The quality of stemming in assessing blasting efficiency

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THE UNIVERSITY OF NEW SOUTH WALES

SCHOOL OF MINES

THESIS FOR THE DEGREE OF

MASTER OF ENGINEERING

THE QUALITY OF STEMMING IN ASSESSING BLASTING EFFICIENCY

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ABSTRACT

The process of rock fragmentation by blasting has been used in the mining industry for the past century. Explosives are employed in the blasthole in a variety of ways to fracture the rock and dislodge the broken rock to enable excavation equipment to remove the rock as easily as possible. Explosives are extremely energetic substances in compact form and the utilisation of the energy released in an effective manner is the aim of all good mining operators. The energy released from the detonation of an explosive charge occurs in millisecond time frame and extracting useful work from this energy is of prime concern. A major portion of the explosive energy is in the form of gas energy and it is this portion that is said to perform the movement of the fragmented rock mass. The transmission of this gas energy into the confining medium in the blasthole will determine the success or otherwise of the blast performance.

A laboratory program was designed to investigate the properties of stemming materials and the effect of these stemming materials on the blast performance. Experimental equipment was also designed and built to measure some of the physical properties of the stemming column in an effort to simulate the stemming column under load. The results of these laboratory experiments were the basis of small scale blasting trials using concrete blocks as the confining medium for the explosive. Field study were undertaken to validate the findings of the laboratory studies, and production blasts with varying types of stemming confinement were monitored using high speed photography.

Effective containment of the explosive gases to do more useful work on the confining medium has shown to produce fine fragments which were thrown at a higher velocity. Generally the results showed that as the stemming material particle size decreased the resistance to ejection also decreased and the rule of thumb of one tenth the blasthole diameter as the particle size was found to be optimum.

Unstemmed blastholes can do some work on the confining medium but if the explosive gases are contained until the free face rock mass begins to crack and move, the energy released by the explosive can be utilised more efficiently. Stemming materials that have large interstices act as a relief valve to reduce the pressure on the stemming column and form an interlocking bridge to resist ejection from the blasthole.

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

NOMENCLATURE

air blast the noise, pressure, emitted from a blast

aspect ratio the ratio of particle dimensions

blast properties properties of the blast that have an effect on the final muckpile and for fragmentation

blasthole hole drilled in the rock mass to accommodate the explosive

blasting breakage of material by the use of explosives

bubble energy the gas energy of an explosive

burden movement movement of the rock mass between the explosive column and the free face

C-J plane plane defining the detonation process of an explosive

cast blasting the use of explosives to throw the broken rock mass during blasting

collar top of the blasthole at the open end

competency of stemming material stemming column that remains intact during the blasting process

competent structure reasonably strong and consistent geological structure

compressive strength strength of the rock in compression

confining medium the material into which the explosive is placed or that into which the blasthole is drilled

crack network the inherent or blast induced cracks in the rock mass

detonation pressure the pressure generated at the reaction front as an explosive detonates

detonation a chemical reaction occurring at super sonic speed

discontinuity breaks or cracks in the rock mass

dynamic response movement under high pressure loading

dynamic properties properties relating to movement

emulsions a stabilised mixture of ammonium nitrate liquor and fuel oil by the use of surfactants

explosive a chemical which reacts at supersonic speed

FEL front end loader excavation equipment

flyrock uncontrolled throw of rock fragments

fragmentation small pieces of rock produced by blasting

frit a porous disk with holes in the micron range

gangue the waste material covering minerals, coal etc.

gas release rate the rate at which the explosive gases are vented to the atmosphere

gas pressure pressure of the explosive gases

gas confinement keeping the explosive gases in the ground to do useful work

gigapascal 1,000,000,000 Pascals pressure

high speed photography cinematic photography at 500 frames per second during this research work

hydrodynamic theory the study of the flow of particulate material

interstices the spaces between individual grains in a column

jig-saw effect the interlocking between individual grains in a column environment

K_{50} the quadratic mesh size where 50% of the material of certain dimension will pass

kinematics the study of particle movement under the influence of gravity

locking effect the forcing of grains into the interstices in a column under a high pressure

muckpile the pile of broken rock mass thrown after the blast

nitroglycerine is a nitric acid ester glyceryl trinitrate

orifice a hole above the blasthole to control the venting of the explosive gases to the atmosphere

packing density the mass per unit volume of a column of particulate material

PETN a high explosive PentaErythritolTetraNitrate

picaseconds 0.000000000001 seconds

porosity gaps between grains in a column of particles

primer a detonator sensitive explosive that initiates a non-detonator sensitive explosive

reflected stress return stress wave from a density discontinuity

relief valve releasing of gases to the atmosphere through the stemming porosity

scabbing the break away of surface slabs of rock at the free face

shock wave the initial pressure wave produced by a supersonic chemical reaction

stemming inert material placed on top of the explosive column to create confinement

tensional stresses the pulling apart of the rock mass as distinct from crushing the rock mass

throw to propel the broken material during the explosion

velocity of detonation the speed of the chemical reaction of an explosive

venting to atmosphere a mode of explosive gas escaping to atmosphere

VOD acronym for Velocity Of Detonation

voidage the overall volume of free space in a column of particles

water gels a stabilised mixture of ammonium nitrate liquor and fuel oil by the use of gelling agents

CHAPTER 1

INTRODUCTION

The extraction of valuable minerals from the earth's crust is accomplished by a variety of means. These minerals can be close to the surface or buried deep underground and in both cases the host rock is usually fairly competent and requires breaking or fragmenting before extraction can be accomplished. If the mineral is relatively weak, such as coal, in some instances mechanical mining methods can be used to excavate it. Here an array of picks are dragged or rotated through the mineral to loosen and fracture the mineral then a shovel or scoop is used to pick up the lumps for transportation to the stockpile or processing plant. Other means of fragmentation have been used successfully in the past and still exist today such as manual pick and shovel (for exploiting precious stones etc.) and hydraulic mining (requires an abundant water supply).

However, as mankind's ever increasing need for raw materials continued the extraction rates of the raw materials also needed to increase. Towards the later half of the 19th century the discovery of nitroglycerine and its use in rock fragmentation accelerated the extraction rate of raw materials from both underground and open cut mines throughout the world. Because of the large amount of energy released by a relatively small quantity of explosive substance the confining material absorbed this energy and thus fracture or crack networks occurred. This crack network gave rise to fragments which were easier, because of their size, to transport to the next stage of the winning process.

In the early days of blasting it was found, possibly by trial and error, that drilling a hole and placing the explosive deep in the ore body would cause more rock to be fractured and released. Eventually it was found that covering the explosive with the drill cuttings or some other inert material caused more fragmentation and ore release. Thus the process of stemming the blastholes evolved and this was possibly as a matter of safety in underground coal mines where the chance of fires from dust or gas explosions would have been of paramount concern. Over the years the role of the stemming material has been investigated and its ability to contain the explosive gases to obtain a more efficient blasting process have been shown. Other functions of stemming are to reduce noise and fly-rock.

Many operations stem the blastholes with drill cuttings because they are easily available and are cheap. However, depending on the rock structure a more resistance to ejection material is sometimes needed if fragmentation in the upper part of the borehole is important. In these cases crushed rock is often used. In underground coal mines the practice of stemming is legalised and materials such as water with dissolved salts is used to eliminate the risk of fires caused by sparks from explosions. Some mechanical devices such as plugs and stoppers have been tried with varying degrees of success to contain the explosive forces to get more "bang for your buck".

1 ROCK FRAGMENTATION BY BLASTING

In the early days before explosives had been discovered the winning of minerals from the ore vein was accomplished by pick and shovel, wedges and thermal shock. The first two processes relied on existing crack networks or weaknesses in the ore vein being able to be opened up with the pick or wedge and lumps of ore being forced out of the ore vein. It was found that thermal shock placed stresses on the rock structure resulting in an extension of the existing crack network and eventually "scabbing" of the surface. All three of the processes basically rely on the inherent crack network in the ore vein to release the ore. It is also only the exposed surface which extends back into the solid to about 1 metre that is of interest in these processes, and any large crack was quickly exploited for its ease of winning the ore.

Gunpowder was possibly the first explosive used in a mining application and its ability to extend the crack network was seized upon with enthusiasm. The ability to remove more ore for less work was probably one of the reasons the industrial revolution occurred in the 16th century.

In the 19th century when nitroglycerine was discovered and safe initiating devices were developed the potential for rock excavation by blasting was seen as enormous at that time. The modern accepted practise had been developed and since then only refinements and blasting efficiency has improved.

Rock fragmentation by blasting has been investigated by many workers over the past few decades and the general theory of blasting has developed over this time. Chiappetta, 1989 breaks the process into four separate phases, with each phase being critical to the overall blasting process. These four phases are:-

T1. The initiation phase where the oxidant and the fuel of the explosive mixture are forced together to begin the chemical reaction.

T2. The shock wave generation phase where the explosive reaction rate sets up a shock wave which is transmitted to and through the confining medium. A crack network is formed.

T3. The gas pressurisation phase where the gaseous products of the reaction are at extremely high temperature (above 3000°K) and extremely high pressure (above 10Kbar). Localised compression of the confining medium begins and gas dissipates into the crack network.

T4. The confining medium movement phase where the force of the pressurised gases is transmitted to the fractured confining medium, and tensile and compression forces result in fragmentation and movement.

If we look at the first phase, initiation, when the booster or detonator is initiated a shock wave or high pressure impulse force is generated. This force overcomes the molecular forces holding the explosive ingredients in a stable form at atmospheric conditions. These molecular bonds are then broken and the oxidant and fuel components are forced together on an atomic scale to form compounds that are more stable at the reaction conditions. With the formation of these new compounds and a change in the overall system conditions, large amounts of energy are released. This energy manifests itself in the form of heat and pressure to maintain the reaction front and drive it in the direction of unreacted material. Chapman, 1889 and Jouguet, 1901 very early in the 20th century postulated that a reaction zone occurs where the reactants are brought in contact with each other. This reaction zone is bounded by the reaction front and a plane known as the C-J plane. This reaction zone is very "narrow" for ideal explosives and "wider" for most industrial explosives which explains the slower velocity of detonation of industrial explosives.

After the chemical reaction has begun the second phase, shock wave generation, occurs. The shock wave radiates from the explosive charge in all directions and travels at a constant velocity through the material (note: this velocity is material dependent). This shock wave, compressional in nature, causes differential movement within the material and although particle movement is small, minute cracks begin to form and existing cracks are exaggerated. At this point in the blasting process only the shock wave is acting on the confining medium. The shock wave travels through the medium at a velocity approaching the speed of sound and soon comes to a free surface. At this interface a major change in density occurs and the shock wave is partly reflected in the reverse direction in a tensile nature. This new tensile stress can be greater than the tensile strength of the confining medium and the medium can fail (note: tensile strength of rock is often 1/10th of the compressive strength). If the distance between the explosive charge and the free surface is small enough the tensile stress energy can be high enough to form "scabs" at the surface which spall and are dislocated at the free surface.

The energy in the shock wave decreases with distance from the explosive source. As the confining medium in mine blasting is anything but competent in structure the continual compressive and tensile application on this material causes major structural changes. Inherent crack networks exist before blasting occurs and as these cracks act as concentration points for the energy in these waves further crack growth occurs. Field and Ladegaard-Pedersen, 1972 discuss the crack network set up in plexiglass and show a significant difference in the crack growth due to the reflected stress wave from the free surface. As only 5-15 percent of the explosive energy appears in the stress wave; the reflected wave can not be regarded as a major driving force, but it is significant. When a crack grows longer than other cracks it is quickly extended, in preference, because for equal stress at the root of the crack, a longer crack is extended and accelerates more easily. Interaction between cracks and stress waves from adjacent cracks caused between 4-8 major cracks to appear. As would be expected major cracking and deformation occur very close to the explosive source.

So from the stress wave a crack regime has been generated with inherent crack elongation and new crack formation occurring. The next phase of the blasting process, gas pressurisation, begins and during this phase the major part of fragmentation is enhanced and accomplished. The explosive reaction occurs at extremely high temperature and pressure and the products of the reaction are usually all gaseous. A typical carbon-hydrogen-nitrogen-oxygen explosive mixture will produce such gases and vapours as carbon dioxide, nitrogen and water vapour under ideal conditions. These gases at normal temperatures and pressures would occupy a volume approaching 1000 times the volume of the explosive mixture. When the elevated temperature is taken into account this volume can be up to 100 fold again. The detonation pressure, a characteristic of explosives, is the pressure at the reaction front driving the reaction forward. However, the pressure exerted on the confining medium, the borehole pressure, is a function of the high temperature and pressure gaseous products of the reaction. It is this gas pressurisation that causes the majority of the fragmentation and movement of the confining medium.

Industrial explosives today are prepared to cater for a wide range of applications. An explosive that performs well at one mine site might fail completely at another. Approximately 80 percent of the explosive energy is manifested as the gas energy to perform the final fragmentation and movement of the burden material. The degree of fragmentation or the coarseness of the final fragments is related to the confining material properties and this "surplus energy" in the explosion. If, for example, the confining material has a compressive strength of 200MPa then as the gas pressure falls below this value no further damage due to compression will occur. But induced cracks can still fragment after the pressure drops below the confining material's compressive strength due to stress relaxation and in flight collisions. Many computer modelling packages that predict fragmentation and muckpile shape have a cut-off point below which any remaining gas pressure will have no effect on the predicted result. It is postulated that below 100MPa the pressurised gas does not have enough energy to cause any damage and simply dissipates through the crack network to be vented to the atmosphere.

The final phase of the blasting process, confining material movement, is accomplished as a result of the gas pressurisation within the confining medium. It is this phase of the process where explosive location and initiation sequence have an important effect on the final outcome. As the gas is forced into the cracks and the pressure build up forces open these cracks fragmentation and movement begins. The gas pressure dissipates throughout the crack network and applies a pressure in all directions on the confining material. When this pressure is higher than the strength of the confining material, movement begins in the least resistive direction. The pressure exerted by the explosive gas products applies a force over the surface area of the crack network. This force is a product of mass and acceleration. Thus when the applied force is greater than the confining material strength an acceleration is applied to the confining material which causes movement.

The distance between the explosive charge and the free surface will determine which way the fragmented material will move and basically how far it will be thrown. This distance is called the burden. In surface mining the burden is controlled by drilling the blastholes at an optimum distance from the free face to achieve the required muckpile configuration. In underground mining as only one free surface exists (ie. the vertical face) a series of relief holes are drilled to create extra free surfaces. When multi-row blast patterns are fired, the time between individual holes initiating is designed to produce a new free surface. The timing between holes or rows of holes is in the order of tens of milliseconds for surface blasting and seconds for underground blasting. In surface blasting this delay time has to be long enough for the burden to begin to move and detach from the confining material mass before the next hole is initiated. The type of muckpile configuration required will define some of the firing sequence parameters to control the direction in which the fragmented material is to move.

In underground blasting the fragmented material has to be moved in two directions at right angles to each other to place the broken material in the position required. Thus the need for larger delay times before subsequent hole initiation. The fragmented material in underground blasting has to be first moved to the free surface of the relief holes and then out of the relief holes before the next hole is initiated otherwise clogging and possible hanging wall or pillar damage will result.

The crack network set up by the confining material's inherent structure and the shock phase of the explosive process does not produce the fragments of the final size from the blasting process. As the confining material begins to move and this usually is in an upward and expanding direction individual fragments collide with other fragments causing further reduction in size at existing internal crack surfaces within the fragment. This process can occur many times during the flight of the fragments from time zero to the final resting position of the muckpile.

1.2 STEMMING AND ITS PROPERTIES

The generally accepted theory of the blasting process can be used to describe the events that occur in fragmentation by blasting. However, the energy and the time frame over which this energy partition takes place necessitates a means to confine this energy release in order that useful work is carried out. The effect of the intense shock wave in producing a crack regime and the gas pressurisation and movement of the fragmented material has been discussed above. This energy released will take the line of least resistance to the lower pressure of the atmosphere, ie. venting of the gases to the atmosphere. The role of the stemming material is to minimise the reduction in this energy level until the confining material has begun to move when venting to the atmosphere will occur through cracks in the burden.

A typical blasthole configuration consists of a hole drilled in the confining material to the desired depth and correct location. An initiating device and the explosive are then placed in the hole at the required level and the stemming material is then placed on top of the explosive. If no stemming material is placed on top of the explosive charge to confine the energy released to do useful work, noise, fly-rock and inefficient use of the released energy will result. The explosive is detonated and the burden material is moved in the designed direction. Although the detonation process is nearly instantaneous the effect on the confining material takes some finite time before, for example, movement begins. High speed photography at framing rates of 500 frames per second has been used in open cut coal mines to examine the burden movement times and quantify their effect. In general, burden movement times in the order of 10 milliseconds per metre of burden, BHP Research, 1989 have been observed. It is very difficult to be more precise than to postulate "rules of thumb" as these burden movement times are very site specific and even differ from one blast pattern to another. The main controlling factor for burden movement given similar explosive conditions is the structure of the ground and the effect the explosive pressure has on this ground structure.

When an explosive is initiated two types of energy are formed, ie. shock energy and gas energy. Both of these energies "react" with/on the confining material to perform useful work in fragmentation and burden movement. If the blasthole is left open above the explosive charge, the above parameters (shock energy, gas energy etc.) will be ineffective. The gas pressure will be released prematurely to the atmosphere resulting in a waste of energy input into the fragmentation process. Although fragmentation and movement will occur, the desired result will be far from acceptable causing very little material movement and large fragments which could necessitate the need for further secondary blasting.

Accompanied with inefficient blasting when no stemming is used is the environmental aspect of the blast, noise or blast over pressure. One way the energy manifests itself in the blasting process in the atmosphere is noise or blast over pressure which can be a potential cause for concern when neighbours are in close proximity to the blasting operation. If this noise can be reduced it can add to the total useful work done by the explosion in fragmentation and movement.

When stemming material is placed on top of the explosive in the blasthole it in effect seals off the explosive gases from the atmosphere thus absorbing energy in the confining material before escape to the atmosphere is possible. The quantity or height of stemming material placed on top of the explosive is important to the final outcome of the blast. If the explosive charge is buried deep in the ground and the distance to a free surface is large, near charge pulverisation and cracking will be the only result. Some surface heave or movement could be evident if the charge is close to the surface. When the burden distance is selected to optimise movement of the confining material and the stemming height is the same or a little higher the explosive gases are trapped in the confining material long enough to perform useful work. This useful work will manifest itself in the form of optimum fragmentation and movement, and the final muckpile will be ideal for the excavation equipment being used.

1.3 TYPES OF STEMMING

Stemming has been described as an inert material placed on top of an explosive charge to contain the product gases from the explosive reaction. The materials used as stemming are many and varied, and in most cases depends on the local availability and cost. The basic purpose of stemming is to retain the gaseous products from the explosion until the confining material begins to move thus allowing escape to the atmosphere. Once the burden has cracked and moved and the gases begin to escape through these cracks then the stemming has performed its purpose. As stated above the types of stemming are many and varied and some workers have argued, Zaburunov, 1990 that refilling the blasthole with inert material is a wasteful procedure and have put forward the use of stemming plugs. Below is a discussion of the types of stemming materials used today in blasting operations throughout the world.

Drill Cuttings. When a blasthole is drilled in the ground small particles (<15mm) are broken away and pumped to the surface and distributed in a cone around the collar of the blasthole. These drill cuttings when used as stemming material provide a most cost effective and convenient material. A majority of surface mining operations employ this material as stemming. Even though for some large blastholes the particle size can be small, ejection from the blasthole is minimal if enough, height, of the material is used. However when water is present to the collar of the blasthole this stemming can become fluid and ejection from the blasthole can be of some concern.

Crushed Aggregate. In some blasting applications control of noise and flyrock are of prime importance because of local environmental concerns. Crushed aggregate or crushed rock in sizes up to 25mm is used in these situation as it has been observed that this type of rock causes a "locking" action and holds the stemming in place for the entire time of the blasting process. The size of the crushed aggregate used is very much dependent on the blasthole diameter. This type of material, because of its angular shape, is very good in blastholes that are full of water.

River Gravel. This type of material is the same as crushed aggregate except that the individual particles are rounded due to the eroding action of the water over the river bed. The individual particles come in many shapes which affords excellent "locking" characteristics in the blasting process. This type of stemming is also ideal in wet holes and where noise and flyrock is considered a problem.

Water Ampoules. Water itself is not recommended to successfully confine the explosive gases because of its fluid nature. When water is packaged in ampoules or cartridges, in many different sizes, it can be used as an effective stemming material. Water ampoules have been used in underground coal blasting to minimise the risk of blasting induced fires and also to reduce the dust levels from blasting. The use of ampoules or cartridges allows the material to be tamped or packed tight in the collar of the blasthole providing more resistance for the explosive gases to overcome before ejection becomes a problem.

Clay, Powders and Pastes. As a dry material clay would fit into the drill cuttings category but when mixed with small quantities of water its cohesive properties increase and resistance to ejection increases. Clay and other materials mixed with water to form thick pastes and even packed in ampoules or cartridges has been used as stemming material. This type of stemming material is tamped or packed into the collar area of the blasthole and in effect completely fills the blasthole above the explosive. When it is firm against the blasthole wall it resists the force of the explosive gases long enough for fragmentation and movement of the burden to be accomplished.

Wooden Plugs. In some underground small diameter blastholes a wooden plug or wedge is forced into the blasthole collar to effectively seal off the blasthole. The plug offers resistance to the premature escape of the gaseous products before effective work has been carried out. In small operations this practice can be effective and the cost can be absorbed, but in larger underground mines the cost of the wooden plugs can, in some cases, be excessively high. There is also the risk of damage to the initiator (in hole detonator) downlines if caught between the wooden plug and the blasthole wall.

Inverted Cones. Other mechanical devices have been used to stem the flow of the explosive gases and one device that has also been patented for this application is an inverted cone. The device consists of a cardboard or thin metal cone lowered into the blasthole so the open base of the cone rests on top of the explosive charge. When the explosive is detonated the gas pressure acts on the inside of the cone, spreading the cone against the blasthole wall and retaining the gases in the blasthole long enough for useful work to be done. In large scale operations this device can add considerable cost to the overall operation and for this reason has not proved very popular.

Tamped Explosive Charge. Underground operations where small diameter blastholes are used (approximately 50mm diameter) sometimes employ no stemming but tamp the charge into the blasthole. Cartridged explosives and blow loaded bulk products have been used in these cases with the cartridges being tamped, with a wooden stick, to force the charge tight against the blasthole wall. The final two cartridges in each blasthole are tamped or rammed very firmly into the blasthole to effect a good seal to prevent the premature escape of the explosive gases. This practice is justified by the fact that no time is used to "stem" the blastholes with inert materials although blasting efficiency is often questioned.

1.4 AN OVERVIEW

The literature reviewed has shown how the blasting process occurs and the effect of these forces evolved. Crack networks have been shown to develop independently of gas pressurisation and the extension of existing discontinuities plays an important role in fracturing the confining medium. The importance of stemming the blasthole has been shown by many workers and the effect of various properties has been reported. It is the efficiency or transmission of the blasting forces to the confining medium and the resultant effects that will be reported in this thesis. Laboratory investigations into the properties of stemming materials on the locking in the blasthole as well as the implementation of the properties in small scale laboratory trials will be discussed in detail. Field trials with high speed photographic monitoring of a quarry blast will be used to quantify the quality of stemming in assessing blasting efficiency.

CHAPTER 2

LITERATURE REVIEW

Explosives are often used to provide a convenient and controllable means of exposing minerals or rock which are usually covered by thick layers of overburden or gangue. This thick layer of overburden or gangue has to be loosened to facilitate its removal so the valuable mineral can be easily exposed and excavated for further treatment.

When an explosive is detonated a large amount of energy by way of a shock wave and gas pressurisation is imparted in a very short period of time. It is generally thought the shock wave enhances and reinforces the inherent crack network in the surrounding media and the gas pressurisation extends and opens up this network to form the initial fragmentation. An explosion generates a large volume of gas, compared to the volume occupied by the original explosive, at an elevated temperature, the associated force on the surrounding medium imparts motion to the surrounding medium in a direction of least resistance. The stemming of an explosive in the blasthole plays an important role in blocking the easy path for the explosive energy released. Little or no stemming can result in premature release of the explosive gases before movement occurs whereas too much stemming means less explosive column which can result in poor fragmentation and this may necessitate secondary blasting.

