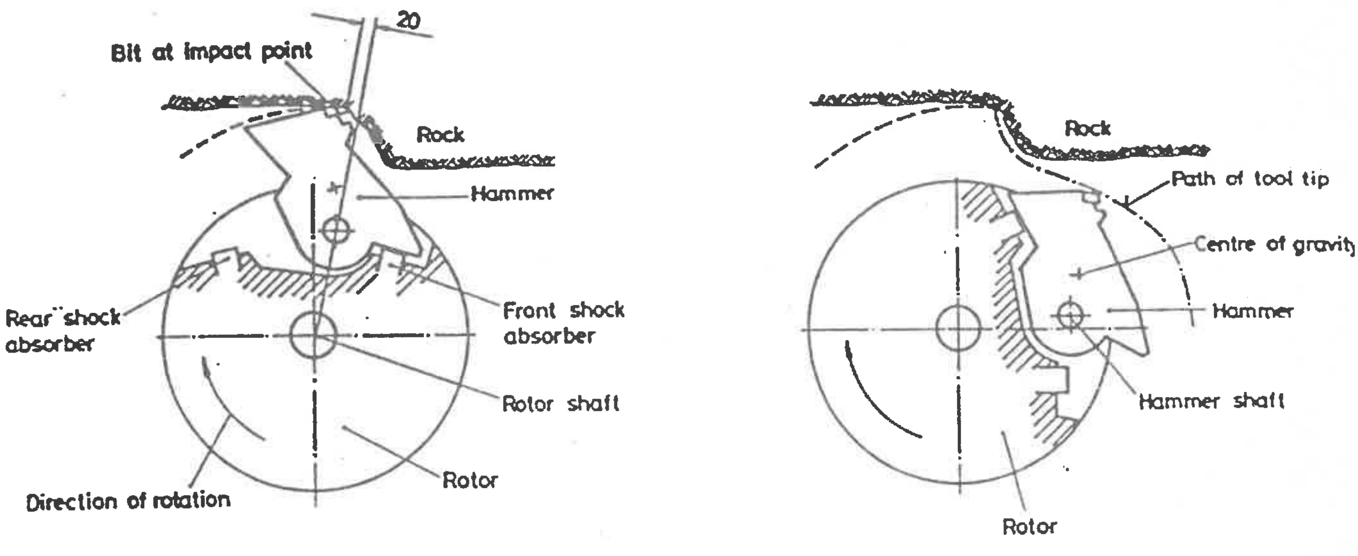


Figure 57. HAMMER ACTION OF THE SWING HAMMER MINER

determined under the difficult test conditions. Therefore rockhandling considerations settled the argument in favour of case B. Case A has been tested with the swinghammer miner and was intended to be used with the impact planer.

2.5.4.2 Application of impacting to slotting

This concept was developed by Boart Research Laboratories in 1974. It operated with a conventional rockdrill which moved the tool tip at the end of the drill steel in an arc back and forth to cut a slot. Slots of 600 mm depth were cut on Sallies at a rate of 1 m^2/h with a compressed air powered rockdrill. It was estimated that slotting rates could be doubled with oil-hydraulic rockdrills.

At the same time some laboratory experiments were carried out by the Chamber based on a new hydraulic rockdrill from Ingersoll Rand with an operating frequency of 150 Hz. However, this rockdrill was not reliable enough to conduct extensive tests.

All of this work was terminated when it was realized that mining methods based on slotting followed by several secondary activities to break and handle the rock could not meet the productivity targets.

2.5.4.3 Application of impacting to face mining with the swinghammer miner.

This principle was first experimented with by Anglo

Transvaal Consolidated Investment Company in 1969 and was developed by the Chamber until 1977. It was the first concept of a fully integrated mining system which could break the full stoping width, load the mined rock onto a face parallel conveyor, and convey the rock out of the stope into a strike gully. It therefore had the potential to overcome the labour productivity problems encountered with the rock cutter.

The impact mechanism was based on mechanically driven ballistic hammers. Power was supplied to an electrically driven drum to which hammers were hinged, which would strike the rock. The hinged configuration as shown in Figure 57 [67] prevented stalling of the rotor, should the rock not be broken on first impact. The hammer was kept in the striking position by centrifugal force. Rotors were built with four and six hammers, which had a blow energy from 500 to 950 J. The drum speed varied between 150 and 170 r/min. and the power output of the hammers varied between 7 and 11 kW. Two rows of hammers operated in parallel and the face was mined by cutting a web of between 100 and 150 mm in three different height positions as shown in Figure 58. Mining could be up-dip or down-dip.

Two different test sites were chosen, one at a depth of 2 000 m at the Hartebeestfontein and another at 3 000 m at East Rand Proprietary Mines, to study the effects of different rock fracture conditions.

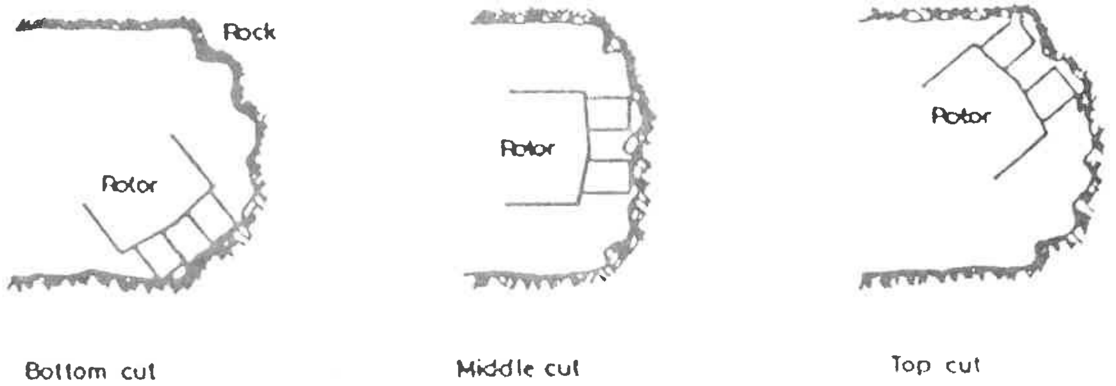


Figure 58. IMPACT POSITIONS USED WITH THE SWING HAMMER MINER

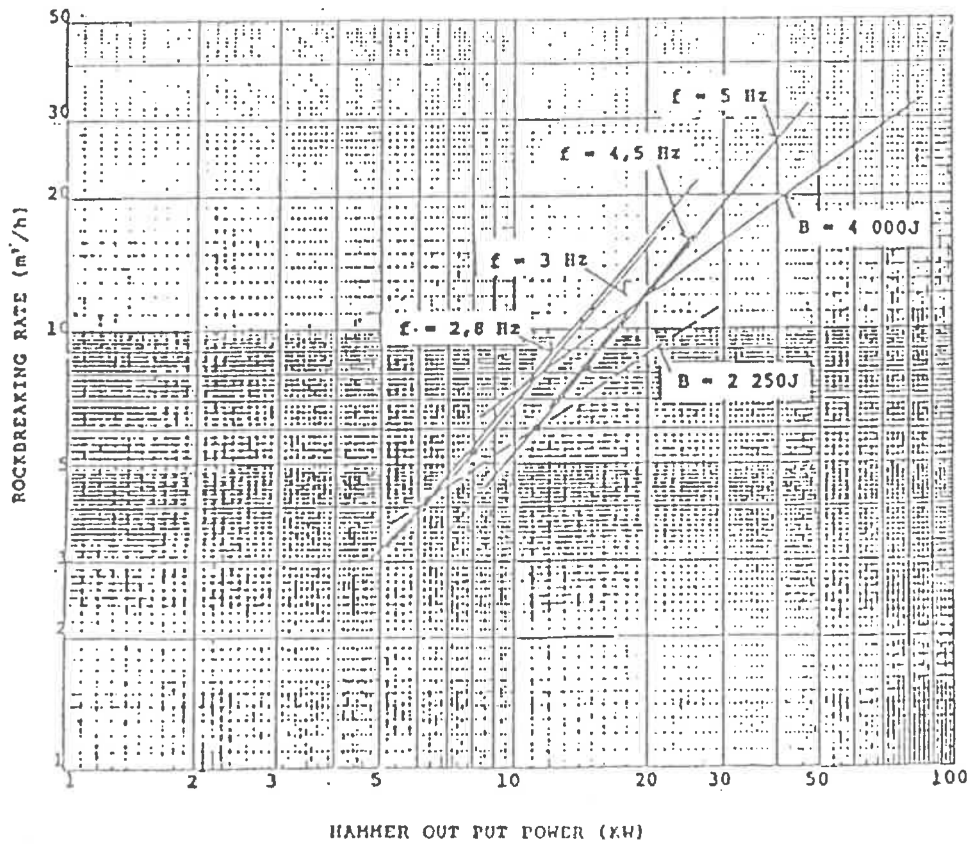


Figure 59. ROCKBREAKING RATES FOR IMPACT MINING MACHINES AS A FUNCTION OF HAMMER POWER, BLOW ENERGY AND FREQUENCY

The performance results are shown in Table 7, although it must be noted that the areas mined were very small. In the first case only 50 m^2 were mined and in the other case 220 m^2 and 155 m^2. The rockbreaking rates were very low resulting in high specific energies. This can partially be explained by the fact that low blow energies were used. The breaking action of this principle appears unfavourable resulting in a high number of tangential blows with very shallow attack angles. This causes high velocity skidding with heavy tool wear. Unfavourable attack angles became particularly evident in hard rock where the easy to break corners were removed more quickly, leaving very shallow inclined surfaces for the impactor. Contrary to expectations the tests at greater depth resulted in lower performance. This