2.1 GAS PRESSURISATION AND FRAGMENTATION

When the earth's crust began to form many eons ago and as time has progressed since the beginning of the earth, the formation of ore bodies or rock structures on a global basis usually occurred in a layer structure. The earth's crust rests on a liquid core and as such is in continual movement. Extremely high internal stresses are placed on localised areas within the earth's crust by this continual movement and these stresses have formed minute crack networks within these relatively thin layers. Sometimes the movement can be large resulting in a complete break or fault in the layer of material.

Britton et. al., 1989 reported that when an explosion is detonated the generated gases provide a force which is exerted on the surrounding medium. In this work the measured shock energy in the confining medium was found to decrease as the explosive decoupling ratio increased. A decrease in the crater weight was also measured for a decrease in decoupling ratio. They concluded by stating "the ever increasing evidence points to the rapid expansion of gases as the primary mover of rock". A recently published work by the author with others (Armstrong et. al., 1993) also substantiates this statment.

The effects of shock and gas penetrations in blasting was investigated by

Brinkmann, 1990 where he monitored production sized underground holes with steel liners to separate the shock and gas energies. Brinkmann reports that explosives of larger particles "have slower reaction rates, deliver less shock energy and greater heave energy than the finer grained explosives such as emulsions". He also found that different explosives give different blasting results ie. "break out is controlled by gas pressurisation and fragment sizes are governed by shock" and that stemming plays a major role in containing the gases in the blasthole. Stemming the blasthole can reduce energy loss by at least 50%.

The partitioning of the energy between shock energy and gas energy has been reported by many workers, Brinkmann, 1990; Zakharova, 1983; and Udy and Lownds, 1990. For competent structures typically deep underground, Brinkmann 1990 reports that "fragment sizes are governed by shock" which has been the line of thinking for many years. The shock energy, reported to travel at speeds of 2000 m/sec, arrives at a particular point, in the first instance, forming a crack network which is later enhanced by the arrival of the high pressure gases. However, there is no distinct separation between the function of both energy parts, and Udy and Lownds 1990 report on the effects of the reaction rate of non-ideal explosives. Decreasing the release rate of the energy or "post-CJ release of energy" was reported to be beneficial in some cases and "the observation of low VOD clearly suggests slow reaction of part of the explosion composition" which does not necessarily mean that energy is wasted but may be directed to a later stage of the blasting process ie. heave or throw. Udy and Lownds also report on a predictive model to determine critical diameters and

VOD's. Ideal behaviour is not necessarily the optimum in all blasting situations.

The effect of the detonation properties was investigated by Baranov and Kovalenko, 1975 where they took a theoretical approach to the topic. Baranov and Kovalenko found the effect of the detonation properties are "very pronounced at distances (1-10)R_0 in the so-called local zone of the explosive". They continue by saying "the effects decrease and at about 50R_0 levels out as regards their detonation characteristics". Discussion is centred about the energy transfer from the explosive products to the stress wave. They conclude by saying "the fracture of weak rocks by powerful explosives (high shock) is inefficient and the residual energy in the explosive gases provides the forces required to move the fractured rock".

Blasting theory and the progress of the breakage process as put forward by Chiappetta, 1989 discuss the events as an explosive detonates. Gas pressurisation is the third step in the process and he states "high pressure and high temperature gases impart a stress field around the blasthole that can expand the original bore hole, extend radial cracks and jet into any discontinuity". The time for the gas pressure to provide surface movement ie. confinement times, have been measured from 5 to 110 milliseconds after the blast. The variation in these times is a function of the confinement of the surrounding medium, its state of fracture and the rate of travel of the shock wave through this confining medium.

The question must be asked of any energy process, "Is the energy being used efficiently?" Many papers have been written about the chemical composition and energy output from explosive compositions. Hardesty and Kennedy, 1977 looked at the effective specific energy of an explosion particularly related to metal particle acceleration. The model they used was one derived by Gurney in 1943 which partitions the energy "between the kinetic energies of the metal and the products in a manner dictated by an assumed linear velocity profile in the products". A thermochemical code, TIGER, was used to calculate the properties in the C-J plane for various explosives and a good agreement was found between measured and calculated values. They conclude by saying "the ballistic performance of a high explosive can most appropriately be expressed as a specific energy" which in effect is the work done by the explosive, in this case, the bullet.

For many years parameters such as energy strength, bubble energy etc. have been used to rate commercial explosives. These terms have been used in a multitude of ways and in some cases have led to confusion. Many test methods are used to rate the performance of explosives at the conditions of the test method employed. A paper by Mohanty, 1981 looks at the practical aspects of explosive performance and shows "that no single technique alone....can lead to realistic rating of explosives in terms of expected blast results". In this paper he looks at energy and computer code calculations, ballistic mortar tests and trauzl lead test and explosive strength, cylinder tests and underwater tests to quantify explosive performance. He then discusses mechanics of rock blasting which is often over looked by laboratory and computer studies when applied to blasting operations. He concludes by saying "There is a definite need for a two-tiered rating of explosives, one based on absolute energy potential and the other based on blasting performance".

A paper by Kristiansen et al., 1990 looked at the practical aspects of fragmentation and heave related to the explosive properties. They discussed the similar test mentioned by Mohanty and the practicability of these rating tests. They looked at explosive composition and gaseous products formed and these effects on fragmentation size and fragmentation velocity using concrete blocks as the modelling material. They found that "the performance of heterogeneous explosives is very much affected by the properties of the surrounding medium". In a practical sense one type of explosive is not necessarily good for all types of rock blasting.

The quantification of the partition of the energy released from an explosion into shock energy and gas or heave energy has long been a point of discussion in the blasting field. Many tests and models have been used to quantify the energy of explosions and an approach by Cole, 1948 looked at what is now termed the "bubble energy" of commercial explosives. Cole discusses the sequence of events from the shock wave to the actual bubble formed by the product gases. He shows the oscillation of the diameter of the gas bubble is "related to the internal energy of the gas and the hydrostatic pressure, being proportional to the cube root of energy and inverse five sixths power of pressure". This approach has been investigated over many years and a standard procedure called the "Pond Test" has been developed. A paper by Cameron and Torrance, 1990 discusses in detail the set up of a pond test procedure and the calibration and measurement techniques needed to measure the "bubble energy". They conclude by saying "This technique provides valuable data for the assessment of product consistency and also for evaluating new product formulations".

2.2 CRACK GENERATION AND PROPAGATION

The modern theory of detonation of explosives, Chiappetta, 1989 breaks down the process into 4 separate stages (see page 4). About the shock wave propagation he says "The pressure next to the borehole wall will rise instantaneously to its peak and then rapidly decay exponentially". Chiappetta discusses the stress wave propagation and the tensile (hoop stress) and compressional forces involved. He also discusses the compressional stress wave and the transfer of energy across discontinuities. He concludes by saying "The interaction of stress waves in the outgoing compression and tensile modes around discontinuities and flows....is an area of intense research...".

Cook, 1971 in his classic works on high explosives looked at the properties of solids under explosive attack. He discusses, in much detail, the fundamental principles of materials under the high pressure and high temperature regime experienced in an explosive environment. Being able to predict the properties at any instant in time depends on the equation of state used, which is discussed in depth. His work and that of others presented here is centred around the properties and effects of the explosion and the "internal" reaction that is occurring.

Another classic text by Johansson and Persson, 1970 discusses the "various mechanisms and phenomena of initiation and detonation of high explosives". However, they also looked into the mechanical effects in the surrounding media and the shock waves produced by the detonation process. In elastic solids the stress wave propagation "moves back and forth as a compression in one and a tensile wave in the other direction". Much work has been done to measure the velocity of the shock front associated particle and gas front velocity at distances close to the blasthole. They find the cracking extensive after the disappearance of the shock wave "is dominated by the extension of the radial cracks towards the free surface under the influence of the quasi-stationary stress field, due to the remaining pressure of the gas in the cracks".

The propagation of stress waves in solids is treated in great detail by Kolsky, 1953 where he looks at the stress on elastic material from a first principle basis. Kolsky discusses the surface or Rayleigh wave formation and also the reflection of elastic waves at a free surface. He discusses the "reflection of the compression pulse at a free boundary" and talks about the fractures produced by stress waves. The formation of "scabs" as the stress waves reaches material of lower density aids in the fracturing of the confining medium. The fracture patterns produced by different explosive location is also discussed.

The growth of a crack is discussed by Nilsson, 1984 where he looks at the stress intensity factor K_I of the crack tip. "When K_I reaches a critical value, K_{Ic}, crack growth is initiated " and the effect of rapid dynamic loading on this growth is discussed. He says the process is non-linear "in the vicinity of the tip" and the non-linearity disappears "as soon as the crack tip starts to move". One criterion for dynamic crack growth studied is the critical energy flow which he shows in many cases to be zero "thus precluding the use of an energy criterion". His work has shown that most materials do not behave elastically in the vicinity of the crack tip and his models assume small scale yielding with prior material "information about the surrounding material...through the stress-intensity factor". Experimental studies are reported with various notched specimen configuration and his conclusions are that geometry independence of K_{Id} has not been verified and the significance of "testing details may exist".

"All fracture is dynamic by nature..." is the beginning of a paper by Curran et al., 1986 where they look at modelling the formation of cracks on both a microscopic (the stress force in the crack model) and a macroscopic scale (the fracture mechanics model). They discuss the microstructural failure "by the nucleation, growth and coalescence of many voids or cracks" by looking at the grain structure of rolled steel. They continue by saying that a flaw begins at a submicroscopic level when the material no longer appears as a continuum. "In short, the damage kinetics interact with the load kinetics to make failure of a solid a highly rate, and history, dependent non-linear process".

In a paper by Field and Ladegaard-Pedersen, 1971 they discuss in detail the effect of the reflected stress wave on crack propagation. Their conclusions are that the reflected waves are of prime importance in determining both which fractures develop and in which directions they travel "but as only 5-15 per cent of the energy is in the stress wave the reflected stress wave assists the cracks at a critical stage".

A more practical study was undertaken by Fourney et al., 1988 into crater blasting as applied to vertical modified in-situ retorts in an oil shale research program. In their study they used granite, hydrostone and plexiglass blocks of approximately $0.03m^3$ in volume and explosive charge weights of 0.8g of PETN. The crack pattern or discontinuities formed by these experiments are similar to those reported by Field and Ladegaard-Pedersen. One of their biggest problems was the scale up from these low charge weights to predict with confidence the field behaviour. On their stemming studies they conclude "that the use of stemming has very little effect on the fracture or fragmentation results obtained in a single hole crater blasting situation" which was supported by field trials. They also found that for "homogenous isotropic materials radial cracks are formed before the crater formation process begins" which implies that the "mechanism of crater formation is heavily dependent upon stress wave effects and depends only in a minor way upon borehole pressurisation".

2.3 THROW

The next stage in the blasting theory process is the transfer of the energy from shock and fragmentation of the confining medium to moving the broken material in directions as per the blast design.

Some of the parameters that affect the throw of material are discussed in a paper by Hagan and Just, 1974. Blast geometry and initiation include a range of variables which can be controlled by the operator. This is a valid statement and many blasting engineers are always looking for ways to improve their blasting practices. Initiation sequence is one way of controlling the throw and the type of excavating equipment will define the type of muckpile required by the mine operator. Changing from row firing to delay firing "will cause much of the bubble energy conserved by the reduction in throw to be used in vertically shearing the burden". However, in some timing configurations "there is considerable throw but little fragmentation is caused by in-flight collisions or by vertical shearing". In ideal multi-row bench blasting the burden should be moving well before the next row fires, but in practice as the number of rows increases the back rows tend to have vertical movement resulting in "flyrock and tight bottoms (hard toe)".

After the passage of the shock wave the resultant force applied to the confining medium of a blast will cause the fractured material to move. If confinement is minimal ie. a small burden, high particle velocities will be experienced and fly rock and face bursting will result. Some mining operations want a tight but easy to dig muckpile. A paper by Kristiansen et al., 1990 discusses fragmentation and heave. They measure the heave velocity (velocity of fragmented pieces) in a laboratory apparatus and found a good relationship with "the calculated final state pressure". They conclude by saying "The reliability of this relationship is very much dependent on the accuracy in the calculation of the final state pressure".

The main purpose of throwing the fragmented material is to produce the desired muckpile for a particular excavation equipment being used. Over the past decade muckpile shape predication modelling has been investigated extensively and many models now have good agreement with actual shapes. A paper by Preece and Taylor, 1990 details a new code to model the motion of particles resulting from a blast. This code uses "spherical element, explicit time integration and a new contact search algorithm" together with "physically correct mechanisms for capturing the bulking behaviour of the rock mass". In their work they take into account physical parameters such as aspect ratio, friction and rotation of the particles in flight. The output from their modelling was directed at crater blasts and shows velocity profiles and final muckpile shape to be in good agreement with field measured data. They point out that some deviation from field data could result from "treating the gas pressure in a very simple manner" and that more sophistication in this area is required.

A paper by Postupack, 1989 takes a more practical view on the economics of overburden casting. In his paper he talks of "using chemical energy to supplement mechanical energy" to move overburden material to its final resting place. Some of the factors which should be considered in overburden casting are bench height (>10 metres), pit width (1.5*bench height) and delay timing (entire row). He points out that powder factors must be increased so some form of energy is needed to put the rock in its final position and a trade of mechanical energy for chemical energy is needed.

2.4 STEMMING AND ITS PROPERTIES

Stemming has been investigated over a long period of time and two papers by Taylor, 1938 and 1945 looked at the static and dynamic behaviour. In his first paper on the static behaviour he looked at "some general laws on the behaviour of stemming" and said not only the weight or inertial effects but such properties as friction and cohesion must also be considered. His work is mainly concerned with coal mining and in particular coal blasts and he segregates the stemming into plastic and granular materials. He built an apparatus to simulate a blasthole made of steel with a hydraulic press attached to one end. The stemming material was packed in the tube and the pressure at which the stemming began to move was measured. His basic conclusions are that plastic "stemming materials have a resistance directly proportional to its length" and "granular stemming obey an exponential law the resistance increasing rapidly with length of stemming". Plugs which exert substantial resistance are most effective in long stemmed holes.

In his second paper, Taylor 1945, he investigated the behaviour as an impulse force was applied to a column of stemming material. An apparatus was designed where a falling mass was directed onto a column of stemming material supported on calibrated "crush plates". As the mass was dropped the compression of the crush plate gave a measure of the force applied to the stemming by the impulse force. Another dynamic apparatus consisted of a simulated borehole with a low power explosive charge under a column of stemming. The time for ejection was taken as a measure of the resistance of the stemming material. Similar trends were obtained to those in the static experiments for plastic type stemming materials. From these tests he was able to rank stemming materials in order of their resistance.

A novel way to investigate the efficiency of blasting as determined by the stemming material was reported by Snelling and Hall, 1912. They used a trauzl lead block loaded with explosive and stemmed with various amounts of stemming material. The volume of the explosive cavity in the trauzl block was found to increase as the mass of stemming increased. This volume increased rapidly to a near constant volume with increasing amount of stemming added. Similar trends were noticed with different explosive compositions. Their basic conclusions were that "tamped moist fire clay, or similar plastic material, is the best stemming for all explosives, and dry pulverised material, such as dry fire clay, is least efficient".

The role of stemming as part of the blasting process has been discussed for many years and over time some rules-of-thumb have been developed. The effects of stemming are discussed and the general conclusions, supported by published works, are the use of stemming can open up the pattern hence reduce explosive consumption, (Anon., 1987). The effect of stemming on fly rock and air blast, flash arrester, dust suppression, fume generation and protection of explosive charges is discussed. The stemming length can be estimated by a formula and the use of decking and the type of stemming material best suited for a particular operation is also discussed.

Conventional stemming material is normally granular material of various sized particles. In underground coal mining water has been used as a stemming material and is reported in a paper (Anon., 1970). This paper describes the practices adopted in Belgium, France, West Germany, Great Britain and Holland. The water (100 - >250 ml) is contained in plastic ampoules and can have salts to increase the viscosity of the water. It is generally thought that water stemming reduces "the amount of dust produced by shot firing" and "the partial elimination of toxic and irritating fumes produced by the detonation of explosives".

A paper by Konya and Davis, 1978, is a continuation of the work of Snelling and Hall, 1912. They state that the forces involved are inertial, resistance to flow, and shear along the walls. Basically they divide stemming materials into 4 categories, liquids, free flowing granular materials, interlocking grains and completely solid materials. In conclusion they say the extremely high gas pressure acts on the base of the stemming only, the material moves till bridging occurs across the borehole and this bridging fails in compression or shear of the walls when movement occurs.

"Stemming is used in blasting operations to help contain explosive gases as long as possible" is the opening sentence of a paper by Kopp, 1987 where he investigated the types and amounts of stemming desirable in underground blasting. In his work he looked at various types of stemming, fine drill cuttings and coarse crushed stone with high and low energy explosives. High speed photography was used to monitor ground movement and stemming ejection and the burden velocity and expansion velocity were calculated from many crater blasts. His conclusions are "When sufficient stemming was used, ejection of the stemming was prevented" and he found that a length of 26 times the diameter "was found to prevent premature ejection of stemming". He also found the scaled depth of burial to be a simple relationship to maintain burden movement and no stemming ejection.

The effect of size distribution on stemming ejection was reported by Otuany et al., 1983 who monitored air blast and flyrock velocities. "Known weights of explosive charges were detonated in a simulated blasthole when confined with various types and size distributions of stemming". Their experimental apparatus was a 50mm diameter high tensile steel cannon and PETN was used as the explosive. They conclude by suggesting that a 41% reduction in the amount stemming can be achieved by selecting the optimum size distribution and "particle sizes of about 1/25th the diameter of the blasthole result in the best charge confinement". Reductions in air blast and ground vibration are also reported.

Some devices have been developed over the years to act as resistive plugs for the explosive gases to remain in the blasthole as long as needed. A paper by Zaburunov, 1990 discusses a stemming lock, patented by Worsley and Nixon, 1993, which is essentially an inverted cone which wedges against the borehole wall as the gas pressure is generated. The main purpose of the plug is to reduce the wasteful act of refilling the blasthole with inert material, thus allowing more explosive to be placed in the hole.

Just and Lamont, 1979 looked at stemming practices through out Australia and laboratory testing procedures to evaluate the relative efficiency of stemming materials. They discuss the many and varied practices of stemming in both underground and opencut blasting and look at blast noise and borehole pressure as measures of stemming resistance efficiency. A square section steel tube (150 mm square) with a 32mm diameter hole simulates the blasthole and the blast pressure, 20 metres from the charge, and the borehole pressure, attached to the borehole, were recorded for some 80 tests. Their general conclusions are similar to other published works that "the value of even small amounts of stemming material in reducing blast noise levels and increasing borehole pressures". In some underground mining operations they observed that "the introduction of low cost bulk-loading explosives has led to the virtual elimination of stemming materials".

CHAPTER 3

THEORETICAL DISCUSSION

3.1 INTRODUCTION

The usual industrial blasting practice is to drill a blasthole at the required location place the detonator, primer and explosives in the hole and any "decking" that may be required then fill the hole with an inert material, the stemming, to seal the blasthole. The main purpose of stemming the blasthole is to provide a resistance to the explosive force to be wasted and encourage this force to act on the surrounding medium. If this explosive force can be contained long enough, useful work will be performed on the surrounding medium in the form of fragmentation and burden movement. The correct amount of stemming to place on top of an explosive charge has been investigated over many years and a general rule of thumb developed has been the height or length of the stemming column should be equal to the burden.

However, the effect that the type of stemming material has on the final blast properties used to measure the efficiency of the blasthas often been neglected. The quality of the stemming material will effect the transfer of the energy from the explosive to the confining medium and thus the blast efficiency. How these properties of the stemming material affect the blast efficiency needs to be taken into account when designing blast patterns.

3.2 DYNAMIC RESPONSE

A typical industrial explosive consists of a fuel and oxidiser which can usually exist in contact with each other at normal temperature and pressure. As the chemical reaction proceeds, and because of its extraordinary rapidity, measurement of temperature and pressures at the reaction front are extremely difficult. However, a p-v diagram can be determined for explosive compositions and a typical curve is shown in Figure 3.1.

When the explosive, confined at volume V1, is detonated the gaseous products are formed and exert an extremely high pressure P1 on the confining medium. If the volume occupied by the gas is considered to expand adiabatically due to the high pressure of the gases, the p-v curve in Figure 3.1 results. As the gases expand to V2 the pressure of these gases will be at some lower value P2. Now if the strength of the confining medium is greater than this value P2 no extra fracturing of the confining medium can occur due to the lower pressure of the gases. Continual expansion of the gases to volume V3 results in the pressure dropping to P3, or in the normal blasting process, atmospheric pressure. It is in the region V1 to V2 that the extremely high pressure acts on the confining medium and play a primary role in fragmentation and burden movement.

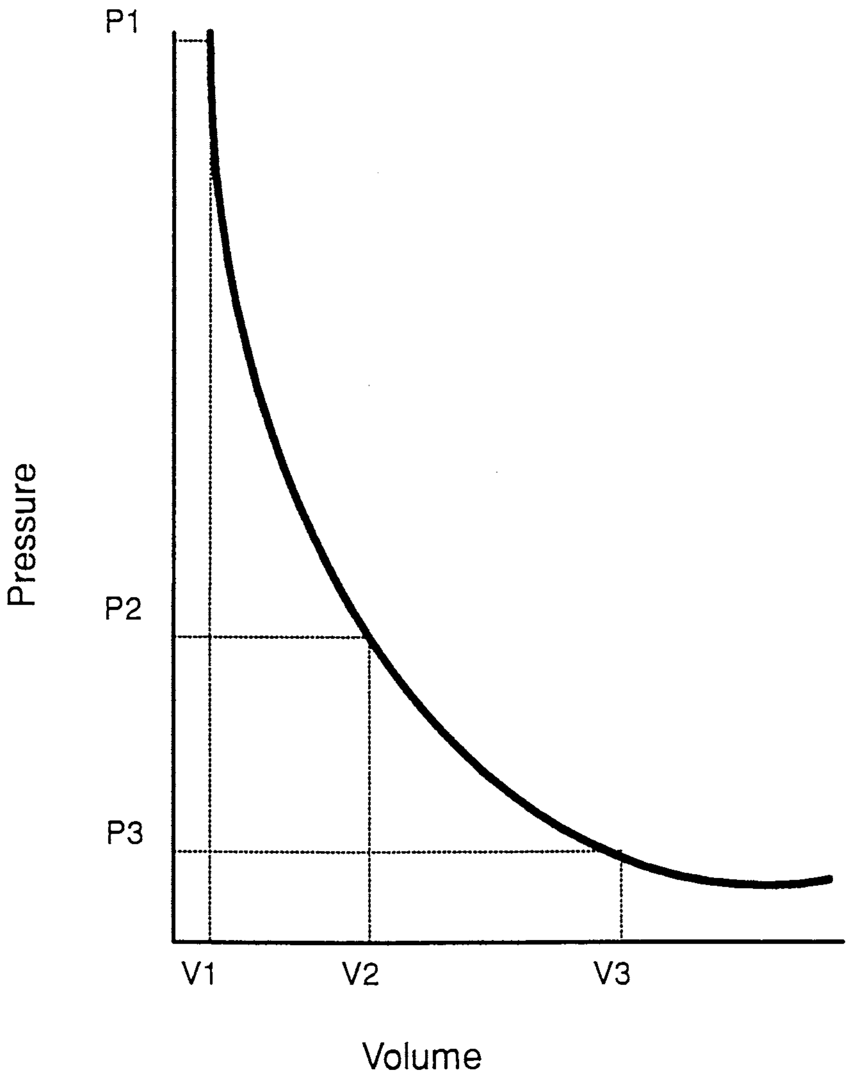


Figure 3.1 A typical pv curve for explosive gases

The confining medium is not only the rock into which the blasthole is drilled but also the stemming material placed on top of the explosive charge. These high gaseous pressures also act on the stemming material and even though no fragmentation, in the blasting sense, occurs in the stemming. This stemming material must pose some form of resistance to the premature release of these gases if useful work is to be performed by the explosive.

Industrial explosives generate large quantities of gases when compared to primary explosives and also have a lower shock component. Table 3.1 shows a comparison of some industrial and primary explosives.

Table 3.1. Comparison of Explosive Properties (after Johanson-Persson, 1970)

	Composition	VOD	Gas	Det Press.
		km/sec	L/kg	MPa
Primary Explosives				
Mercury fulminate	$HgC_2N_2O_2$	5.4	315	28000
Lead Styphnate	$PbC_6HO_2(NO_2)_3$	5.2	407	18000
Lead Azide	PbN_6	5.6	308	32000
TNT	$C_7H_5(NO_2)_3$	7.0	690	17000
PETN	$C_5H_8(NO_3)_4$	8.0	780	23000
Industrial Explosives				
Nitroglycerine	$C_3H_5(NO_3)_3$	7.6	715	20000
ANFO	$NH_4NO_3{:}CH_2$	5.0	970	4300
Dynamite		6.5	944	13000
Watergel		6.0	980	11000

Typically bulk industrial explosives ANFO, dynamite and watergels are initiated by explosives such as PETN and TNT. But the bulk industrial explosives produce more gas, which usually translates to heave energy, and less shock energy than primary explosives. This volume of gas produced has an effect on fragmentation and heave, or burden velocity of the confining material.

Considering the gases evolved under ideal conditions we can estimate the pressure inside the blasthole as the charge detonates can be estimated. The detonation pressure of an explosive is the pressure behind the reaction front driving the chemical reaction forward, and this pressure has been estimated for many types of explosive and some are shown in Table 3.1.

The detonation pressure is calculated from the following equation:-

$$D_p = 2.16*10^{-6}*\rho*VOD^2 \text{..................}(3.1)$$

where D_p = detonation pressure (MPa)

ρ = explosive bulk density (g/cc)

VOD = velocity of detonation (m/sec)

This equation has been derived empirically and relates the pressure in the C-J plane of the detonation process as a function of the initial density of the explosive. When this empirical density relationship is employed in the momentum equation the equation 3.1

results. This equation has been found to be accurate within the density range of 0.6-1.6 g/cc.

The blasthole pressure must therefore be less than this detonation pressure and could be related to this pressure and the volume of the explosive gases produced at the temperature of the chemical reaction. It has been found to be extremely difficult to measure the borehole pressure as the higher detonation pressure would destroy any measuring equipment before the blasthole pressure of the gaseous products was attained. Estimates have been made of the blasthole pressure ranging from 30 to 70 percent of the detonation pressure.

If we assume ideal conditions and a reaction temperature of 3000°K then a relatively simple estimate of the borehole pressure is the gas produced (l/kg) expressed in MPa. This value is extremely low when compared to the strength of some rock types and even when compared to the blasthole pressure estimates based on the detonation pressure of the explosive. Cook, 1971 talks about the blasting criteria in relation to the explosive properties and says according to the hydrodynamic theory the detonation pressure is equal to the product of the initial density, the velocity of detonation and the particle velocity.

$$P_{det} = \rho * VOD * W \text{..................(3.2)}$$

However, generally in condensed explosives the particle velocity, W, is normally approximated by one quarter of the velocity of detonation. Hence

Cook then goes onto say that the detonation pressure can be "roughly" approximated to the blasthole pressure by multiplying the detonation pressure by

$$P_{det} = \rho * (\frac{VOD}{2})^2 \text{..................}(3.3)$$

0.5. This last statement is similar to that put forward by Chiappetta, 1989. The VOD is found to normally vary linearly with density according to the relationship:-

$$VOD = a + b\rho \text{..................}(3.4)$$

where a and b are empirically derived constant and when this is substituted with $P_{det} = 2 \times P_{hole}$

$$P_{hole} = VOD^2 * \frac{(VOD-a)}{8b} \text{..................}(3.5)$$

can be used to estimate the blasthole pressure exerted by an industrial explosive.

This blasthole pressure causes a force to be exerted in all directions on the confining medium. It is this force which provides the movement of the confining medium and also, the fragmentation of the rock mass. Although there has been a large number of conjecture as to the cause of the fragmentation, the movement of the rock mass separates the fragments into individual pieces which forms the final fragmentation size.

As can be seen in Table 3.1, the pressure applied to the confining medium which includes the stemming material can be extremely high, in the order of thousands of MPa. This force is applied in all directions starting at the point of initiation of the explosive column, the primer, and usually travelling up the explosive column. So as the explosive detonates the reaction front is moving upwards followed by the force of the gaseous products which causes a strain on the blasthole wall and eventually reaches the base of the stemming column. When the force associated with the pressure reaches the stemming material, the stemming column resists the force and tries to contain the force within the explosive cavity.

Firstly, there is the inertial force of the stemming column that must be overcome before the stemming column can begin to move or be ejected from the blasthole. This force can be quite large when considering 8 metres of stemming column height in a 380mm diameter hole is considerable. This type of stemming column configuration is not uncommon in open cut operations. As the inertia of the material is the force to overcome due to its stationary position and gravitational forces are acting in a downward direction these forces must be exceeded before stemming ejection can occur.

Secondly, frictional forces will add to the resistive force of the stemming column to help contain the explosive force in the blasthole. When a body is moved along the surface of a stationary body the force causing the movement is resisted by the cohesion of the two surfaces. In the blasting process the blasthole wall is stationary and as the explosive force begins to push the stemming column the cohesion between the stemming material and the blasthole wall provide the frictional resistance to movement. Similarly, frictional forces occur between individual grains as they are forced to move relative to each other.

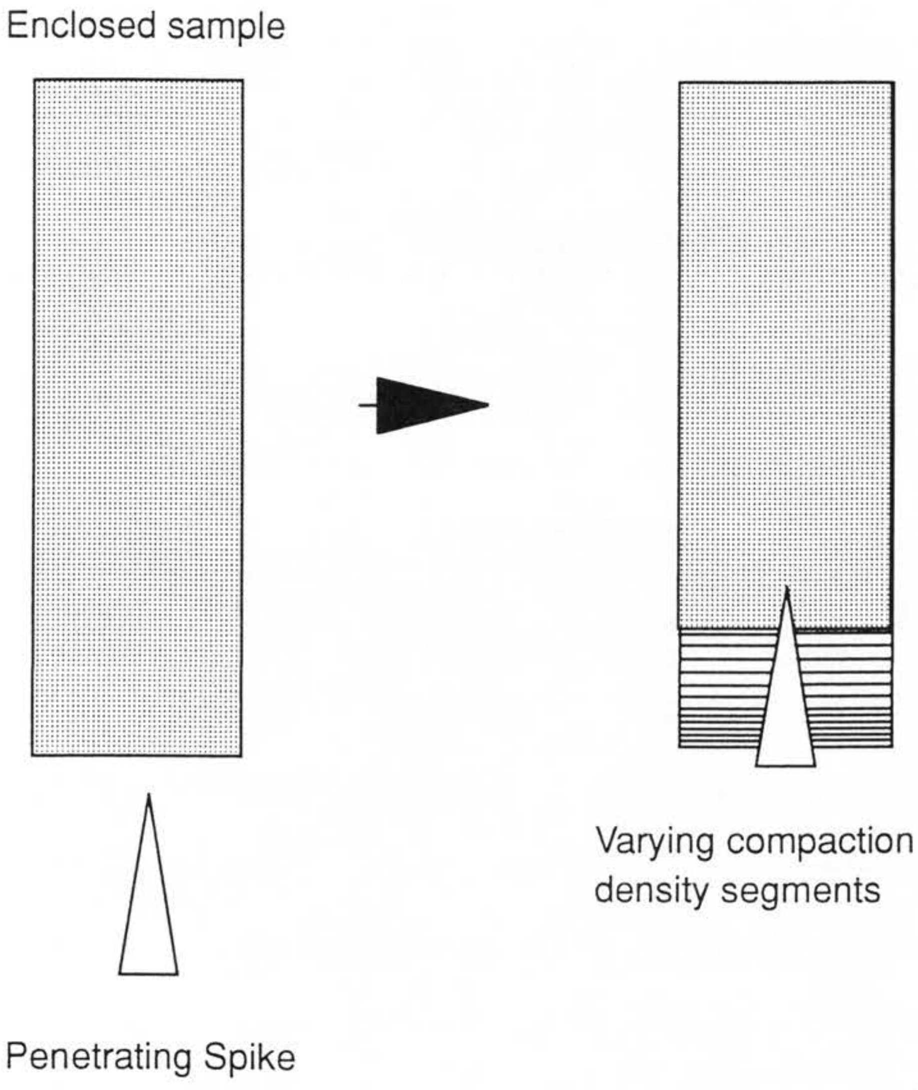


Figure 3.2 Action of penetrating spike on enclosed stemming material.

If we consider an enclosed container of stemming material so that no material can escape and a device is pushed into the sample material until it can penetrate no further we can gauge the relative resistance of different stemming materials. If the device penetrating the sample is a spike, the compression and intergranular behaviour of the material can be monitored. Such a device was designed and reported in Section 4.2.5.2.1. A schematic representation is shown in Figure 3.2.

As the spike enters the sample two effects occur. Firstly, the sample material resists the ingress of the spike and as the material can not move, in bulk, out of the sample container local rearranging of the material around the tip of the spike occurs. When the spike penetrates further into the material this rearranging and re-packing causes a change in the packing density of the material till the void spaces become a size which can not accommodate any more movement between the grains. Obviously the force applied to the spike is great enough some local crushing of the material close to the spike may occur.

While the spike is penetrating the stemming material this re-packing provides a resistance to the penetrating spike and the force applied to the spike dissipates and eventually comes to equilibrium. At this point in time the spike is at rest and its forward velocity into the stemming material is zero. If the force applied to the spike is small compared to the strength of the sample material, ie. dropping the spike onto a bed of material as in the drop rig apparatus, then negligible energy will be lost in crushing the material. Thus the energy in the spike as it free falls through the sample material will be used in rearranging the grains of material to a higher packing density and friction as the spike and grains slide relative to each other. If the distance travelled through the material by the spike is small then the frictional effect of the spike on the material will be kept to a minimum and compared to the energy to rearrange the material can be neglected.

A measure of the amount of energy required to resist the penetration of the spike is the rate at which the velocity of the sample canister decreases with distance. This decrease in velocity is accompanied by an increase in the packing density of the material as the spike penetrates the material. The increase in packing density occurs due to the "local" interstices being taken up by the movement of the grains and some "localised" crushing . The packing density would increase to a maximum, at the point where the spike comes to rest.

The results shown in Table 4.5 details the effect of packing density and velocity retardation of the spike as the sample canister comes to rest. As the spike penetrates the sample material it can be looked upon as analogous to the explosive force being applied to the column of stemming material. Similar trends could be expected to occur when the column of explosive is initiated from the bottom of the blasthole. The force travels in an upward direction till the reaction front reaches the base of the stemming column. This force acting on the stemming column base would tend to form a velocity gradient from the centre of the stemming column to the borehole wall. The maximum effect of the force would be experienced in the centre which would begin to compress the material close to the blasthole wall thus increasing the material packing density. Any interstices in the stemming material would be quickly taken up and in doing so would begin to offer increasing resistance to the explosive force. This increase in packing density of the stemming material would approach the true density of the material as the magnitude of the force is extremely high, and even many orders of magnitude higher than the compressive strength of the stemming material.

As the results in Table 4.5 show, with an increase in the packing density an increase in resistance is shown by the decrease in the velocity gradient of the sample canister with distance. This negative velocity gradient with distance travelled is found to be independent of material size and also independent of material type. Even when an increase in mass of the canister from 1kg to 14kg occurs, the velocity gradient remains similar. However, when the initial velocity is increased by raising the drop height to 2 metres from 1 metre, the velocity gradient decreases at a higher rate with distance. Because of the time involved in the blasting process similar trends would be expected to occur in relation to material size and material type. The stemming material in the blasthole experiences an impulse loading of a high magnitude but a similar time frame to that experienced in the drop rig experiments. Thus the rearranging within the stemming column and the resistance to movement would be independent of material type and material size in the near vicinity of the explosive force as it is applied to the base of the stemming column.

The movement of the stemming material in a blasthole can only occur when the explosive force is much greater than the resistive force of the stemming material. As stated above, the resistive force of the stemming material is made up of several components. One of these components that is alluded to in the field is the locking of the grains of stemming material as the explosive force is applied. This process is thought to be due to the "jig-saw" effect of the individual grains filling the interstices and forming a tighter and more resistive barrier to the explosive force. When blasting in wet ground where the blastholes can have metres of water at the bottom, this water is displaced to the top of the explosive column as the explosive is added. When the drill cuttings, fine material <10mm, are used in this situation, a fluid slurry can be formed which is relatively easy to eject from the blasthole, and is ineffective in containing the explosive gases to perform useful work. If coarse rocks are mixed with this slurry then the rocks, being denser than the slurry, will sink and form a more competent stemming material in the blasthole.

The locking effect can be thought of as a bridging of the grains of stemming material similar to the forming of arches using bricks. Each grain moves as the force is applied into the interstices forming a more densely packed structure. The outer edges of the column would be more difficult to move being held in place by the wall resistance of the blasthole and moving less freely than the particles in the centre of the column. As the pressure increases this readjusted layer thickness increases providing more resistance to movement of the column.

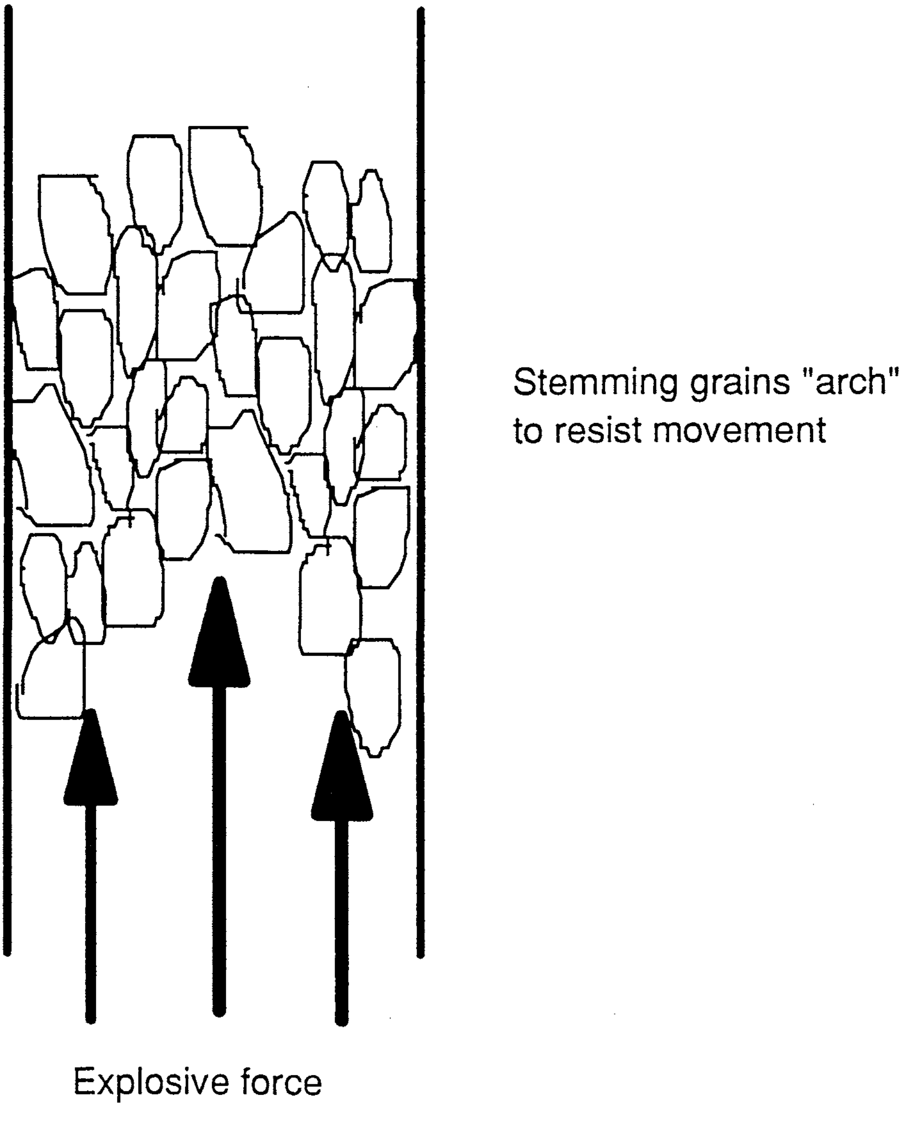


Figure 3.3 Locking or bridging of granular material

This bridging effect is only talked about in relation to large grains of stemming material or crushed rock material. Although costly, the grain sizes in the order of 10-20mm are usually thought of as being ideal stemming material because the material "locks" in place forming a competent force resistance to the explosive force. An indication of this "locking" ability is the aspect ratio described in Section 4.2.3.1. It is postulated that the particles in the 10-20mm grain size should have a more "elliptical" structure and the aspect ratio measured would be much less than 1. However, as shown in Table 4.2 this is not the case as all samples measured from particles as fine as 42 microns to rocks as coarse as 27.5mm showed no significant difference in their particle aspect ratio. So this "locking" effect would be evident in all particle sizes because of the ability to fit together like a jig saw and thus causing resistance to the explosive force.

3.3 DEPRESSURISATION

The detonation pressure has been studied for many years by researchers around the world and a fairly good understanding of the events that occur during detonation is now available. Many computer models have been calibrated using experimentally derived data to make sure the theoretical models predict events that actually occur in the field.

However, the area where "rules of thumb" are still in use is the estimation of the pressure in the blasthole or the force exerted on the confining medium. The detonation pressure for most industrial explosives is in the range of 10 - 50 GPa and the rule of thumb for the blasthole pressure is some where between 30% - 70% of this detonation pressure. Even if the low end of this rule of thumb is used the pressure inside the blasthole and the pressure providing the force on the confining medium is still in the order of gigapascals, which is well above the strength of the confining medium. The compression strength of rocks or confining material in the blasting process varies considerably from a few kilopascals, for weak weathered material, to hundreds of megapascals, for more competent rock. However, the tensile strength of most rock materials is very low due to their brittle nature and their values in the region of 5 to 15% of the compressive strength.

When an explosive detonates, the shock energy results from the chemical reaction of the fuel and oxidant and the sudden release of molecular energy from bond breaking to form more stable compounds. If the chemical components can be made to react more efficiently, as in the case of emulsions compared to ANFO, then higher shock energies will result. The shock energy is extremely high in magnitude but short in duration. The gas energy on the other hand is the result of the products of the chemical reaction and the conditions at which the reaction propagates. The gas energy is basically the pressure of the post reaction products on the confining medium ie. the blasthole pressure. It is this pressure over which there can be some control for a particular explosive.

If a confined gas at elevated pressure, relative to some more stable state is looked at then the state of the confining medium of the high pressure gas can be seen. Now if a gas at a high pressure is enclosed in a flexible container (ie. a balloon) and is connected to the outlet of this container by way of a valve to another flexible container at a lower gas pressure. When the valve is opened the system soon stabilises to equilibrium. The high pressure balloon deflates and the low pressure balloon inflates, as depicted in Figure 3.4. This is analogous to the high pressure explosive gases in the blasthole immediately after all the explosive has detonated. If there is an outlet to the lower pressure atmospheric state the gases will escape and the pressure will decrease inside the blasthole.

The flow of a gas under the influence of an elevated pressure is determined by the velocity imparted to the gas through an orifice and the area of the orifice. The following relationship exists for ideal conditions.

$$Q = A * \sqrt{2gh} \quad \text{..................(3.6)}$$

where Q is the volumetric flow of the gas, A is the area of the orifice, g is the gravity constant and h is the pressure difference across the orifice. If we look at the explosive gases under ideal conditions in a confined situation we can estimate the time required for the pressure to drop to a particular magnitude. A plot of pressure reduction with time for different orifice diameters is shown in Figure 3.5. By a simple iterative process the pressure decrease was calculated at various time steps and the new pressure calculated assuming ideal conditions.

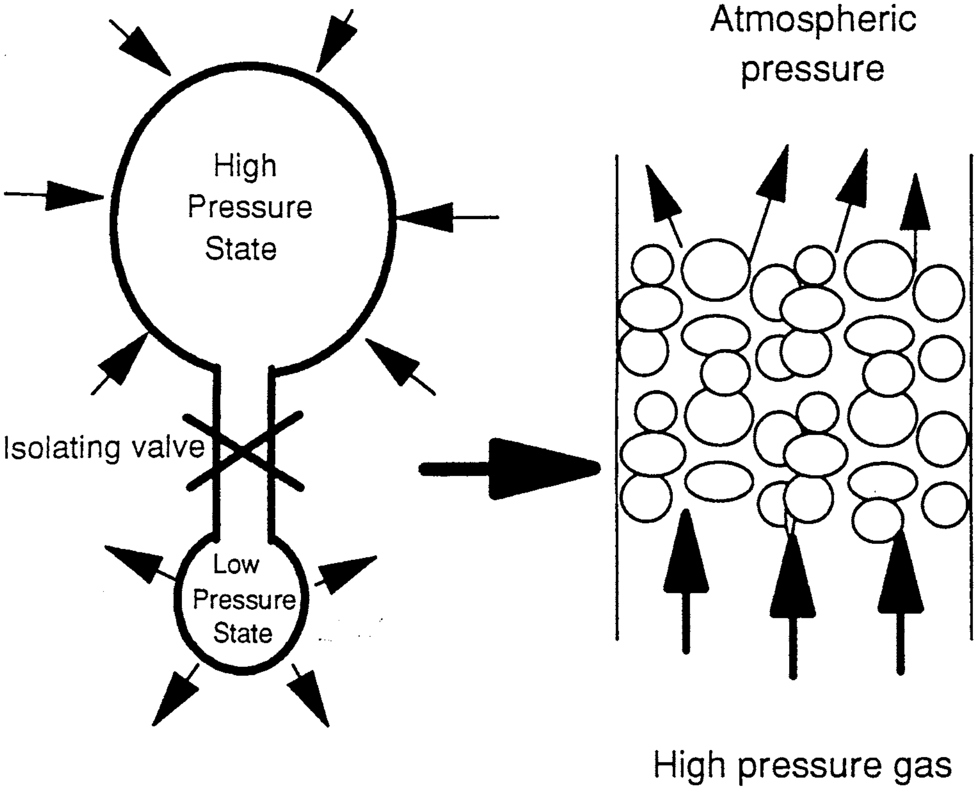


Figure 3.4 High pressure gas release analogy

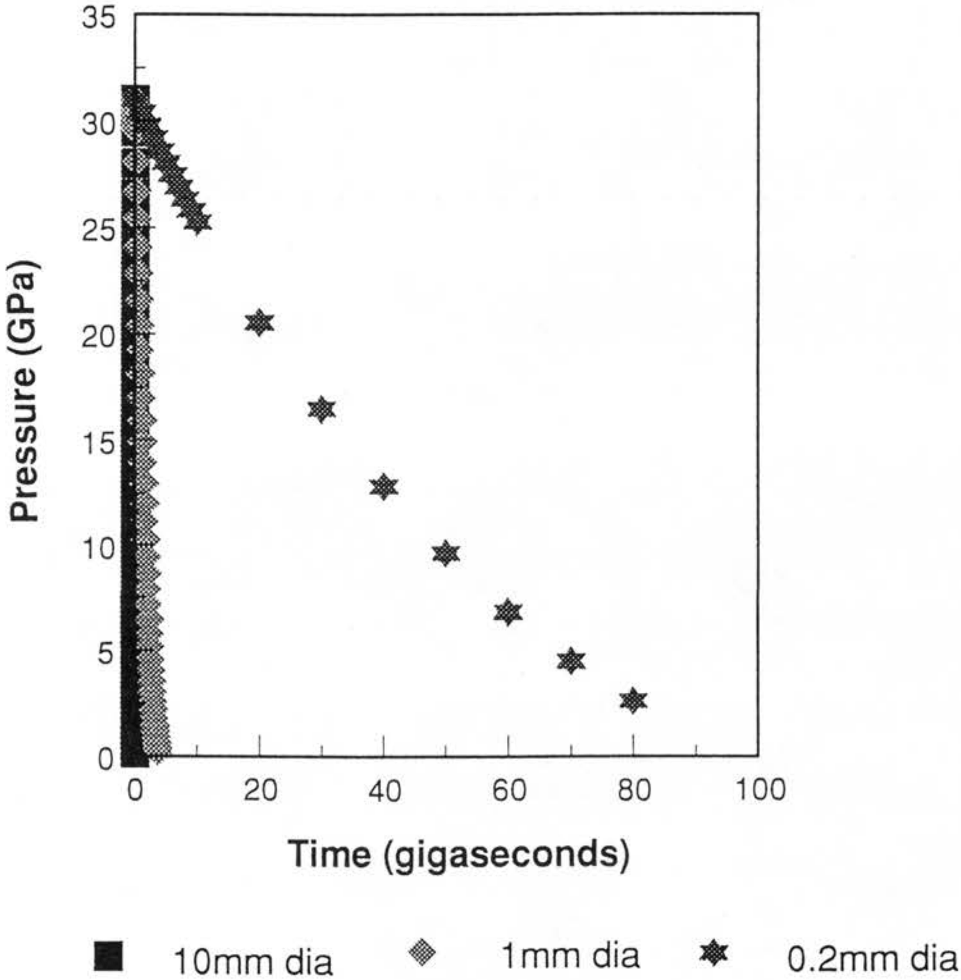


Figure 3.5 Pressure drop time relationship for various orifice diameters

Thus if the extremely high blasthole pressure is allowed to dissipate through voids or cracks, a time for its effectiveness can be estimated. If the compressive strength of the rock material is taken as a guide and any pressure less than the compressive strength of the rock mass is defined as valueless for performing useful work then a cut-off point can be nominated. A typical cut-off point has been reported at 100Mpa, ICI Explosives, 1993, above which pressure the explosive gases perform useful work in movement and fragmentation.