was caused by the occurrence of hard patches in the otherwise well fractured rock, which considerably reduced the average performance figures. Although the swinghammer miner worked very well as a loader, further work was discontinued in 1979 because of the fundamental problem of deteriorating attack angles as the rock became harder and because a solution for overcoming these problems was available with the impact ripper. Therefore the development of the swinghammer miner was discontinued in favour of the impact ripper.

TABLE 7 THE AVERAGE EXCAVATION RATES ACHIEVED IN E.R.P.M. AND HARTEBEESTFONTEIN

Description	E.R.R. Min	Max	Hammer blow energy (E_b) J	Impact power kW	No of impact paths	Excav-ation rate m^2/h	Total energy Et M/J/m^2
Tests in E.R.P.M. with 1st rotor	23	196	494	7,2	2	0,82	31,6
Tests in Hartebeest-fontein with the 1st rotor	3,9	8	494	7,2	2	1,22	21,3
Tests in Harbebeest-fontein with the 2nd rotor	3,9	9	690	6,7	4	1,39	17,4

2.5.4.4 Application of impacting to face mining with the impact planer

The impact planer was developed by the Chamber from 1978 to 1983 to overcome the problems of the deteriorating attack angle of the swinghammer miner. The impact planer had a constant attack angle to the face for the two ballistic hammers which operated in parallel to mine the rock very similarly to the swinghammer miner. The only manipulation was height adjustment to mine different parts of the face for subsequent runs along the face. Stand-off for the ballistic hammers was designed to be controlled manually with the driving speed.

At that time the question of whether high blow energy at low frequency, or low blow energy at high frequency, is more advantageous in rockbreaking was unresolved. This particular machine was designed for hammers with a low blow energy of 700 J but high frequency of 30 Hz resulting in a high power output of 20 kW for each hammer. With this high power a greater web depth of 250 mm was intended to be mined, which was also believed to take better advantage of the existing rock fractures.