As the blasting process is dynamic and nothing remains the same as time progresses, whenever the gas pressure is able to perform useful work the structure of the rock mass is changing. Under ideal conditions with competent rock structure and similar stemming the gas pressure would move the rock mass until cracks appeared through to the surface for the gas to escape. These escaping gases would reduce the volume of the trapped explosive gases "in" the rock mass and hence reduce the pressure of the gases. As this process continues with time eventually the pressure would be so low that no extra useful work can be accomplished and the remaining gas will vent harmlessly to the atmosphere.

Although the formation of the explosive gases occurs in a very small time increment the magnitude of the force coupled with the short time frame causes an impulse force to act on the confining medium and the stemming material. As stated previously, a good stemming material is one with a large particle size with no fines to fill the voids thus forming a "bridge" or "locking" effect when the explosive force is applied. This is more than likely one reason why large particle stemming material has gained favour from mine operators when flyrock and noise are to be minimised. But, another reason is that stated above ie. a reduction in the explosive gas pressure with time as the gases permeate through the voids between the large rock particles. This fact will be shown in the laboratory experiments in Section 4.2.5.2.1. When the pressure was rapidly applied to the base of the stemming column in the stemming ejection rig tests, a drop in the maximum pressure under the stemming column was noticed for coarse sized stemming materials. Although the magnitude of the ejection pressure is many times less than the pressure in an explosive situation similar trends would be expected to occur. Immediately the gas pressure is applied and being at an elevated pressure dissipation to a more stable state begins.

The void space between stemming materials of different size is shown to vary significantly (see Table 4.3). The area of a void between individual particles is determined by assuming spherical particles and a close packed hexagonal structure. When the volume of the void space is calculated for unit cross sectional area, a void area can be estimated for individual sized stemming materials. As shown in Table 4.3 this void area can be quite large as the particle size of the stemming material increases. When conventional stemming practices such as the use of drill cuttings are examined it can be seen that an even distribution of particle sizes can cause the void area to be reduced significantly. For example for 10.5mm particle size stemming material the void area is

32.96mm^2, however when the maximum size of the material is 10.5mm with an even size distribution some of the void space is filled with these smaller particles hence reducing the void area available for gas release.

The effect of the pressure inside a closed vessel by the reduction in void area can be seen by following the pressure drop with time for different diameter release ports. Three such plots are shown in Figure 3.5. For a void area of 78mm^2 (approximately 10mm diameter) the time for the pressure to drop from a blasthole pressure of 30GPa to atmospheric pressure is in the order of 100 picaseconds. Whereas if the area is reduced to 0.03mm^2 (approximately 0.2mm diameter) the time increases to approximately 90 gigaseconds or an order of magnitude increase of 1000 times. So when this reasoning is applied to the void spaces within the stemming material it can be seen that a change in the blasthole pressure can occur as the gases escape to atmosphere through the larger void spaces in the larger particle stemming materials. Of course, this is an idealised view of the dynamic nature of the process but the time scale involved for gas depressurisation is conceivable. When the time before initial movement in rock blasting is in the order of milliseconds per metre of burden which is an order of magnitude of 1,000,000 times slower.

This is not to say that the depressurisation, which is occurring at the same time as the energy is being imparted to the confining medium, is causing a detrimental effect by releasing useful energy. On the contrary, this depressurisation helps in the transfer of energy from the explosive gases to the confining medium by providing a release valve. As the force from the gases is applied to the confining medium local pulverisation and cracking appears close to the blasthole and the cracks radiate for some distance being driven by the pressure of the explosive gases. The resistance offered by the confining medium is considerably less than the explosive force and the confining medium begins to move in the direction offering least resistance usually to a free face or the bench top. This movement causes a volume increase which is taken up by the introduction of void spaces as the cracked confining medium begins to expand. These void spaces are filled by the explosive gases and the overall pressure of the gases begins to fall.

It has been observed from the analysis of the high speed film that the time for the ejection of the stemming column after the confining medium moves has varied for different sized stemming particles. This observation will be discussed in Section 5.3.3.2. But when the confining medium begins to move away from the blasthole the "expansion" of the blasthole in the stemming region will make the stemming less effective and more easily prone to complete failure by ejection. Ideally the stemming should stay in place till the confining medium completely moves away from the rock mass in a "forward" direction. Then and only then is the function of the stemming material obsolete as the force from the compressed gases is imparted to the moving rock mass and the gases begin to vent to the lower atmospheric pressure.

At the onset of the rock mass movement the stemming material would also experience movement and the coupling of the stemming material to the blasthole wall would be reduced. This would cause a lessening of the resistance to ejection and failure of the stemming will occur due to the high pressure of the gases inside the blasthole. If a "relief valve" is built into the stemming material and the pressure allowed to escape to atmosphere. Although it may be only a slight reduction, then the time before the stemming failure occurs would increase thus allowing more energy to be transferred to the confining medium. As stated above this phenomena was observed in the field study.

So the use of stemming material of a large size in relation to the blasthole diameter is extremely beneficial in extracting more useful energy from the explosive. If the stemming material remains in place for a longer period more energy can be imparted to the confining medium thus performing more useful work in fragmentation and rock mass movement.

3.4 BURDEN MOVEMENT

The two main reasons for blasting in both surface and underground operations are to produce fragmented material of the size required and to throw the broken rock to form an acceptable muck pile in order to suit the excavation equipment in use. Many parameters are under the control of the shot firer to produce these desired properties and deviations from the optimun design can cause errors which at times can be costly to rectify. The parameters that the shot firer has under his control start from the time the blasthole is drilled. The correct angle, burden and spacing and especially the depth of the hole are all important parameters to "get right" even before explosive is loaded into the blasthole. When the explosive is loaded in the hole, the type of explosive, collar depth, primer position, down the hole delay are all parameters which must be considered before the tie-in sequence is "wired in". When all these parameters are correct and the design of the shot is compatible to the local geology the timing sequence used to fire each hole will decide the direction broken rock from each hole will be thrown. This is possibly the part of the blasting process which can cause the most "damage" in producing a poor blast result. Local geology and rock structure play an important part in determining the outcome of the blasting process. Blast parameters which work against the rock structure will poor or even dangerous consequences and may even need to be blasted again.

However the transfer of energy from the detonation of the explosive and the resultant gas pressurisation on the confining medium is the single most important aspect of the blasting process. The enormous amount of energy released by the explosive has only a very short period of time to perform useful work and during this time the more effective the transfer of energy from the explosive gases the better will be the final blast result. As will be shown in Section 4.3 and in the field study, (Section 5.3), the longer the energy is in contact with the confining medium more useful work can be done in fragmentation and burden movement.

In the laboratory study this effect of time and the transfer of energy to the confining medium was investigated by looking at releasing the gases produced by the explosive at various rates and measuring the properties of the blast (Armstrong et. al., 1993).

The force on the rock mass by the high temperature and pressure gases of explosives is extremely large compared to the resistance offered by the rock mass. Because of the pressure differential between the gases from the explosive "inside" the rock mass and the pressure of the atmosphere, to equilibrate, these gases need to migrate to the lower pressure atmosphere. This migration is accomplished through the crack network and the expansion of the confining rock mass which causes extra tensional stresses hence extending cracks to the surface forming a path for these gases to migrate through. During this migration to an equilibrium pressure state the expansion of the crack network also causes an expansion of the confining rock mass which in effect causes movement of the rock mass. It is this rock mass movement which is an important property of the blast as it will define the final resting place of the fragmented rock mass or the shape of the final muck pile.

But how is the energy in the explosive gases given upto the confining rock mass? Considering that the confining rock mass at rest there is an inertial force that must be exceeded before this mass can move. Once movement begins then work has been carried out on the system, and as work is function of the applied force over some distance the following relationships are considered (in an ideal situation).

$$W = f*s \text{..................}(3.7)$$

where W is the work performed, f is the applied force over some distance s. From the second law of motion we know that an object will accelerate according to

$$f = m*a \text{..................}(3.8)$$

where f is the force produced by a mass m accelerating at a rate a. Also from kinematics an object will have its velocity changed by

$$v^2 = u^2 + 2as \text{..................}(3.9)$$

where s is the distance over which the velocity changes from some initial rate u to a final rate v. So when these equations are combined together an equation for work done on a stationary body can be evaluated for an ideal case.

$$W = \frac{1}{2}*m*v^2 - \frac{1}{2}*m*u^2 \text{..................}(3.10)$$

So the pressure in the explosive gases, force per unit area, is translated to useful work by accelerating the fragmented rock mass from rest to a velocity v over some finite distance.

One of the most important ways to effectively translate this explosive energy to the confining rock mass is to contain the explosive gases in the confining rock mass for as long as the rock mass is of a competent nature. This containment is accomplished by stemming the explosive in the blasthole before initiation of the explosive.

The competency of the stemming material as far as its resistance to ejection will be discussed in Section 4.2.5.1.1 and the ability of the stemming material to act as a pressure relief valve to help transfer some of the energy from the explosive to the rock mass will be discussed in Section 4.3. Experimental work was carried out in a blast chamber to measure the effect of releasing the gas pressure prematurely, and this work is reported in Section 4.3.2.1. Basically releasing the explosive gas prematurely can increase the fragment size and reduce the burden velocity from the blast. The controlled release rate was accomplished by placing an orifice above the blasthole and monitoring its effect on the blast properties. In a practical sense the premature release of the explosive gases from the blasthole would occur if no stemming material or very little stemming was placed on top of the explosive causing stemming ejection occurred.

Useful energy is lost if the explosive gases are allowed to vent to the atmosphere before performing some work. The flow of gas through an orifice is a function of the pressure of the gas and the area of the orifice and is shown thus :

$$Q = \sqrt{2gA^2(P_f - P_i)} \quad \text{..................(3.11)}$$

where Q is the gas flow rate,g is the acceleration due to gravity, A is the area through which the gas is flowing and Pi and Pf are the initial and final pressure of the gas. This relationship is only strictly valid under ideal conditions.

As can be seen from equation (3.11) that the area of the orifice is directly related to the quantity of gas flowing through the orifice. Hence the transfer of the energy from the explosive gases to the confining medium can be significantly improved by reducing any orifice (ie. by stemming the blasthole) or preventing the escape of gases to the atmosphere.

In the laboratory experiments, different diameter orifices were aligned over the collar of the blasthole to exhaust the explosive gases at different rates. If the transfer of energy from the gases to the confining rock mass is related to the quantity of gas from the explosion then removing some of the gas will reduce the blast performance proportionally. If the gas is allowed to "escape" through an orifice at a "controlled" rate then the changes in the blast performance should be related to the changes in the gas quantity or a good correlation between the orifice area and the blast performance should be obtained. The results for both the burden movement and fragmentation are shown in Figure 3.6 along with the correlation coefficients for these properties as a function of the orifice area.

As can be seen from Figure 3.6 a very good linear relationship exists for fragmentation which could possibly be influenced by experimental error to some degree. A reasonable correlation coefficient exists for the burden velocity although some other effect might have an influence over the burden velocity. This high degree of correlation exhibited from both of these blast properties tends to validate the hypothesis of containing the explosive gases performs useful work.

The blasthole diameter in the laboratory scale blasting trials was 10mm and as shown in Figure 3.6 even with no containment of the explosive gases some work was done on the confining rock mass. So in a practical sense the explosive gases must be retained long enough to impart much of its stored energy to the confining rock mass. As discussed in Section 3.3 the reduction in gas pressure can occur quite rapidly if the orifice equivalent diameter is large. In the blasting process this is a time dependent phenomena which also depends on the expansion rate of the confining rock mass. The expansion of the confining rock mass opens up crack networks to the lower pressure atmosphere for the higher pressure explosive gases to escape. At this point containment of the explosive gases can no longer occur and any remaining gas energy is lost to atmosphere.

Before equilibrating to the pressure of the atmosphere these gases still impart kinetic energy to the now fractured and moving rock mass. The rock mass is moving under the influence of gravity and ceases to be driven by the force of the explosive gases when the rock expansion (increase in volume) with time is such that the pressure of the explosive gases is equal to atmospheric pressure. It is

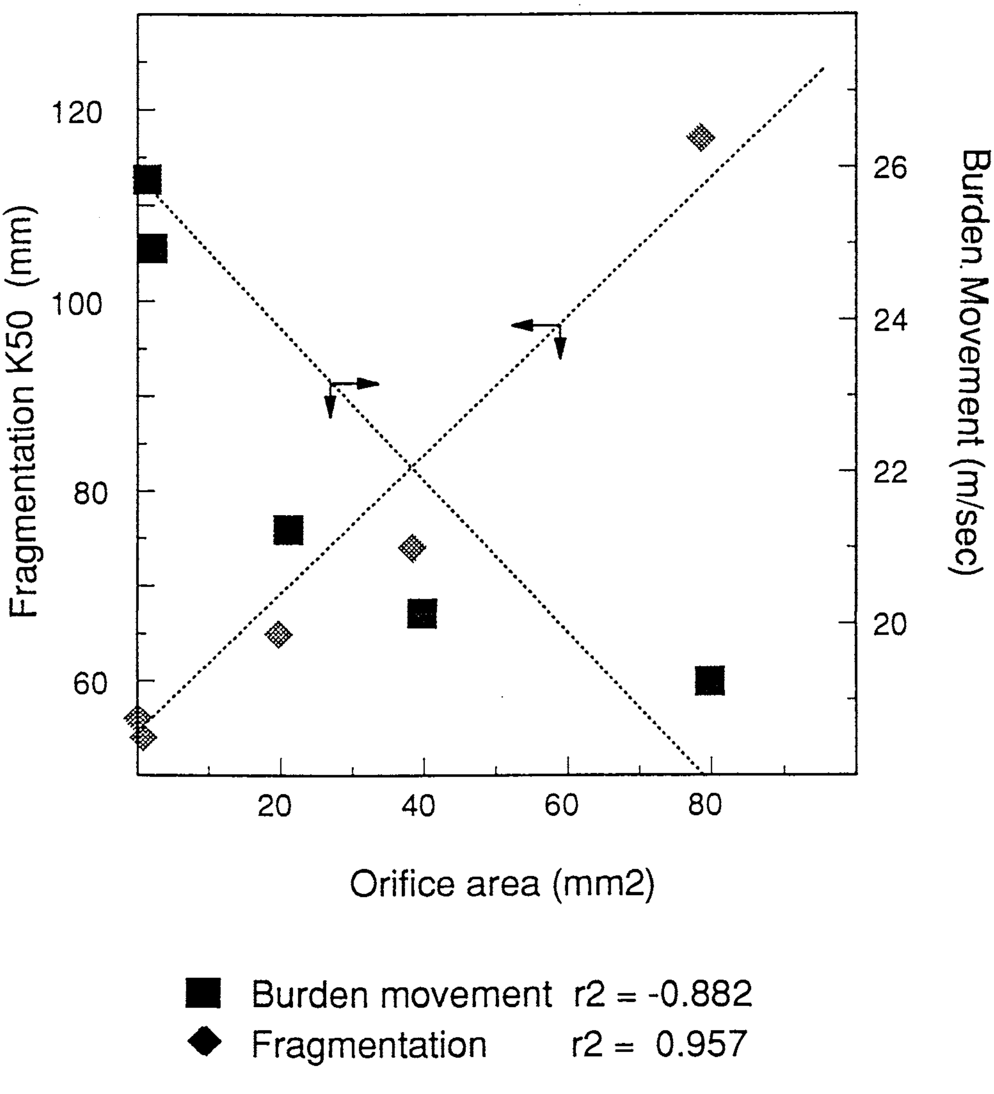


Figure 3.6 The effect of orifice area on blast properties.

assumed that the rock mass expands at a rate of r mm/msec (the front burden velocity) then a simplified case can be represented by Figure 3.7. As the front burden moves the entire rock mass from the blasthole to the free face begins to expand and in doing so opens up cracks to the atmosphere for the gas to escape. Simply this can be represented by the triangular area ABC in Figure 3.7. As the rate of expansion of point C is for example r(t) mm/msec then the area "opened up" by the expanding block is for a face of 1 metre high.

$$a = \int^{t=t} 0.5r(t)\ dt \ldots\ldots\ldots\ldots\ldots\ldots (3.12)$$

It is a simple calculation to determine the time for the volume of the triangular area to be equal to the volume of the compressed gas. This time for expansion (for the gases to escape and thus to perform no more useful work) can be quite small, in the order of tens of milliseconds even in the ideal case.

It is in this very short time frame that the fractured rock is given the energy to overcome the resistive forces to movement. This burden velocity is a function of the explosive energy input and the total resistive force working against this energy. This force (from the explosive gases) also has to perform some of the fragmentation work so the energy available to move the fragmented material is reduced.

The energy partitioning in the blasting process, ie.shock energy and gas energy, is fairly well known from empirical determination and mathematical models. This

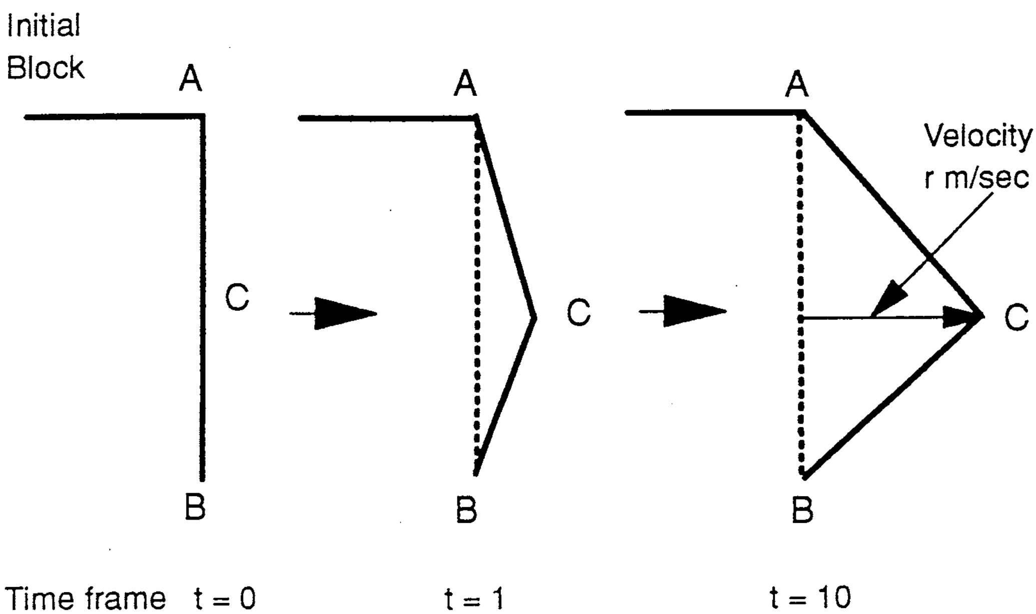


Figure 3.7 A simplified approach to gas dissipation

gas energy is further partitioned into fragmenting the confining medium, providing the energy to move the rock mass and also a good portion is lost to the atmosphere as the gas pressure equilibrates. It is this partition of the gas energy that is not well understood and research in this area is being carried out by workers in the blasting arena throughout the world. An empirical indication of the effect of containing the gas energy in the blasthole long enough to perform useful work has been given in equation 3.12. The quality of the stemming in transferring the "trapped" gas energy to the confining rock for efficient blasting can not be over emphasised.

CHAPTER 4

LABORATORY STUDY

4.1 INTRODUCTION

A laboratory study was carried out to investigate the physical characteristics, as well as the blasting characteristics, of the stemming material and the effect of these characteristics on blast performance. The physical properties of the stemming material included the particle aspect ratio (defined in Section 4.2.3.1) or the angularity of individual particles and its effect on "locking" within a simulated borehole under an applied pressure. The porosity or voidage of the stemming column is also an important parameter as this indicates the free space between particles for movement of the stemming grains and passage of the gases as the explosive force is applied.

The blasting characteristics investigated included the "controlled" release rate of the explosive gas pressure and its effect on burden movement and fragmentation. The effect on blasting performance was further examined with different types of stemming material ranked on a resistance to movement basis. Finally the effect in an actual blasting situation was investigated at a rock quarry where a production blast was monitored using high speed movie photography to assess the blast performance. The results of these blasts are discussed in Chapter 5.

4.2 LABORATORY INVESTIGATION

The laboratory study was divided into two sections :-

1) Physical properties of stemming materials.

2) Effect on blasting performance of stemming materials.

Each section will be discussed in detail with the emphasis placed on the quality of stemming in assessing blasting efficiency.

4.2.1 Physical Properties of Stemming Materials

Typically, stemming is looked upon as a granular material placed on top of the explosive column to contain the explosive gases until useful work has been done on the confining material. The individual particle size of the stemming materials ranges from extremely fine dust to rocks approximately 25mm in diameter. As discussed previously the size of the stemming material dictates the purpose for which the stemming is used. In this thesis the following properties of the stemming material were investigated :-

a) Particle aspect ratio which indicates the angularity of the individual particles and thus is an indication of the "locking" of the stemming as a force is applied to a column of the material.

b) The porosity or voidage of a column of stemming material indicates the space available for its movement or rearrangement, and gas permeation of the stemming column as the explosive force is applied.

4.2.2 Sample Materials

The sample materials used in these trials were those that would be used in future laboratory scale blasting trials. Sand, crushed limestone, crushed iron ore and crushed granite were selected as the sample stemming materials. All materials were washed and dried to remove any contaminating gangue to ensure relatively constant density material for the different size ranges tested.

The material characteristics, chemical and physical, are detailed in Table 4.1. The four types used in the laboratory study were considered to cover a wide range of materials that would be experienced in actual blasting applications. These materials were relatively free of gangue so minimising errors due to impurities. Their size distributions shown in Table 4.1 were consistent with that which would be required in the laboratory scale blasting trials (ie. the laboratory scale blasting trial blasthole diameter was 10-16mm). The size distribution for all, except the sand, were such that individual size fractions could be readily obtained without crushing or screening excessively large quantities of materials.

4.2.3.1 Particle Aspect Ratio

The particle aspect ratio was determined by measuring the maximum and minimum size of individual particles and calculating the ratio of these sizes. These "sizes" were measured on a digitising board and were based on two dimensions as it was assumed that a free falling particle would come to rest so that the particle centre of gravity was at a minimum height and the two largest "sizes" would lie in the same plane. Thus the largest "size" of the particle (L_{max}) and the smallest "size" (L_{min}) could be measured with reference to a scale. A pictorial representation is shown in Figure 4.1 and photographs in Figure 4.2.

Table 4.1 Stemming Material Characteristics.

Constituent	Sand	Limestone	Iron Ore	Granite
$CaCO_3$	17.01	97.32	0.27	-
SiO_2	80.75	0.81	3.30	73.50
Fe_2O_3	0.50	0.35	92.10	-
Al_2O_3	0.61	0.42	1.41	23.13
Density	2.67	2.72	5.00	2.65
Size Distribution (Cumulative Percent Passing)				
4mm				90.5
2mm		79.6	59.1	65.3
1mm		51.6	36.0	55.7
0.5mm	86.2	34.0	23.0	40.1
0.25mm	4.3	21.2	14.7	28.7
0.125mm	0.5	14.8	9.9	12.3

Approximately 100 individual grains each of various types and sizes of materials were measured and the individual aspect ratio was calculated as follows :-

$$P_{ar} = \frac{L_{min}}{L_{max}} \text{....................(4.1)}$$

As P_{ar} approached a value of 1 the particle was spherical in shape and conversely as P_{ar} approached 0 the particle was more angular in shape and needle like.

4.2.3.2 Results

The average aspect ratios for each particle was calculated together with the standard deviation and is reported in Table 4.2. The particle aspect ratio results are shown in Table 4.2 indicate a fairly wide spread of data even when closely sized materials are considered. The values measured ranged from spherical particles (aspect ratio of 1.00) to an elongated needle shaped particles (aspect ratio as low as 0.254). This large variation is also evident in the large standard deviation and coefficient of variation calculated for each material size fraction. The coefficient of variation is a means of comparing the variation on a common basis and when this statistic is compared for both the Laboratory Scale Materials and the Crushed Rock Materials there is no significant difference between either group of materials. Thus this variation can be assumed to be "normal" for the preparation of these materials indicating no bias in the particle aspect ratio for the Laboratory Scale Materials and the Crushed Rock Materials.

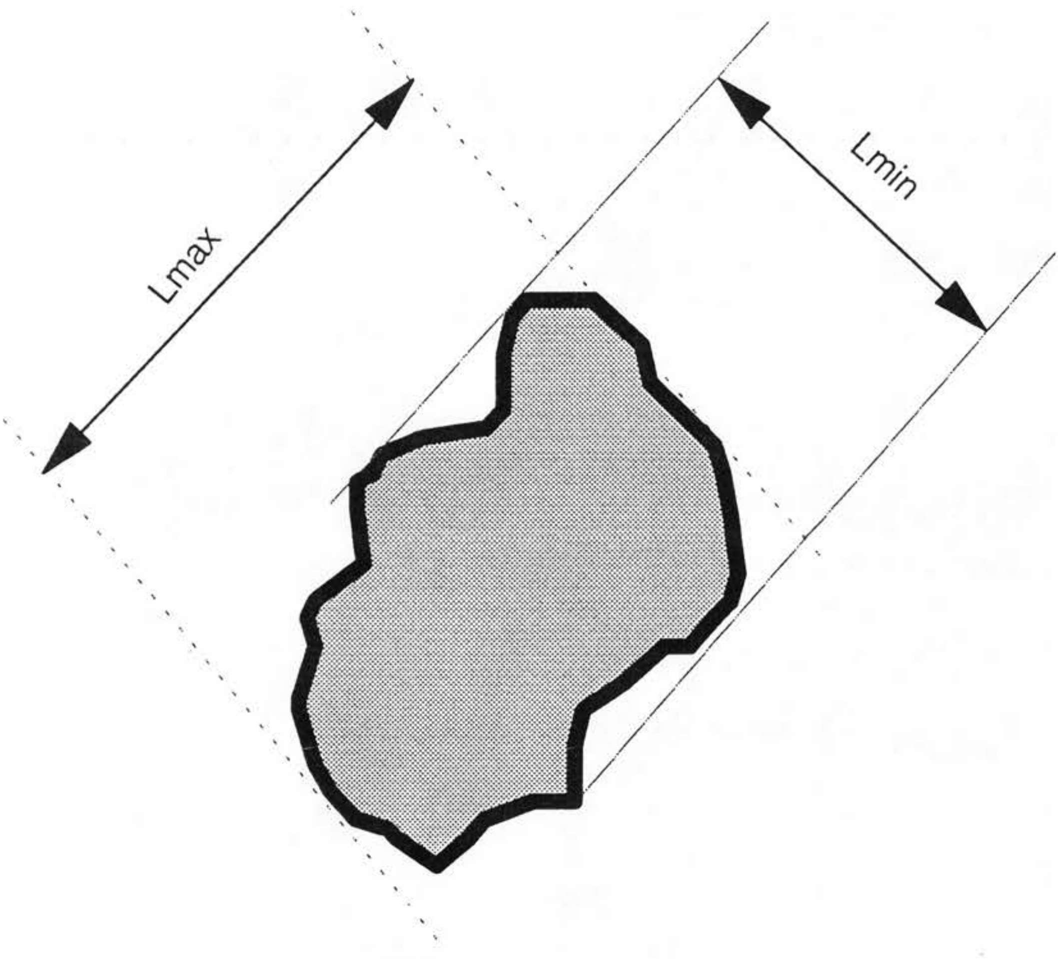
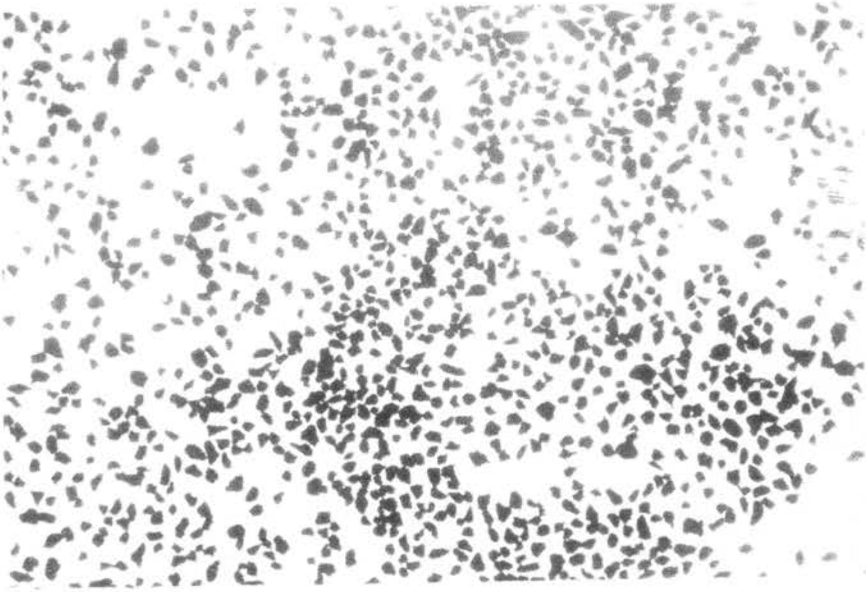


Figure 4.1 Particle aspect ratio schematic



a) 0.2mm crushed limestone

b)15mm crushed rock

Figure 4.2 Photographic analysis of particle aspect ratio

Table 4.2 Particle Aspect Ratio Results

Material	Size	Aspect	Ratio			Coeff
	(mm)	Maximum	Minimum	Average	Std Dev	Variation
Laboratory Scale Materials.						
Iron Ore	0.042	0.995	0.357	0.742	0.153	20.6
	0.188	0.996	0.424	0.725	0.146	20.1
	1.500	0.992	0.410	0.778	0.144	18.5
Limestone	0.042	0.998	0.387	0.828	0.145	17.5
	0.188	0.994	0.402	0.769	0.148	20.4
	1.500	0.997	0.388	0.738	0.156	21.1
Sand	0.800	0.995	0.441	0.787	0.130	16.5
Crushed Rock Material						
Granite	27.5	1.000	0.381	0.702	0.171	24.4
	24.2	1.000	0.381	0.686	0.143	20.8
	20.5	1.000	0.400	0.726	0.148	20.4
	16.3	1.000	0.429	0.727	0.132	18.2
	13.5	1.000	0.375	0.713	0.155	21.7
	10.5	1.000	0.300	0.739	0.167	22.6
	6.0	1.000	0.500	0.844	0.162	19.2
	5.0	1.000	0.500	0.851	0.167	19.6
	3.5	1.000	0.500	0.861	0.188	21.8

4.2.4.1 Column Porosity or Voidage

The porosity ratio or voidage of a column of the stemming material was determined by measuring the volume of water required to completely fill interstices between individual grains in a column. The container was selected to be large enough to minimise errors due to wall effects and both the bulk density and particle density of the stemming materials was calculated. Materials chosen were competent in nature and free of internal porosity as shown by the small difference in the particle density and the material density.

Typically the procedure consisted of filling a container of large enough diameter to a known volume (V_t) with a known mass (M_1) of material of a known particle size (P_S). Water was then added to the same volume and the quantity of water added (V_W) was recorded. The following parameters were calculated :-

$$a)\ Bulk\ density\ D_b\ (g/cc) = \frac{M_1}{V_t} \quad \text{....................(4.2)}$$

$$b)\ Voidage\ E = \frac{V_w}{V_t} \text{....................(4.3)}$$

$$c)\ Material\ Density\ D_m\ (g/cc) = \frac{M_1}{(V_t - V_w)} \text{....................(4.4)}$$

4.2.4.2 Results

The voidage of packed columns of materials are summarised in Table 4.3 along with the calculated void area for each particle size used. The voidage of a packed column of granular material is shown to be independent of particle size as no relationship is evident for either the Laboratory Scale Materials or the Crushed Rock Material. The voidage was measured between 0.368 and 0.550

Table 4.3 Porosity Results.

Sample	Size (mm)	Bulk Density (g/cc)	Material Density (g/cc)	Porosity Ratio %	Voidage Area (mm^2)
Laboratory Materials					
Sand	0.80	1.61	2.52	0.36	0.28
Limestone	0.80	1.47	2.64	0.44	0.35
Iron Ore	0.80	2.03	4.69	0.57	0.58
Granite	3.00	1.36	2.66	0.49	6.73
	1.50	1.40	2.72	0.49	1.68
	0.80	1.39	2.25	0.38	0.27
	0.06	1.45	2.37	0.39	0.02
Crushed Rock Material					
Granite	27.5	1.39	2.63	0.47	439.37
	24.2	1.46	2.69	0.46	264.81
	20.5	1.49	2.70	0.45	142.66
	16.3	1.47	2.71	0.46	65.84
	13.5	1.50	2.68	0.44	50.08
	10.5	1.44	2.38	0.40	32.96
	6.0	1.46	2.69	0.46	16.45
	5.0	1.43	2.63	0.46	10.56
	3.5	1.45	2.39	0.40	4.63

which is consistent with that obtained for packed spheres, Coulson et. al., 1985. Also there does not appear to be any significant difference between the porosity of the Laboratory Scale Materials and the Crushed Rock Material. Thus the voidage available to, both gas dissipation and granular movement as a pressure is applied is relatively constant for both the fine particle sizes used in the laboratory scale blasting trials and also the crushed rock aggregate as used in actual production blasts.

4.2.5 Dynamic Properties of Stemming Materials

In the blasting process as the explosive detonates extremely high pressures and temperatures are exerted on the surrounding medium in all directions. The stemming materials placed on top of the explosive column experiences these pressures and temperatures and to perform its function must be able to resist the force long enough until the burden material has cracked and movement occurs. The dynamic properties of the stemming materials investigated were :-

a) The resistance to movement of a column of stemming material as a force is applied until it is finally ejected.

b) The packing and re-arranging of a column as a force is applied or the "compressibility" of the stemming material.

4.2.5.1.1 Resistance to Movement of Stemming Column

A laboratory rig was constructed where a force was applied to the bottom of a column of stemming material and the pressure monitored at the base until ejection occurred. The ejection rig is shown schematically in Figure 4.3 and photographically in Figure 4.4.

The stemming material is contained in a concrete tube with a 10mm diameter hole drilled in the centre. A porous bronze disk is placed at the base of the concrete tube to support the stemming material. Air pressure at 600 kPa is applied to the porous bronze disk by way of a ball valve which has a 90 degree close to open cycle. A switch is incorporated on the main air ball valve handle to initiate the timing cycle of a digital oscilloscope. A very responsive pressure transducer is fitted below the porous bronze disk and is connected to the digital oscilloscope. A typical event (valve open, pressure on, sample ejected) occurred in approximately 1 second and a typical trace is shown in Figure 4.5.

4.2.5.1.2 Results

When a column of stemming is subjected to the pressure of the explosive gases the column and surrounding medium experiences a force which aligns itself in the least resistive direction. If the stemming material is not coupled to the wall of the blasthole effectively (ie. wet blastholes) the explosive force can easily overcome the inertial force of the mass of stemming and the column can begin to move and in some cases can be ejected from the blasthole. This phenomena was measured in the ejection rig and the results of the stemming column under pressure loading are shown in Table 4.4. From Table 4.4 it is evident, for all three materials tested (iron ore, limestone and granite) at various sizes the work required for ejection increases as the particle size decreases (for iron ore at 1.50mm the work done was 1667 kPamsec units compared to iron ore at 0.063mm the work done was 2294 kPamsec units). However, this increase in work to eject is accompanied by an increase in the pressure at the base of the stemming column (iron ore at 1.50mm a pressure of 90.95 kPa compared to iron ore at 0.063mm a pressure of 97.15 kPa). This would tend to indicate that there is some "leakage" of pressure through the coarse interstices in the larger size material before ejection occurs.

4.2.5.2.1 Stemming Column Under Impact Load

A laboratory rig was designed to monitor the effect of applying an impact load to an enclosed column of stemming material. An enclosed sample of stemming material was dropped onto a penetrating spike and the rate of velocity decrease with distance was monitored as the sample came to rest. The drop apparatus is shown schematically in Figure 4.6 and photographically in Figure 4.7.

Table 4.4 Stemming Column Under Pressure Loading Results.

Sample	Particle Size (mm)	Area Under Curve (mVsec)	Maximum Pressure (mV)
Iron Ore	1.50	1667	90.95
	0.80	1805	93.02
	0.063	2294	97.15
	Mixed A	4203	95.77
Limestone	1.50	2067	94.39
	0.80	2315	95.77
	0.063	2467	97.15
	Mixed A	4354	97.84
Granite	3.00	2301	89.57
	1.50	2460	90.95
	0.80	2549	92.33
	0.063	2653	93.70
	Mixed A	6187	95.77
Sand	0.80	1040	93.02

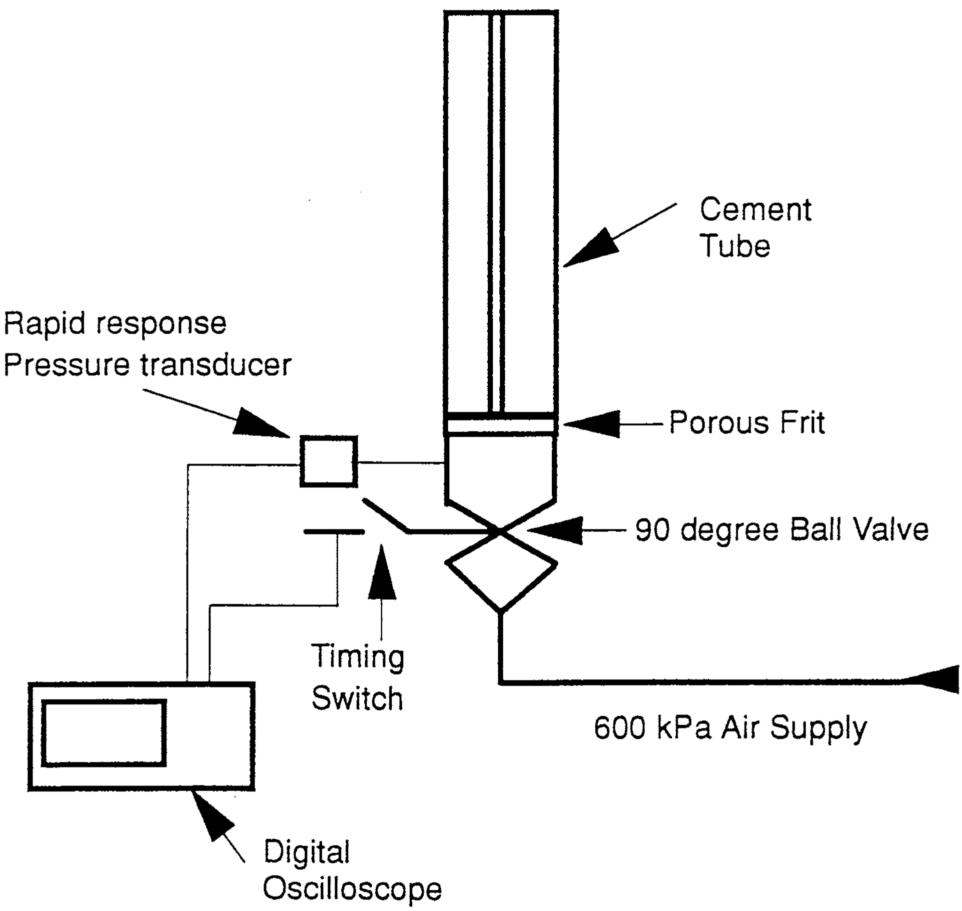


Figure 4.3 Schematic of apparatus used to monitor stemming ejection.

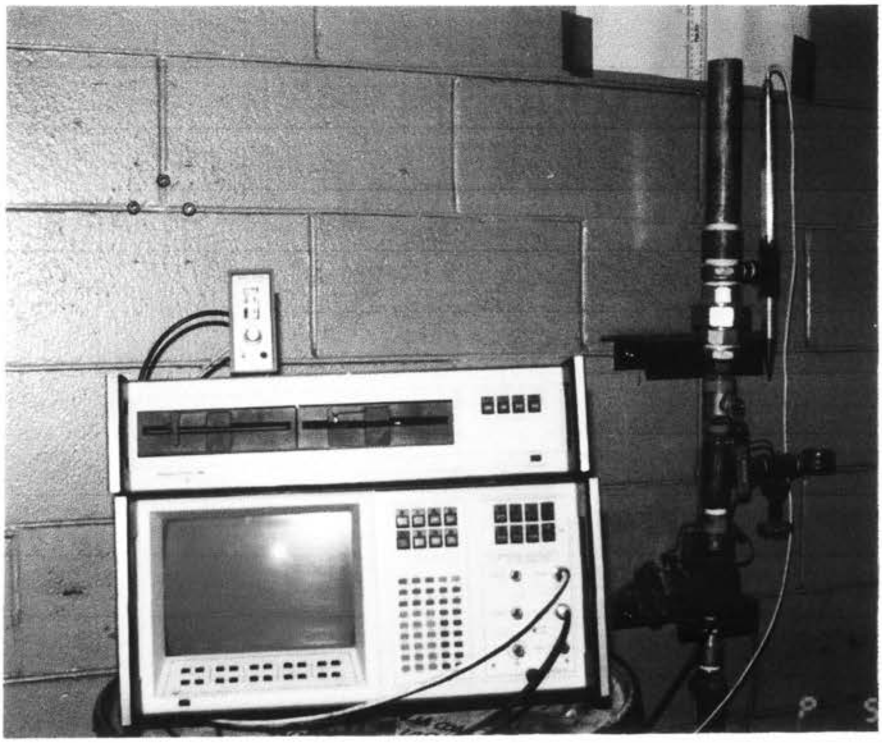


Figure 4.4 Photograph of stemming ejection rig

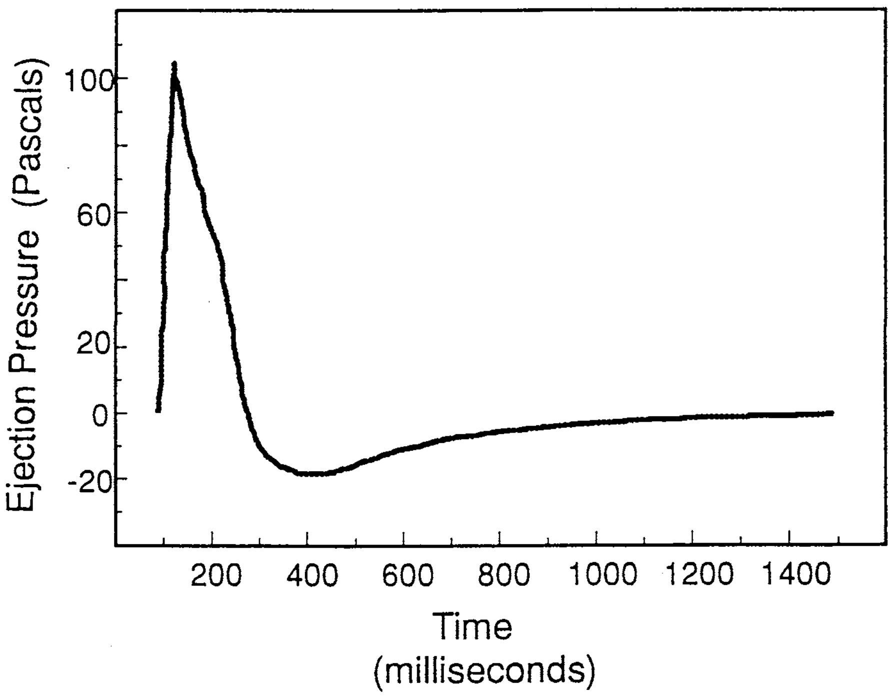


Figure 4.5 A typical stemming ejection rig pressure trace

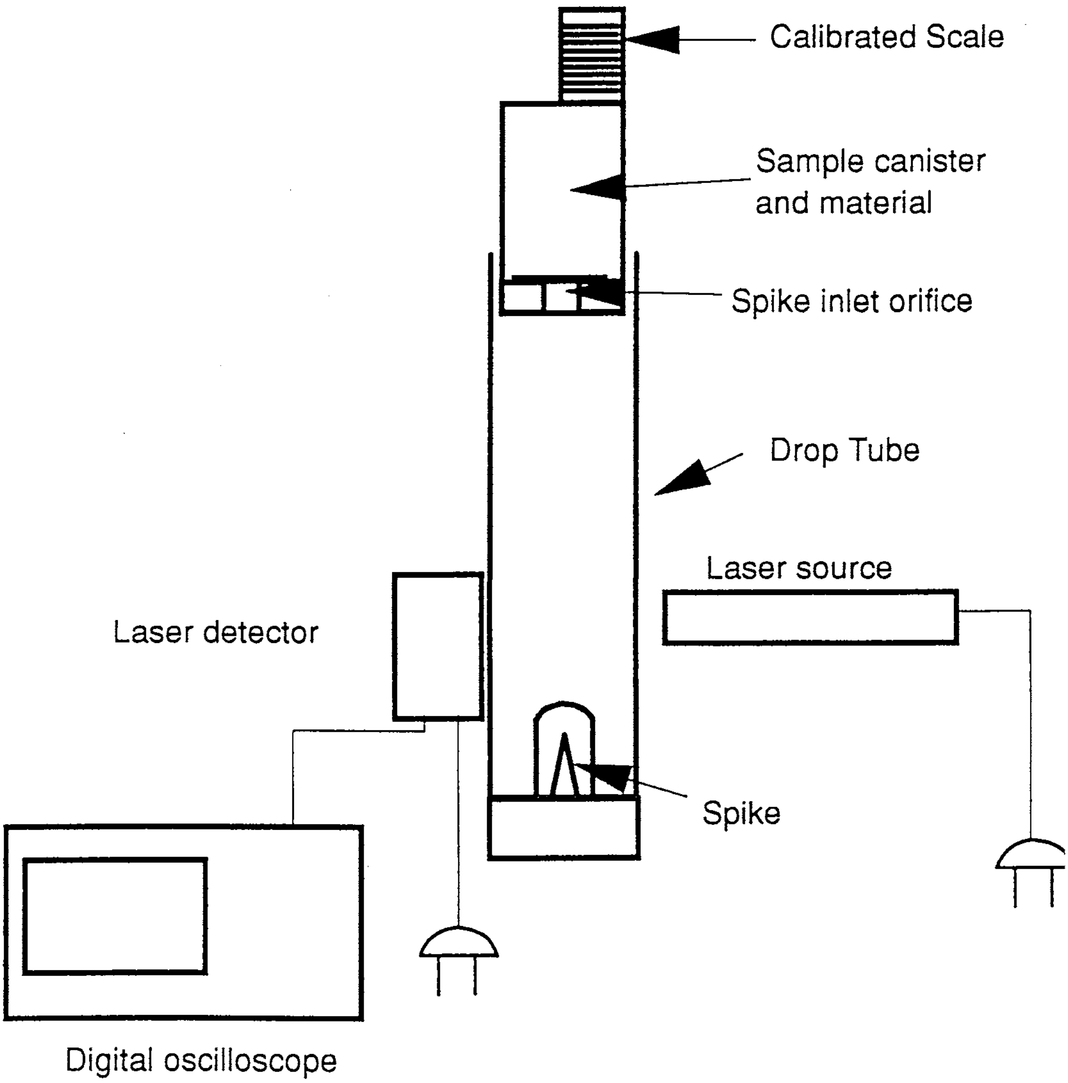
A typical experiment consisted of packing the sample material to the required density in a canister with a paper covered spike inlet orifice. The top of the canister was secured to prevent the sample material being ejected under the action of the penetrating spike. A calibrated transparent scale was attached to the canister top plug and aligned such that the scale markings would interfere with the laser beam as the spike progressed through the sample material. The canister and sample material were raised to the height required, placed inside the drop tube with the spike attached and allowed to free fall onto the spike. A high speed digital oscilloscope was used to monitor the laser beam intensity as the canister was dropped. As the scale passed through the laser beam the intensity of the beam decreased thus a plot of intensity as a function of time was recorded. A typical event as the calibrated scale passed through the laser beam is shown in Figure 4.8.