The oil-hydraulic drive mechanism for the hammers was basically an oscillating spring/mass system driven by a special pump at resonant frequency. In a first version the hammers were designed to impact simultaneously, but this was subsequently changed to a counter reciprocating design to improve the internal efficiency.

Surface tests with this machine revealed several fundamental shortcomings associated with the hammer drive system; the output blow energy was far below target, the mechanization was thrown out of balance when rock was struck, and the machine suffered from severe destructive vibrations. Since the rectification of these problems would have required considerable additional resources, the concept was abandoned in 1983 and never tried underground.

The concept of planing was, however, revived in 1983, but this time with only one hammer and a high blow energy of 4000 J which was crowded up against the rock. Operation was intended to be automatic by controlling the speed by means of the rockbreaking rate. Although this was a promising concept, further work had to be shelved in 1984 because of a lack of resources which necessitated concentration on the then further advanced impact ripper.

2.5.4.5 Application of impacting to face mining with the impact ripper

Impact rippers are deep web mining machines which break out a section of rock from a stationary position before moving to the next position. They can attack the face as indicated in Figure 56 in either a face parallel direction (B) or a face perpendicular direction (C).

The development of the method of mining and the required hardware was a slow process because it was the first time that ripping had been used as a continuous face mining method and answers had to be found for several fundamental questions, such as how best to attack the rock face, how to control hanging and foot walls, and which blowenergy and frequency to use for most effective rockbreaking. Only after a period of extensive experimentation could tentative specifications for engineering developments be drawn up. Considerably effort subsequently went into the investigation of suitable manipulation systems for the hammer and methods for handling the broken rock. This was then followed by developing engineering solutions which would stand up to the environment. Full details of this are discussed in the section on Integrated Mechanized Systems.

Only after suitable methods had been developed and engineering hardware had reached a reasonable degree of reliability could meaningful performance tests on rockbreaking rates be conducted. Therefore results are given for the last four systems only, that have been evaluated. The results in Table 8 were achieved over fairly large areas from 600 m² to 1700 m² and should therefore be reasonably representative. The last system in Table 8 operated with face perpendicular impacting, whilst all the others operated with face parallel impacting.

HAMMER BLOW ENERGY KJ	HAMMER FREQUENCY Hz	ROCKBREAKING RATE m²/h
2250	2,8	4,3
2250	5	6
2700	3	5,3
3200	4,5	8,1

TABLE 8. **ROCKBREAKING RATES OF PREVIOUS MINING SYSTEMS AS A FUNCTION OF BLOW ENERGY AND FREQUENCY**

The given rockbreaking rates must be multiplied by the hammer utilization, which accounts for non-productive times like manipulating and machine haulage, and must furthermore be multiplied by the machine availability to determine the system mining rate. Efforts to improve both the hammer utilization and system availability are part of the development programme of impact ripping systems and are discussed under Integrated Mechanized Systems.

The rockbreaking rates shown in Table 8 were used to answer the question whether high blow energy or high frequency gives better rockbreaking performance for identical power input.

In an adaptation of a theory [61] which had some verification from field results, a diagram as per Figure 59 was developed [66]. This suggests that the rockbreaking rate may be related to blow energy and frequency as follows:

$$R \propto B^{1,2} \cdot f^{0,7}$$

The obvious conclusion, for a given power output of the impact hammer, is to increase the blow energy and to reduce the frequency. That this is principally correct can be derived from Figure 53. More data will be required to verify these preliminary findings.

The rockbreaking rate can also be used to determine the average specific energy for impact ripping. It was calculated as 2,5 MJ/m², if hard patches are excluded. This very low value confirms that impact ripping exploits the existing rock fractures better than any other method and has therefore the best potential to be developed into a successful non-explosive mining method.

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