4.2.5.2.2 Results

When an explosive force is applied to a column of stemming material the initial reaction is for the particles in the immediate vicinity of the force to take up the interstices between neighbouring grains. After this shift in position a more resistive force is applied to the explosive force as the stemming material begins to contain the force of the explosive gases generated. The effect of this resistive force can be measured by the resistance to motion of the spike as it penetrates the confined sample of stemming material.

These results are shown in Table 4.5 , and indicate that the change in velocity with distance is fairly constant for the three materials tested. Also, individually sized fractions of materials show that there is no change in velocity with distance as the spike penetrates the material. The same trend was shown when the mass on the sample canister was increased from 1kg to 14kg, (ie. a change in velocity with distance from -0.474 to -0.487). However, when the sample canister is lifted to 2 metres and dropped, the higher initial canister velocity before striking the spike causes a higher change in velocity with distance (from -0.474 at 05m to -0.728 at 2m).

These results show that the initial re-arranging and resistance to movement of stemming material subject to impact loading is fairly material independent as all sizes and materials show no significant difference. This effect would be experienced by the stemming material as the explosive force is initially applied.



Figue 4.6 Schematic of drop rig apparatus

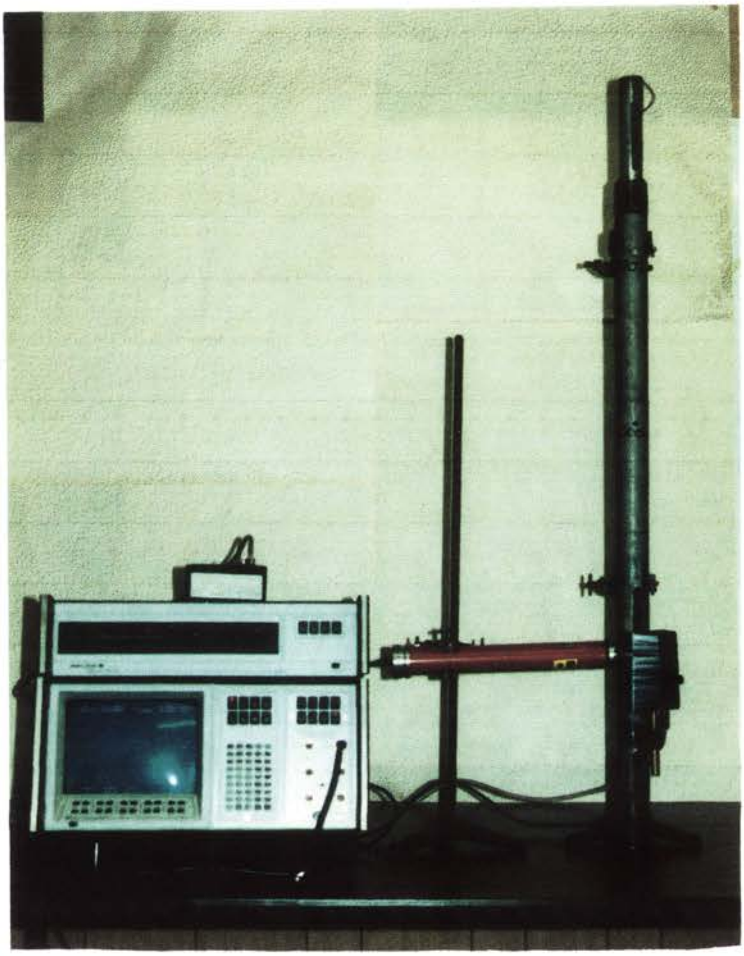


Figure 4.7 Photograph of drop rig

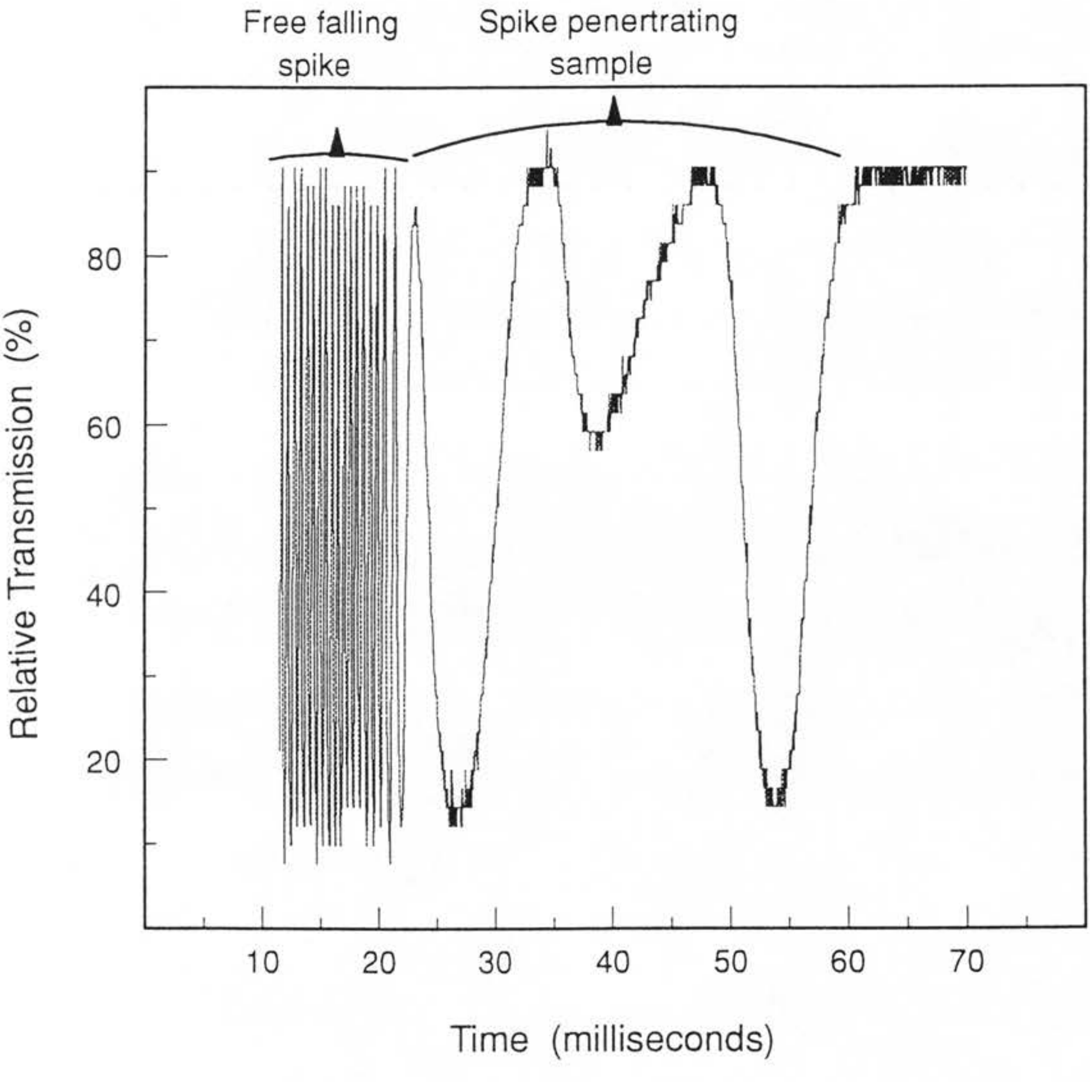


Figure 4.8 A typical drop rig oscilloscope trace

Table 4.5 Stemming Column Under Impact Load Results

Sample	Particle Size (mm)	Packing Density		Velocity Gradient	
		Initial (g/cc)	Final (g/cc)	Average (m/sec)/m	Std.Dev
Limestone	1.50	1.73	1.83	-0.474	0.056
	0.75	1.73	1.87	-0.428	0.080
	0.063	1.73	1.80	-0.383	0.121
	Mixed	1.91	2.16	-0.469	0.072
2m High	1.50	1.73	1.93	-0.728	-
14kg Mass	1.50	1.73	1.95	-0.487	-
Iron Ore	1.50	1.91	2.16	-0.363	0.078
	0.75	1.91	2.19	-0.397	0.074
	0.063	1.91	2.14	-0.412	0.060
	Mixed	2.15	2.32	-0.444	0.115
Sand	0.80	1.73	1.84	-0.424	0.085

4.3 LABORATORY SCALE BLASTING TRIALS

4.3.1 Blast Monitoring Equipment and Confining Medium

Blasting trials were carried out in a blast chamber to measure the effect of :-

a) The gas "release" rate on movement and fragmentation.

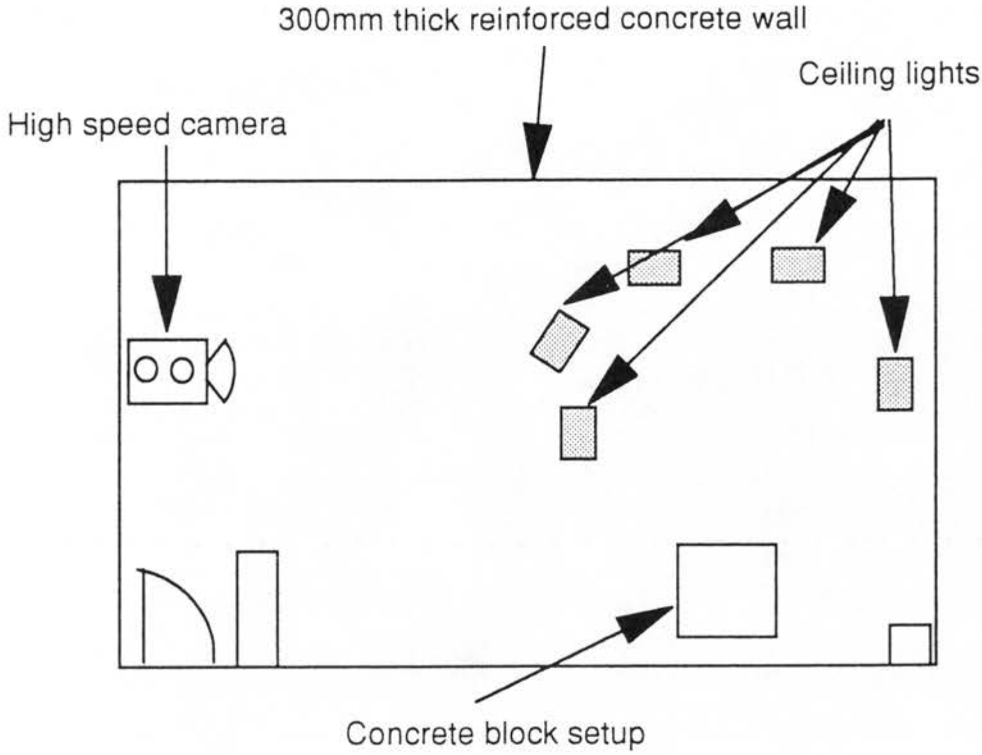
b) The gas confinement effect on movement and fragmentation.

The blast chamber was a reinforced concrete room with 300mm thick walls measuring 4m x 5m floor space and wall height of 3m. This chamber had an explosive rating of 500g of PETN. The walls were protected with wooden slats and the floor was of smooth concrete construction. This chamber was ideal as all the fragmented pieces could be collected for size analysis.

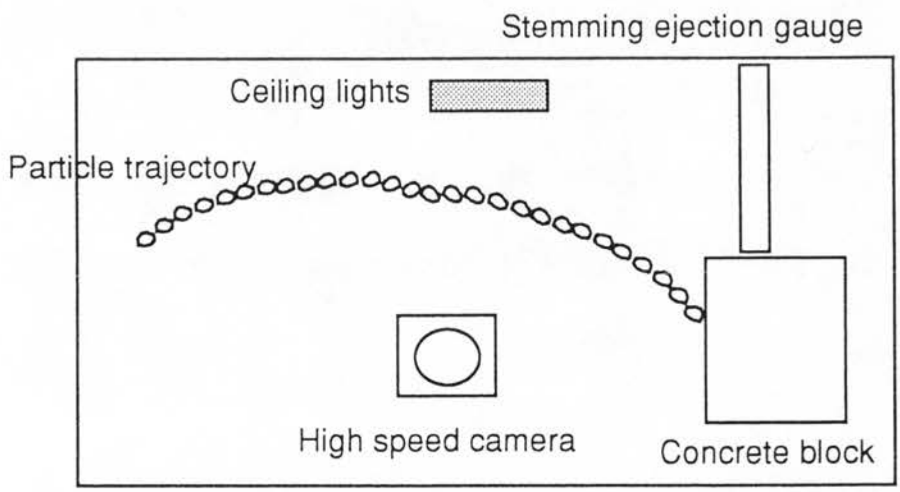
Fragment movement was monitored using a Redlake Locam II high speed movie camera. This camera was capable of film speeds of 500 frames per second and was mounted in a protective housing inside the blast chamber on one wall parallel to the flight plane of the fragmented pieces. The camera was fitted with a wide angle lens (12.5mm focal length) with an aperture of f1.9. Kodak Eastman 7296 16mm 500ASA movie film to capture the image of the individual rock particles through space as a function of distance with time. Lighting was accomplished by 5 x 1500 watt flood lights secured to the ceiling to give an even coverage of light over the expected plane of flight of the fragmented pieces.

The confining medium chosen for all blasting trials was a special concrete mixture which was formed into blocks 500mm x 500mm x 630mm high. During forming of these blocks, lifting points and a 10mm diameter blasthole at the required distance from the free face were set in place before the concrete cured. The wet concrete was vibrated to remove any entrained air then allowed to cure under a moist atmosphere (wet bags) for a period of 7 days before removal from the mould. A further 28 days curing under atmospheric conditions was allowed before the block was used for blasting trials. A numbered grid, 50mm x 50mm, was inscribed on the top and front face of the block so the fragmented pieces could be identified before the size analysis was carried out. Anti-spall plates were cemented to both sides and the rear of the block to prevent spalled material contaminating the fragmented material. Initial movement gauges and stemming ejection gauges, consisting of aluminium foil connected to make/break switching circuits, were put in place on the block. High speed photography markers were suspended from the ceiling and the block was loaded with explosives.

A typical charge consisted of 2-5g of plastic explosive (65% PETN, 15% plasticiser and 20% CN compound) which was tamped on top of an instantaneous detonator placed at the bottom of the blasthole. Stemming material was then placed on top of the explosive charge. A separate instantaneous detonator was connected into the firing circuit to act as a timer for the high speed photography to indicate the frame at which the explosive detonated. A typical blast chamber set up is shown schematically in Figure 4.9 and photographically inside the blast chamber in Figure 4.10.

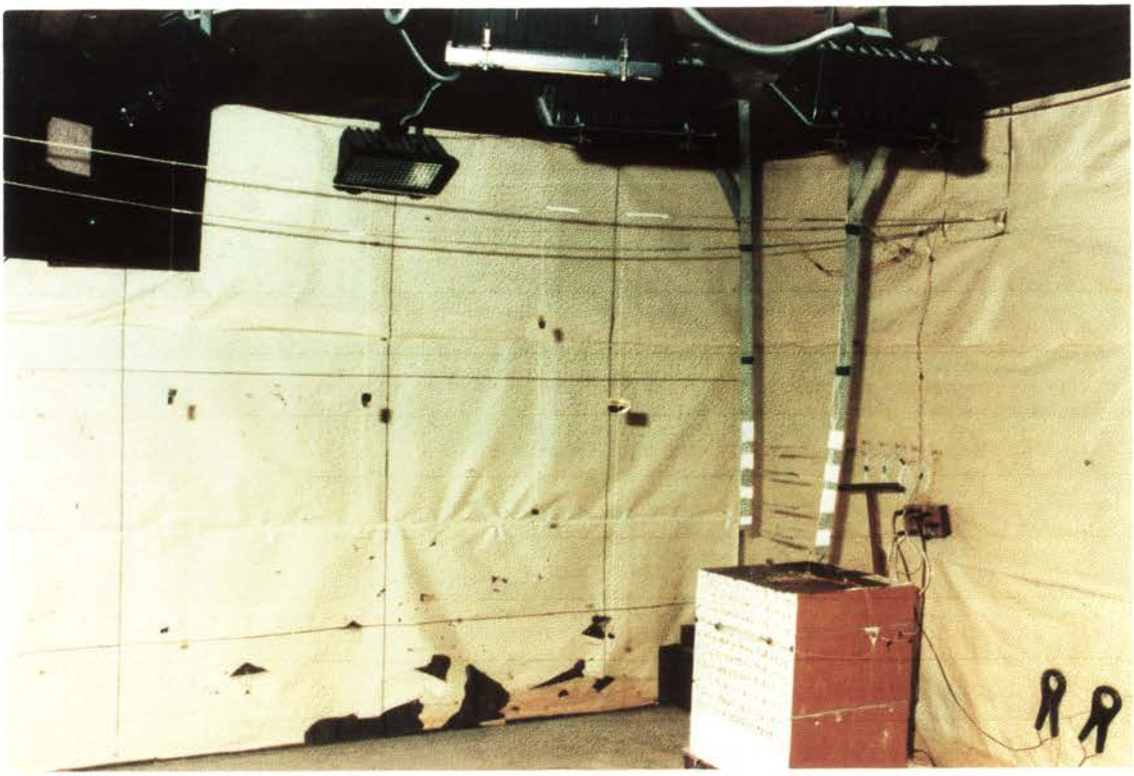


Plan view of blast chamber



Elevation view of blast chamber

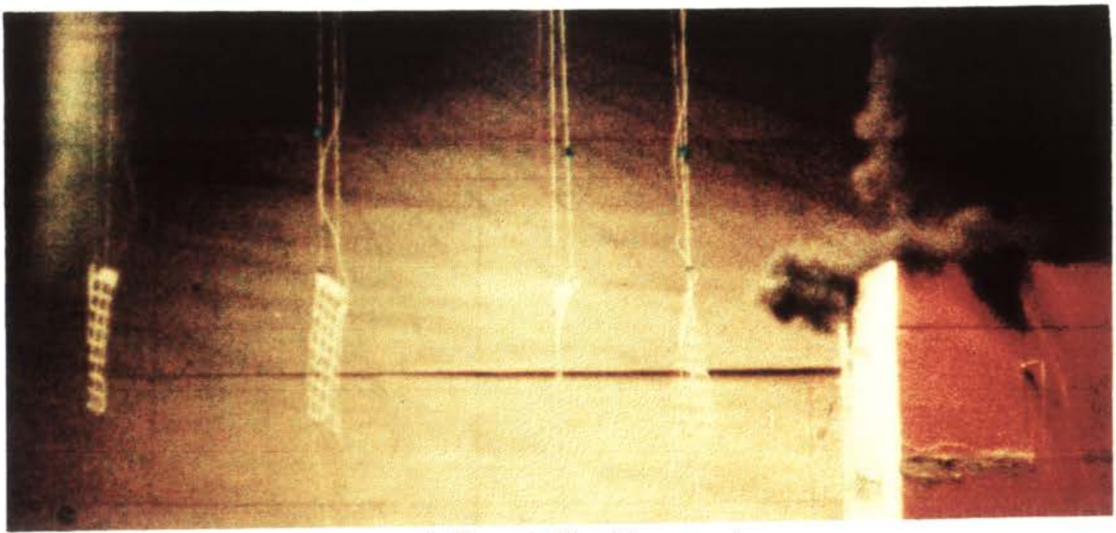
Figure 4.9 Schematic of blast chamber layout



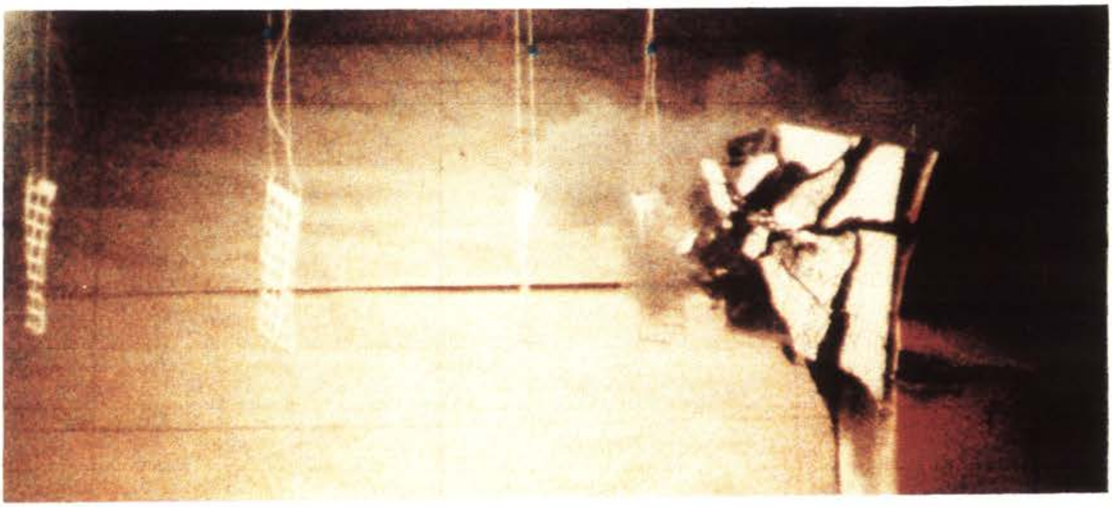
a) Inside blast chamber

b) Concrete block setup

Figure 4.10 Blast chamber and concrete block setup



a) Time t=2 milliseconds



b) Time t=10 milliseconds

c) Time t = 20 milliseconds

Figure 4.11 Fragment movement from high speed photography

A sequence of images from the high speed film at various points in time is shown in Figure 4.11.

4.3.2.1.Explosive Gas "Release" Effect

It has often been observed that the explosive gases produce the energy required to throw or heave the fragmented material. This hypothesis was tested in a series of experiments designed to release the explosive gases from the blasthole at a "controlled" rate and monitor the performance of movement and fragmentation. A steel plate, 12mm thick, was bolted to the top of the concrete block and a hole drilled in the plate at the precise location of the blasthole. The diameter of the hole in the plate can be varied to release some of the gas energy before it performed useful work.

4.3.2.2 Results

A summary of the results of "controlled" gas release rate is shown in Table 4.6 and graphically in Figure 4.12 and Figure 4.13. The hole diameter for controlling gas "release" rate was varied from 0mm to 10mm for a blasthole diameter of 10mm. In all tests the burden was kept constant and the hole diameter was selected to decrease in area by approximately 50% from the previous test.

The fragmentation results in Figure 4.12 show a decrease in the mean fragmentation size as the release rate hole diameter decreased. The mean fragmentation is the K_{50} of the size distribution (ie, 50% probability of passing a certain aperture) of the fragmented material. Thus as the explosive gases are retained inside the blasthole for a longer period of time more work is done on the fragments producing a smaller K_{50} hence a smaller size distribution muck pile.

The burden movement results are shown in Figure 4.13 indicating a similar trend as above. An increase in energy retained in the blasthole causes more work to be done on the broken material. Burden movement is the initial velocity of the fragmented particles and as the gas release rate hole diameter decreases the burden movement increases. Thus more work was done on the fragmented material in throwing the pieces at a higher velocity hence a longer casting range would also have been attained. These velocities were found to be high compared to some practical cast blasting values but this would be expected in this laboratory study as a high powder factor of 1 kg/tonne was used in these experiments because of the "relatively" competent nature material.

Table 4.6 Gas Release Rate Results.

Hole Diameter (mm)	Hole Area (mm^2)	Burden Velocity (m/sec)	Fragmentation (K_{50} mm)
0	0	26.1	56
1	0.8	25.2	54
5	19.7	21.5	65
7	38.5	20.4	74
10	78.5	19.5	117

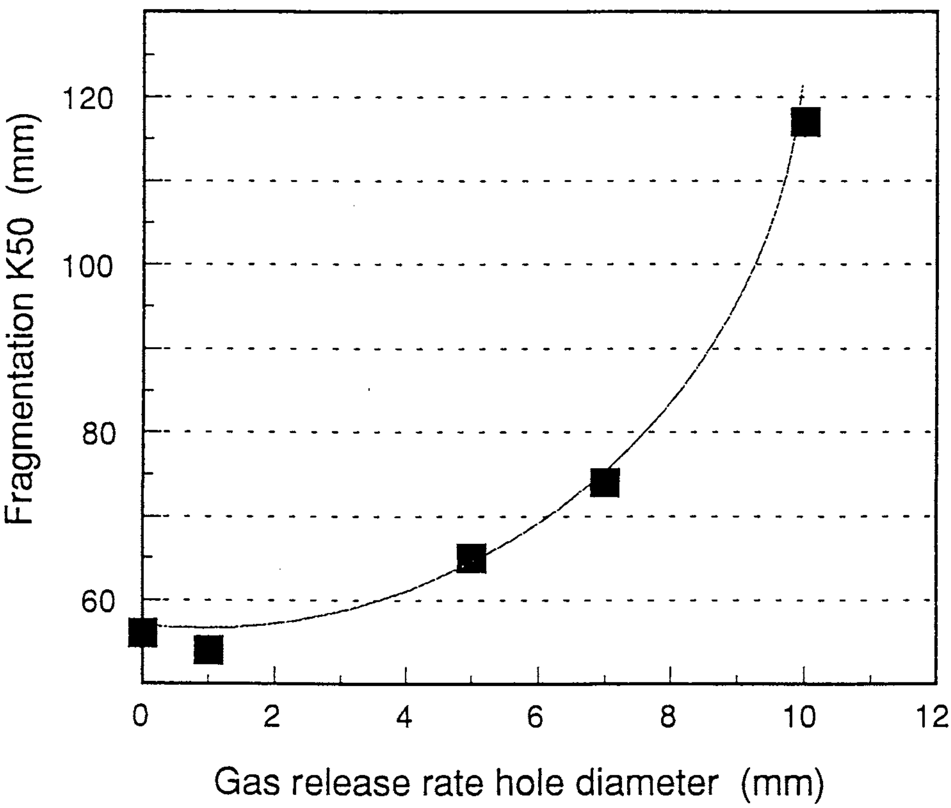


Figure 4.12 The effect of gas release rate on fragmentation

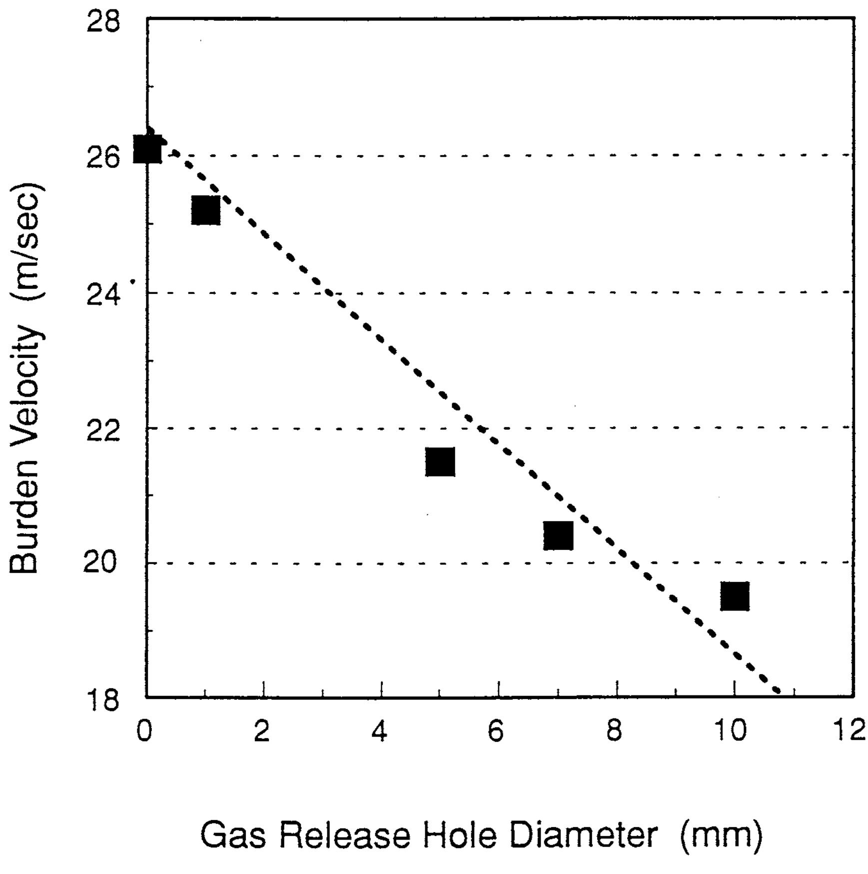


Figure 4.13 The effect of gas release rate on burden velocity

4.3.3.1 Gas Confinement Effect

The results above show that containing the gases for longer periods increases the work done on the confining medium by producing finer fragments and throwing the fragments further. Normal stemming practices rely on fine particles such as drill cuttings being placed on top of the explosive column to contain the explosive gases. So how do different types of stemming materials confine the gases and effect the efficiency of the blast?

4.3.3.1.1 Stemming Material Effect

Several stemming materials and methods were ranked by a modified ejection rig test similar to that described in Section 4.2.6.1. These materials and methods were then used as stemming in laboratory blasting trials. The stemming materials and methods were ranked on their resistance to ejection so a relative degree of stemming strengths or ability to retain the explosive gases could show how effective the stemming materials are on blasting efficiency.

4.3.3.1.2 Results

The results of confinement pressure and its effect on blasting efficiency are shown in Table 4.7 and Figure 4.14 and Figure 4.15. The fragmentation results in Table 4.7 and Figure 4.14 show a sharp decrease in fragmentation as the material ejection pressure increases slightly. When this pressure is further increased to total confinement no significant decrease in fragmentation is evident. This effect is also shown when a cork is placed in the blasthole with various masses to act as a resistive force to ejection. A small force was found to be sufficient to contain the gases long enough for more useful work to be performed.

A similar trend is shown in Figure 4.15 for the burden velocity. As the ejection pressure increases burden movement velocities also sharply increase followed by a plateau. Thus as the stemming material offers more resistance to ejection more work is done on the confining medium by increasing the burden velocity.

Table 4.7 Ejection Pressure effect on Blasting Efficiency

Stemming Material	Block Strength (MPa)	Ejection Pressure log(kPa)	Fragmentation (mm)	Burden Velocity (m/sec)
Nil	20	2.04	93	5.5
Sand	20	2.08	54	12.5
Limestone	20	2.66	51	14.9
Cement	20	4.30	51	14.9
Nil	40	2.04	117	17.4
125g Mass	40	2.05	59	18.0
250g Mass	40	2.07	51	19.2
Sand	40	2.08	53	20.2
Limestone	40	2.66	54	22.6
Cement	40	4.30	52	25.5

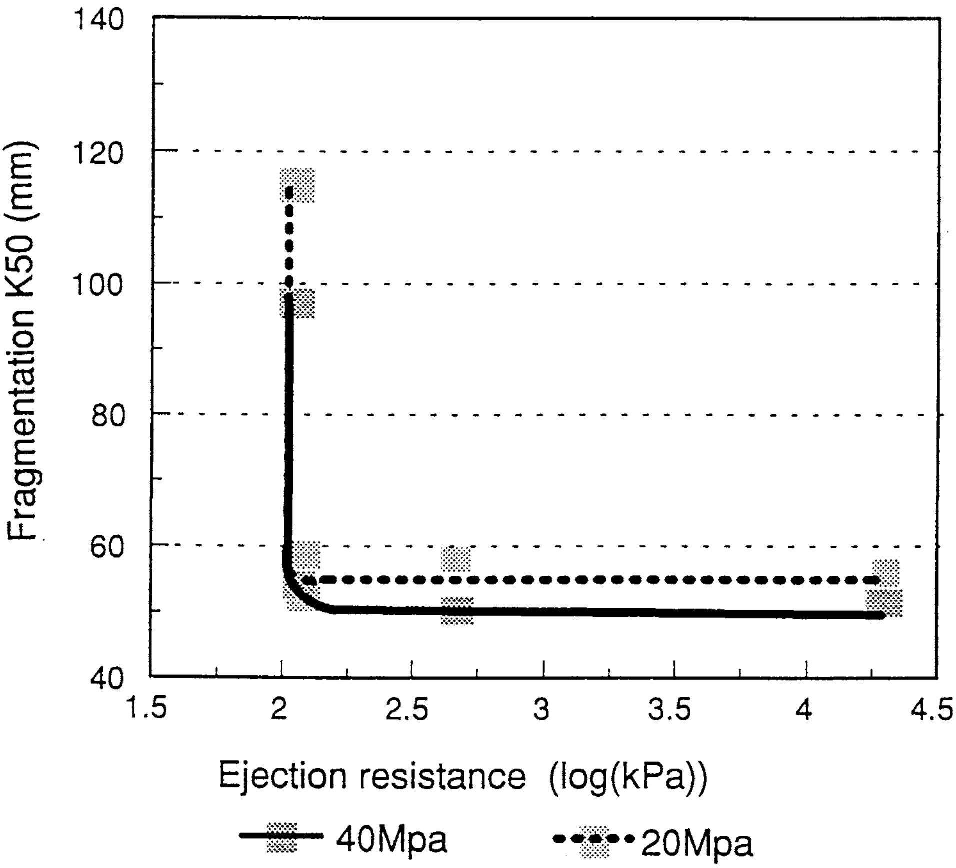


Figure 4.14 The effect of stemming ejection pressure on fragmentation

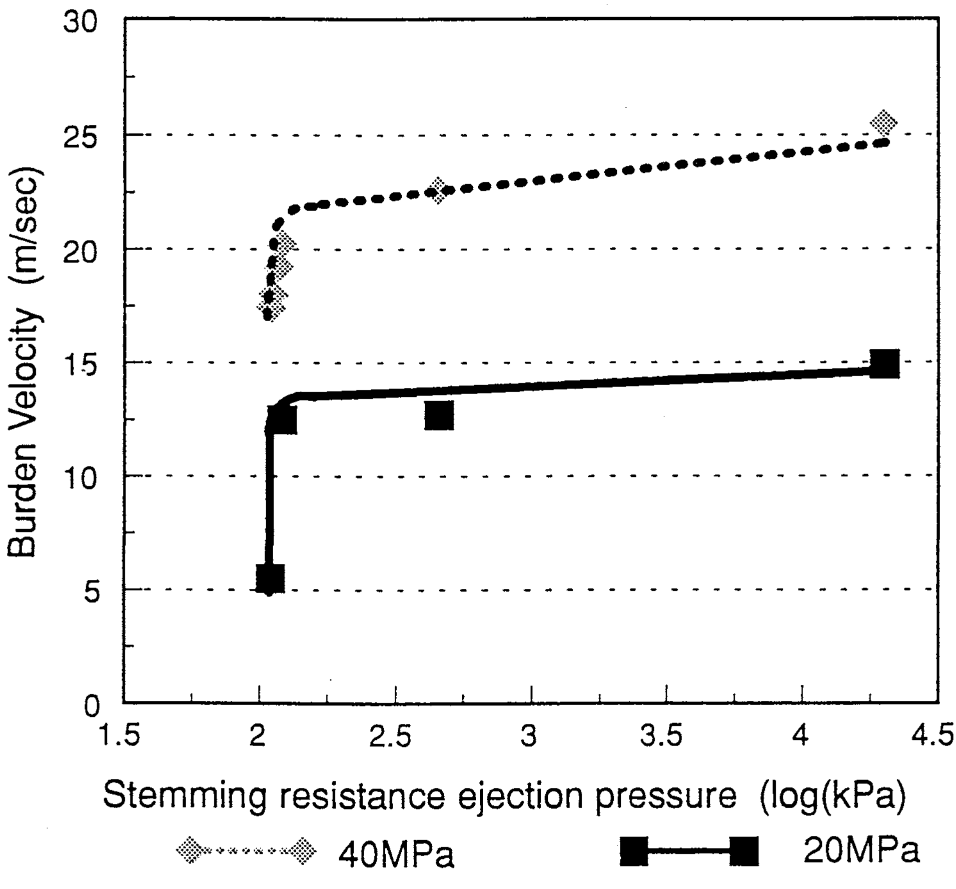


Figure 4.15 The effect of stemming ejection resistance pressure on burden velocity

4.3.3.2.1 Stemming Material Size Distribution

If the stemming material particle size is small compared to the blasthole diameter then as a force is applied to a column of this stemming material it could behave as if it were a fluid and be easily ejected from the blasthole. A series of laboratory scale blasting trials was conducted where the stemming material particle size was decreased and the fragmentation and burden movement monitored. In addition the velocity of the material as it was ejected from the blasthole was monitored.

4.3.3.2.2 Results

The results of the effect of stemming material particle size are summarised in Table 4.8 and also in Figures 4.16, 4.17. As the results show, for both the 10mm and 12mm diameter blastholes, stemming ejection velocity decreases as the particle size of the stemming material increases. This would indicate that the material is "locking" as the particle size increases providing some resistive action against ejection hence reducing the ejection velocity. This trend is also evident for burden movement, see Figure 4.17. As the stemming particle size increases, burden movement velocities increase indicating that more useful work is being imparted to the confining medium and not wasted in ejecting stemming material. Fragmentation does not appear to change significantly for both the 10mm and the 12mm diameter blastholes which would tend to indicate that the energy had been imparted to the confining medium and performed useful work in fragmentation before stemming ejection began.

Table 4.8 Stemming Ejection Velocities.

Particle Size (mm)	Blasthole Diameter (mm)	Ejection Velocity (m/sec)	Fragmentation (mm)	Burden Velocity (m/sec)
0.019	10	131.1	73	13.3
0.076	10	105.1	77	16.3
0.187	10	90.2	75	18.8
0.800	10	81.8	78	20.5
0.041	12	126.2	61	11.7
0.098	12	90.9	63	13.1
1.100	12	49.9	61	15.5

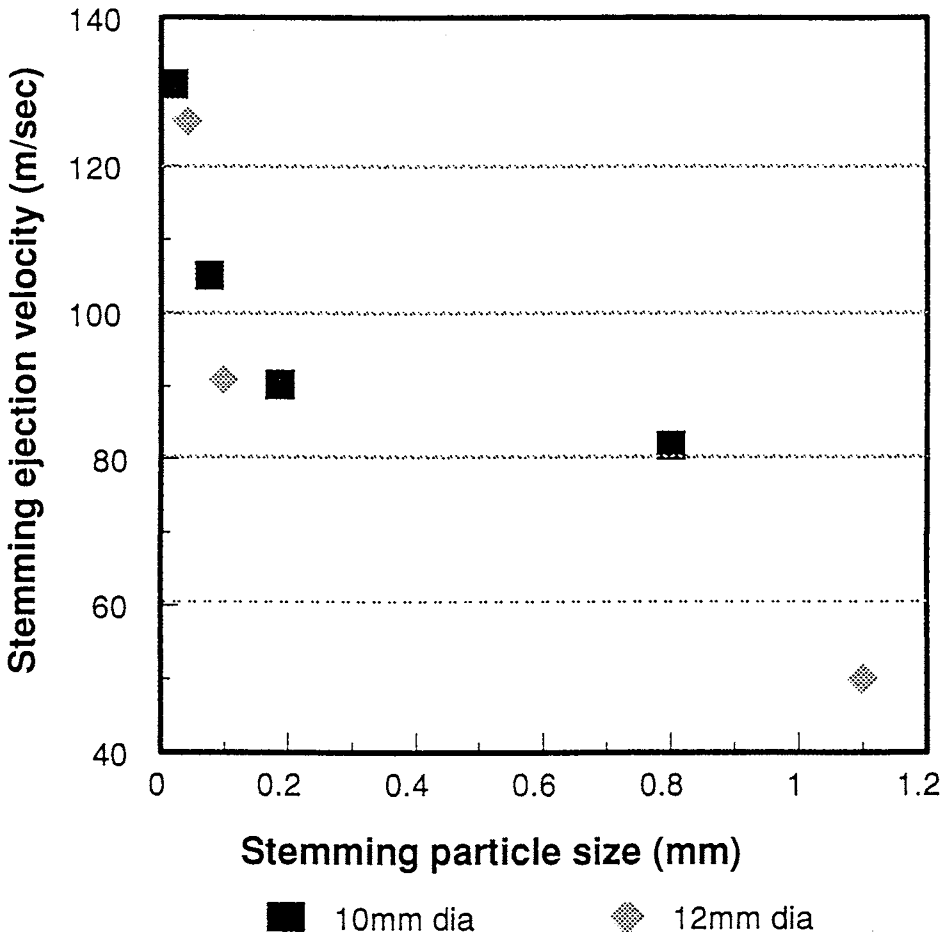


Figure 4.16 Stemming ejection velocity as a function of stemming particle size

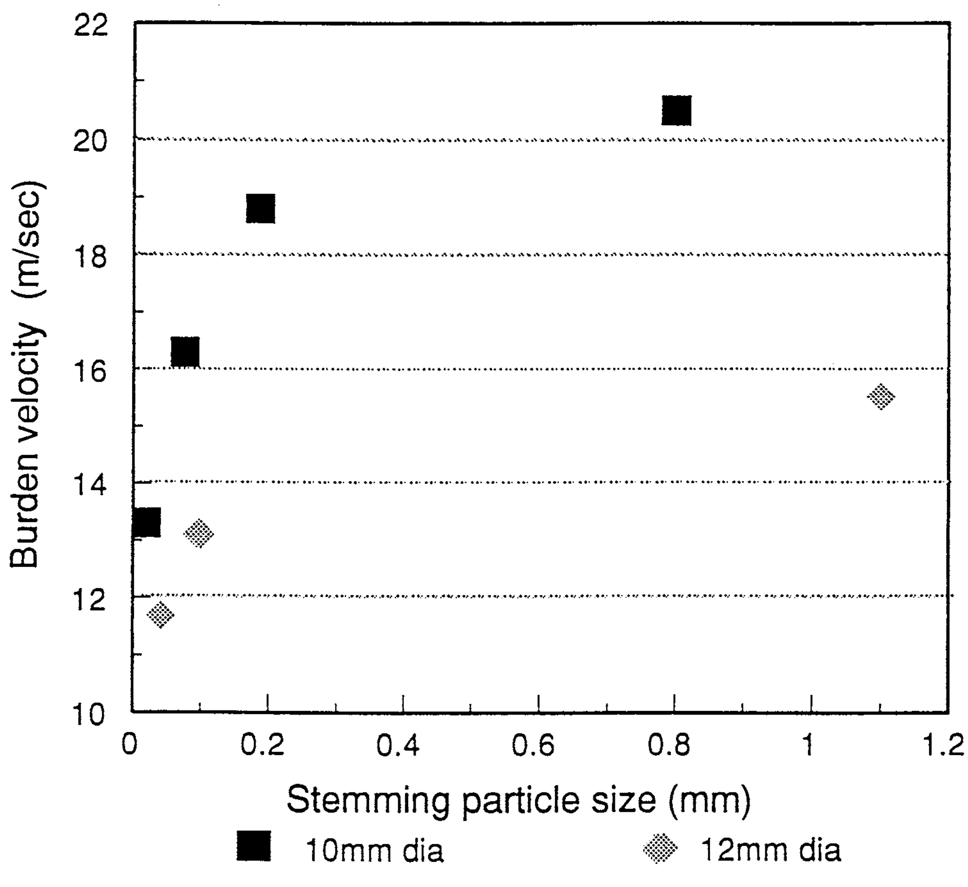


Figure 4.17 The effect of stemming particle size on burden velocity

4.3.3.3.1 Increase in Explosive Energy

It has been stated that the gas energy provides the thrust or throwing power of an explosive charge. To test this hypothesis a series of laboratory scale blasting trials were designed such that the explosive load was increased but the gas pressure in the hole was kept constant for a given stemming height and constant stemming particle size to blasthole diameter ratio. The thought behind this series of tests was that with a constant gas pressure similar burden velocities and fragmentation would be observed.

4.3.3.3.2 Results

The results of increasing the explosive loading in the blasthole are summarised in Table 4.9. The gas generated from the explosive used was assumed to be 1 litre/gram of explosive and the calculations show that the pressure in the blasthole of each of the three tests was the same. This calculated pressure neglects the pressure effect of temperature of the explosion.

As the results in Table 4.9 show both the monitored parameters of burden velocity and fragmentation indicated an increase in energy transfer to the confining material as the explosive load increased. This was also supported by the explosive cavity cross sectional area which increased from 78 mm^2 to 153mm^2. The stemming ejection velocities were not significantly different and any difference could be due to the variation in the stemming particle size to blasthole diameter ratio observed.

Table 4.9 Increase in Explosive Loading Results

Blasthole Diameter (mm)	10	12	16
Explosive Mass (g)	2	3	5
Explosive Volume (mm^3)	2900	4350	7250
Gas Evolved (litres)	2	3	5
Gas Pressure (MPa)	69.7	69.7	69.7
Fragmentation (K_{50} mm)	115	81	47
Burden Velocity (m/sec)	12.4	13.1	20.1
Stem Ejection Velocity (m/sec)	95.1	90.9	95.3
Particle/Blasthole Ratio	131	122	116
Explosive Cavity Area (mm^2)	78	118	153

CHAPTER 5

FIELD STUDY

5.1 INTRODUCTION

The properties of stemming materials applied in a practical sense were examined by monitoring the blast performance in a rock quarry. Modern blast monitoring equipment such as high speed photography and laser profiling techniques were used to monitor production blasts at a quarry. Some of the properties of the stemming material investigated in the laboratory were examined in these field scale trials. One important parameter observed in the laboratory was the effect of the stemming particle size to blasthole diameter ratio on the stemming ejection velocity. The blasthole configuration of a series of front row holes was varied, and the effect captured by the monitoring equipment. Two production shots were monitored in this way with a coarse size and fine size stemming material together with a control set of holes for comparison.

5.2 BLAST PATTERN LAYOUT AND MONITORING SETUP

A rock quarry on the Central Coast of NSW was the location of the field blasting trials. This quarry has production blasts on a regular basis and in the particular area monitored two types of rock were being quarried. The rock type is basically a conglomerate which is fairly well cemented with a weaker weathered layer on top of the more competent broad band of conglomerate rock. The quarry is a truck and FEL operation producing approximately 10000 tonnes of crushed rock per week. The area is usually dry, and wet holes only occur after abnormally wet weather conditions. Wet holes are dewatered by blowing out the water with compressed air or plugging the water with cartridges of emulsion products. The quarry predominantly uses bagged ANFO which produces the desired heave and fragmentation for the operation.

5.2.1 Blast Pattern.

An Ingersoll-Rand 350 Crawlair drill powered by a 850 cfm compressor was used to drill blastholes of 76mm diameter for a face height of upto 15 metres. This type of drill employs a rotating hammer action with air purge to pulverise the rock and blow the drill cuttings to the surface. This drill is efficient and drilling rates of up to 60 metres per hour, depending the ground type, are not uncommon.

Typically the blast pattern is a series of holes in a square configuration on a burden and spacing of 2.1m x 2.4m. This pattern is "opened up" if softer ground is encountered and this option is entirely at the discretion of the operator. Blastholes are drilled in the rock at an angle of 15° to the vertical towards the free face of the pattern. This angle has proved to be successful in the past at providing safe walls where excavation equipment can work safely to remove the broken rock after the blast. Also, this angle helps in "throwing" the broken rock to produce a flatter muck pile for the FEL equipment.

After the blasthole has been drilled the hole is sealed with a bore plug, a piece of rag filled with drill cuttings, to minimise the ingress of surface water and drill cuttings. The number of holes in the pattern depends on production requirements and pattern location in the grade of rock being blasted. A typical pattern for this quarry is approximately 50-60 holes although patterns with up to 200 holes have also been drilled.

When the desired pattern has been drilled the blastholes are loaded with explosives. A typical dry hole charging configuration consists of a nonelectric delay detonator of 200 milliseconds down the hole coupled to a booster of emulsion is dropped into the base of the hole. ANFO is then added to a collar height of 1.5m and then covered with drill cuttings and tamped as the stemming. When a small amount of water is present in any hole extra cartridges of emulsion product are dropped in the blasthole to a height above the water level and then ANFO is added to the explosive column height of 1.5m. The same loading procedure of stemming material was adopted.

The blastholes in the pattern were then "tied-in" to an initiation sequence so that the rock is fragmented and thrown to form the desired muckpile for the excavation equipment being used. A V1 tie-in sequence was typically used but variations to this sequence are also used depending on the final pattern shape. Surface delay elements of 17 milliseconds within an echelon or row, and 42 milliseconds between rows are typically used. All surface delays and in hole delays are of the nonelectric signal tube type and the lead-in line is connected to an instantaneous electric detonator which is fired by a generator type shot exploder.

5.2.2 Monitoring Equipment.

A section of the front row of holes of the blast pattern was used for the experiments. Two to three holes for each stemming material size were used in each experiment. A group of 8 to 9 holes constituted an experiment with 2 to 3 holes set up as per normal being used as the control set. One hole in each set was made up with markers for the monitoring equipment.

The configuration of each of the test holes was closely monitored. Each hole depth, explosive loading and collar depth was measured, and the screened stemming material was carefully placed in each of these holes. A typical experimental pattern lay out is shown in Figure 5.1. The face markers were painted on the rocks and these rocks followed in flight till they were obscured by dust. The trajectory plane markers were four bags filled with dirt placed in the trajectory plane of the moving rock which were surveyed in position prior to the blast being initiated. The hole collars markers were bags filled with a small quantity of dirt to mark the hole to monitor any stemming material that might be ejected as the movement was in progress. All of these markers were located in relation to a reference point by a laser profiling system.

The performance of an explosive in a blasthole can be monitored by measuring the velocity of detonation (VOD) of the explosive. Much published literature is available for commercial explosives regarding their performance in many types of ground. The VOD of an explosive is measured by inserting a coaxial cable in the explosive and measuring its length as a function of time during the detonation of the explosive column. Prior to loading the explosive in the hole the coaxial cable is taped to the primer and lowered into the blasthole. The explosive is then loaded and the stemming material added while keeping the coaxial cable taught. The coaxial cable is then connected to the electronics of the VOD system and armed ready for the hole to be initiated.

After the blast pattern was loaded and all the markers were in place, these marker locations were surveyed so that analysis of the high speed film could be undertaken. A laser profiling system was used to carry out this duty. The laser range finder sends out a light signal which is reflected from the rock surface back to the instrument and the time of flight measured. The instrument also has a vertical and horizontal encoder so that the coordinates of all measured points (horizontal angle, vertical angle and distance) from the instrument set up position can be determined. The instrument can store up to 1700 individual points and

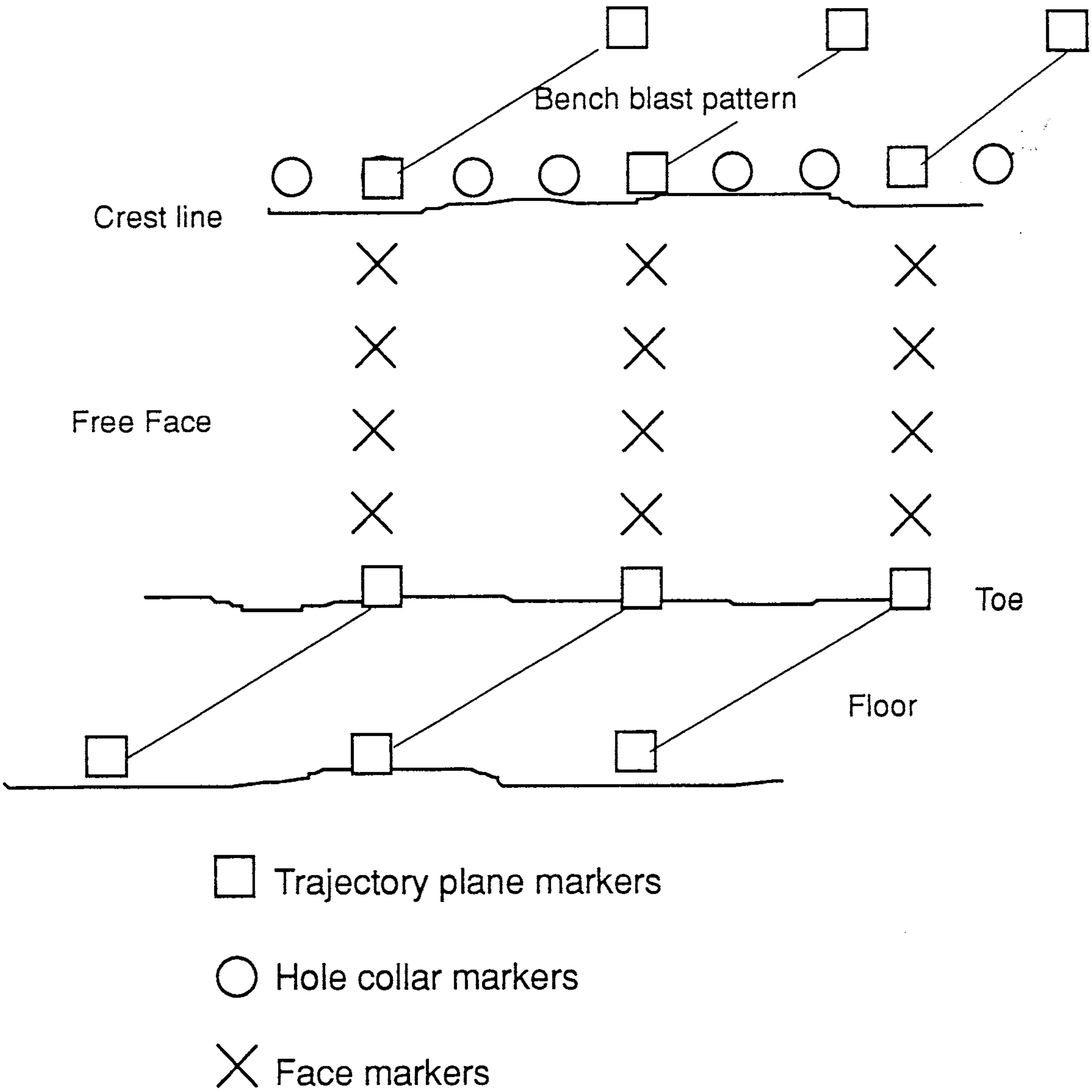


Figure 5.1 Typical experimental blast pattern layout.

a traverse of the face of the "shot" is used to highlight any areas of small burden where face bursting could potentially occur so that corrections to the loading of explosives could be made. The crest line, toe line and hole locations for the pattern was also located to completely define the explosive location in relation to the rock to be broken.

High speed photography was used to monitor the performance of explosives and its effect on the movement of the broken rock. When a blast pattern is loaded with explosives and then initiated the time frame over which an entire event is completed (from surface initiation to final muckpile formation) is in the order of 5 seconds. If a normal movie camera was used to monitor such an event then at 24 frames per second only 120 frames or samples would be taken to represent parameters such as timing sequence, initial movement, angle of movement and stemming ejection. At this rate each frame would represent one image each 42 milliseconds. In quarry blasting where a small number of holes are fired, 42 milliseconds represents the time between rows or up to 3 holes in a row being fired. A movie camera capable of framing rates up to 500 frames per second is able to capture an image each 2 milliseconds which is well within the event times in a blast. A high speed camera with a film canister holding 100feet of film and a zoom lens allow monitoring to be taken at a safe distance from the blast while still capturing a clear image of adequate resolution.

An important parameter used to design the tie-in sequence used for a blast is the initial movement time. The precise moment the explosive column is initiated has to be identified and when the blastholes are tied-in to the initiation sequence, an extra delay detonator of the same delay time as that down the hole is thrown over the face at the hole location. This timing marker indicates the time at which the explosive column is initiated and is picked up on the film.

When all of the film markers have been surveyed and the timing markers have been connected to the tie-in sequence the high speed camera is checked and adjusted for the correct light exposure. As the camera requires approximately 2 seconds to attain the correct operating speed, communication with the shot firer was essential. A countdown from 5 with the camera being switched on at 2 was found to be a successful practice.

5.2.3 High Speed Film Analysis.

The exposed film taken from the blast was developed and a light work print made from which the analysis of the blast parameters was carried out. From this print the location of the film markers, the trajectory of the film markers, the frame numbers at which events occur and the film speed, marked by a timing light generator inside the camera, were logged on a plain sheet of paper for each event being monitored. These marker location with respect to time were digitised using a digitising board connected to a computer. The computer file generated from the digitising operation and the coordinates of the film markers prior to the blast being initiated were used as inputs to a software package which calculates the blast parameters required.

The assumptions and the procedures used in the software package to determine the blast parameters are as follows. Reference is made to Figure 5.2 for the marker locations used in normal high speed film monitoring of a blast. Markers C1 to C4 are the control markers which define the plane of reference through which the front face markers M1 to M4 will move with time during the blast. The high speed camera is positioned as near as possible to be normal to this reference plane and angles of up to 10° from the normal to this plane can be shown to produce minimal errors. Correction for this camera angle to the reference plane are taken into account in the software.

The equation for a plane is given by

$$Ax + By + Cz = D \qquad (5.1)$$

where x,y and z are the dimensional coordinates of the control markers. If 4 points are known then by a series of simultaneous equations the plane of reference can be defined. Now with the plane of reference defined, movement with time relative to this plane can be calculated. When the explosive is initiated and the rock mass begins to move the film markers M1, M2, M3 and M4 are located in the plane for each time frame of interest. Thus the distance travelled over any time period can be calculated and the burden velocity and ejection angle can be measured.

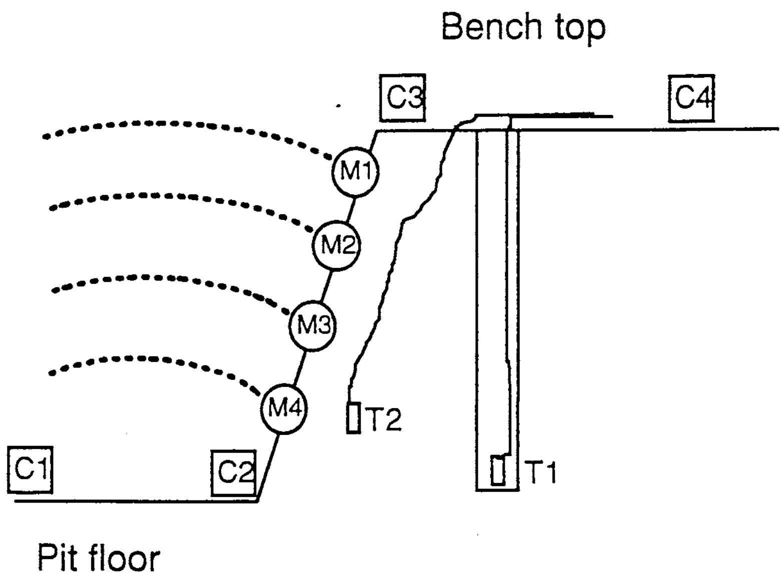


Figure 5.2 Typical high speed film marker locations

Another parameter of interest is the initial movement time of the burden material. When a blasthole is initiated generally the movement of the material is in the direction of the nearest face. So in multirow blasts the front row should be moving and detached from the second row before the second row begins to move. And so on. If the initial movement time is known then appropriate delay times can be sequenced into the pattern to allow for this detachment and movement for efficient blasting results. The initial movement time is determined from the high speed film and is the time difference between the explosive column initiation and the burden moving.

In Figure 5.2, T2 is the initial movement timing marker which is a delay detonator of the same delay time as T1 which initiates the explosive column. This initial movement time is site specific and is related to the structure of the rock mass.

5.2.4 Application.

As with any monitoring scheme the end result is going to be.... How can these measurements be used? In a practical sense the blast parameters measured from the monitoring exercise are used to modify explosive type, blasthole burden or stemming height for subsequent blasts. More efficient blasting is achieved when the blast properties are optimised and cast blasting can benefit from this optimisation.

So, blasting can throw the fragmented material across the pit to form an optimum muckpile which can be of enormous benefit. As stated above parameters such as burden velocity and ejection angle can be determined from the high speed film. A parameter of interest in cast blasting is the casting range which is calculated as follows.

$$R = V_o * \cos\theta * [V_o * \sin\theta + \sqrt{(V_o * \sin\theta^2 + 2gh)}] \;/\; g \qquad (5.2)$$

where R is the casting range, Vo the burden velocity, theta is the ejection angle h is the height above the pit floor and g is the acceleration due to gravity. It can be recognised that monitoring the blast with a high speed camera can yield valuable information that can be used to improve subsequent blasts.

5.3 RESULTS

Two production blasts were monitored at the rock quarry on the 18th February, 1994 and 7th April, 1994. These shots were in the weathered material and the more competent conglomerate rock respectively and the blast pattern parameters were similar. These parameters are detailed in Table 5.1. Both shots were fired on fine days, and noise and flyrock posed no problems. The quality of the blasts were deemed satisfactory and no problems were experienced digging the blasted rock.

Table 5.1 Production blast pattern parameters.

Blast date	18-02-94	7-04-94
Burden (m)	2.1	2.1
Spacing (m)	2.4	2.4
Hole diameter (mm)	76	76
Inclination (deg)	15	15
Bench height (m)	6	14.5
Hole depth (m)	6.5	15.0
Explosive height (m)	5.0	13.7
Stemming height (m)	1.5	1.3
Number of holes	134	96
Volume of rock (m^3)	4400	7300
Inter-hole delay (msec)	42	42
Intra-row delay (msec)	17	17
Mass of explosive (kg)	2412	4116
Powder factor (kg/m^3)	0.48	0.65

5.3.1 Stemming Materials

The stemming materials normally used at the rock quarry was drill cuttings from the blasthole. In both of the blasts monitored dry holes predominated and not a lot of emulsion plugs were needed to compensate for water in the holes. No holes had water above the explosive column so all the stemming material was dry.

The physical characteristics of the stemming materials used in the experiments is shown in Table 5.2. "Normal" drill cuttings were screened at 1.4mm to produce a Fine Fines and Coarse Fines and the size distributions are shown in Table 5.2. A relatively consistent Fine Rock, Coarse Rock and Sand were also used to investigate the gas release effect on the stemming and burden movement initial times. All size distributions are shown in Table 5.2.

All stemming materials were dry before loading into the blastholes as the effect of water adds to the resistance of the stemming to ejection from the blasthole. The range of stemming particle size to blasthole diameter shown in Table 5.2 varied from 1:150 for fine drill cuttings to 1:3 for the coarse rock which is the size ranges that would be experienced in normal production blasting practices.

5.3.2 Blasthole Configuration

All test holes were monitored for hole depth, explosive column height, water, stemming height and primer and these details are shown in Table 5.3.

The blasthole configurations within each blast were similar and in all of the test holes ANFO was the predominant explosive used. Only in one hole, the Fine Fines hole, on the 18-2-94 was excessive water observed and a base charge of emulsion had to be used in this case. The blast dated the 18-2-94 was fired in the highly weathered soft conglomerate material which had a bench height of approximately 6.0 metres. The blast dated the 7-4-94 was fired in more

Table 5.2 Stemming size distribution (cumulative % passing)

Type	Normal	Fine Fines	Coarse Fines	Normal	Sand	Fine Rock	Coarse Rock
Date	18-02-94			7-04-94			
Aperture(mm)							
32							98.0
25							48.3
20						100.0	2.5
12						96.9	
10	100.0		100.0	100.0		66.6	
8	98.3		94.3	98.7		4.6	
4	87.6		69.8	87.0			
2	67.3	100.0	11.9	60.3	100.0		
1	42.6	68.4		37.1	98.7		
0.5	28.8	44.7		25.6	18.0		
0.25	18.0	28.5		17.4			
Ratio	1:54	1:150	1:22	1:50	1:110	1:7	1:3

competent conglomerate rock material which had a bench height of approximately 15 metres.

The stemming heights were reasonably consistent within each particular set of experimental holes except in one case. During loading on 7-4-94 it was noticed that the burden on the Fine Rock hole had broken away to a depth of 3 metres from the crest so less explosives was loaded in the hole to compensate for the small burden at the crest of the hole.

The powder factors for each set of experimental holes are shown to be similar indicating that explosive loading would have a minimal contribution to any variation that was measured in the blast properties for each experiment.
Overall the blasthole configuration and loading details for both sets of experiments were shown to be reasonably consistent and any variation measured should result from changes in the stemming material properties being investigated.

5.3.3 Blast Properties.

The properties of both blasts as monitored by the high speed photography are shown in Tables 5.4, 5.5 and 5.6. These values were obtained by tracking the trajectory of the face markers in the time domain as they are thrown through space by the force of the explosive.

Table 5.3 Blast hole loading configuration.

Stemming	Normal	Fine Fines	Coarse Fines	Normal	Sand	Fine Rock	CoarseRock
Blast date	18-02-94			7-04-94			
Hole depth(m)	5.9	6.2	6.1	15.1	14.2	15.1	14.5
Explosive ht (m)	4.6	4.7	4.6	13.7	13.0	11.6	13.3
Water (m)	-	1.0	-	-	-	-	-
Primer (g)	200	1700	200	400	200	400	400
Stemming ht(m)	1.4	1.5	1.5	1.4	1.2	3.5	1.2
PowFact(kg/m^3)	0.53	0.52	0.52	0.65	0.66	0.55	0.66
Burden (m)	2.1	2.1	2.1	2.0	2.0	0.5	1.5

5.3.3.1 Front Row Burden Properties.

Typically these properties are to the ejection angle, the burden velocity and the rock initial movement time. Both the ejection angle and the burden velocity are important when the casting range is of concern. If we think of the equations of motion then the distance a shell can be fired out of a cannon, for example, depends on the shell velocity and the angle of the cannon to the horizon. This is analogous to cast blasting. The front row burden properties for both blasts are shown in Table 5.4.

The initial movement times are shown to be fairly consistent within each blast although a significant difference exists between the two different types of rock materials. These initial movement times are crucial in designing the initiation sequence for a blast. This initial movement time indicates the time at which each row separates from the subsequent row leaving a "free" face. The difference in initial movement times between the two types of rock materials would tend to indicate more work has to be carried out on the more competent rock in fracturing, cracking thus causing a longer initial movement time.

The ejection angle is the angle the rock moves away from the original position of the free face. The positive value, in most cases, indicates the classical ballooning shape of the front face in a slightly upward direction to help move the broken rock away from the blast area. As the blastholes were drilled at an angle of 15° it would be expected the ejection angles would be in a positive upward

Table 5.4 Front row burden properties.

Stemming Material	Initial Move (ms)	Ejection angle (degree)				Burden Velocity (m/sec)				Expl. VOD (m/sec)
		Top	2Top	2 Bot	Bottom	Top	2 Top	2 Bot	Bottom	
Blast date 18-02-94										
Normal	10	56.6	1.6	21.6	23.9	7.0	6.3	6.8	6.4	3140
Fine Fines	12	44.3	8.4	-22.2	-8.7	6.6	7.8	7.0	6.6	-
Coarse Fines	10	53.5	13.3	27.7	16.9	7.7	7.7	6.9	6.8	-
Blast date 7-04-94										
Normal	19	45.0	27.0	31.8	36.5	13.1	14.4	14.0	12.9	3260
Sand	21	21.7	29.4	20.7	25.9	13.8	15.7	14.9	11.1	3280
Fine Rock	21	32.0	15.6	25.3	5.8	13.7	16.7	17.0	17.0	3520
Coarse Rock	21	50.6	45.0	39.1	31.2	16.4	25.9	18.1	22.2	3460

direction. There would appear to be no significant difference in performance between all the Normal stemming and the Coarse Fines from the blast on the 18-2-94. But some difference is noted in the bottom section of the burden for the Fine Fines of this blast. No significant difference is noted for any of the stemming materials used in the blast dated 7-4-94.

The burden velocities show a trend of a different nature. For the blast on the 18-2-94 no significant difference in the front row burden velocities were noticed as the particle size of the stemming materials were very similar to each other and so similar effects would occur within the blasthole. However, when the stemming particle size was changed considerably, as in the blast on the 7-4-94 the burden velocities are similar in the Normal and Sand stemming material (similar particle sizes). But when particle sizes approaches 1/10 of the blasthole diameter better energy transfer appears to occur and the burden velocities are significantly higher. This is more exaggerated in the Coarse Rock stemming where the burden velocities are increased by nearly 50% over the Normal stemming material results.

5.3.3.2 Stemming Blast Properties.

The blast properties of the stemming material that are important is the length of time after initiation of the explosive column the stemming remains intact and velocity of the stemming ejection. These properties are shown in Table 5.5.

For the blast on the 18-2-94 the stemming remained intact for a long period of time after the initial movement of the ground occurred. This coupled with the low stemming ejection velocities measured would tend to indicate that the stemming and or stemming column height adequately retained the explosive gases long enough to transfer energy to the confining rock mass.

Table 5.5 Stemming Blast Properties.

Stemming Material	Stemming Movement (msec)	Stemming Velocity (m/sec)
Blast date 18-02-94		
Normal	34	30.5
Fine Fines	24	41.6
Coarse Fines	48	14.8
Blast date 7-04-94		
Normal	28	104.1
Sand	48	39.8
Fine Rock	0	-
Coarse Rock	0	-

However in the blast on the 7-4-94 both of the Rock stemming materials were not ejected from the blasthole at all and reasonably high stemming ejection velocities were measured for the Normal and Sand stemming materials. Thus increasing the stemming particle size in relation to the blasthole diameter can have a significant effect on the energy transfer from the explosive to the confining rock mass.

5.3.3.3 Casting Range.

Explosives used in blasting operations are required to throw the fragmented material to form a pile suitable for the excavation equipment used at the mine. The casting range is important and can determine the type of muckpile formed. The casting ranges measured are shown in Table 5.6.

Table 5.6 Front Row Casting Ranges.

Stemming Material	Casting Ranges			
	Top	2nd Top	2nd Bottom	Bottom
	(metres)			
Blast date 18-02-94				
Normal	5.7	4.6	5.2	1.5
Fine Fines	6.3	6.4	1.9	0.9
CoarseFines	6.9	6.7	5.4	1.4
Blast date 7-04-92				
Normal	22.8	25.6	19.4	13.0
Sand	26.0	29.0	15.2	12.7
Fine Rock	26.2	28.6	27.6	9.8
Coarse Rock	29.2	59.3	43.6	13.0

The casting ranges shown for the blast on the 18-2-94 are very low and would tend to indicate a muckpile that would "stand up" and not be spread out on the pit floor. This was the case for this blast as manouverability on the pit floor was very limited at the time of the blast.

For the blast on the 7-4-94 the casting ranges are significantly higher due mainly to the higher powder factor. However there appears to be an appreciable difference between the stemming materials used. There is no significant difference between the Normal and Sand stemming materials and only a slight difference for the Fine Rock stemming material. But in the case of the Coarse Rock material the casting ranges are much higher indicating a better energy transfer to the broken ground for this particular blasthole configuration.

5.4 FIELD STUDY CONCLUSION

The results of the laboratory study and the laboratory scale blasting trials were used as a guide to plan full scale trials in a quarry operation. A section of the front row of two different blasts was used as the test area and modern blast monitoring equipment was used to capture the blast event in a certain time frame.

Two different blasts were monitored, one in a highly weathered top layer approximately 5.5 metres deep and the other in more competent rock approximately 15 metres deep.

In the case of the 5.5 metre bench blast the results from this series showed no significant difference in the blast properties for any of the stemming materials but there was a small increase in ejection angle, burden velocity and hence casting range for the Coarse Fines over the other two stemming materials.

In the case of the 15 metre bench blast in more competent rock structure, the major improvement for this series of tests was the elimination of stemming ejection by using the Fine Rock and Coarse Rock stemming materials. As no energy was wasted in ejecting the stemming material a better transfer of energy from the explosive to the rock mass was indicated in the ejection angle and the burden velocities. Burden velocities increased by approximately 50% in the case of the Coarse Rock hole and a smaller increase in the Fine Rock hole over the Normal stemming material hole was measured. One of the most important parameters is the casting range and again both the rock stemming material holes showed an increase over the Normal stemming material hole.

Overall the quality of the stemming was shown to significantly affect the blast properties in the quarry blasts tested. This field study has shown an increase in the burden velocity hence casting range and possibly fragmentation as more energy is transferred to the confining rock mass instead of being wasted to the atmosphere by ejecting the stemming material.

CHAPTER 6

GENERAL CONCLUSIONS

The generally accepted blasting theory which is used to describe the effect of the explosive force acting on confining rock to produce the desired degree of fragmented material is well documented. The explosive generated force is extremely high and the period over which it acts on the rock mass is in the order of tens of milliseconds. During this short period of time the chemical energy stored in the explosive ingredients must be transferred to the confining rock mass and perform useful work in fragmentation and movement.

The transfer of the energy released from an explosive was studied both in a laboratory and in a hard rock quarry. The effect of this energy transfer to the confining medium, ie. the quality of the stemming material, was the main consideration of this work.

Stemming materials of various types (sand, limestone, iron ore and granite) were examined and tested to compare their physical performance in blasthole simulated experimental rigs under controlled laboratory conditions. The effect of material type, used as stemming, was found to have no significant effect on the resistance to movement of compacted samples as a spike was forced into the sample. The introduction of this spike simulated the movement of the grains of stemming material as the explosive force is applied to the base of the stemming column. The resistance to movement was monitored by the decrease in velocity of the spike as it penetrated the stemming material, and reasonably constant rates were measured for all 4 materials tested. At particle sizes less than 4mm these materials were considered to be reasonably competent and resist breakage during movement.

The explosive force in the form of pressurised gases applied to the base of a column of stemming material acts on the confining material for a very short period. A laboratory rig which simulated the effect of pressurised gases on a column of stemming material showed how effective different sizes of stemming materials can be. This pressure at the base of the column was shown to decrease as the stemming particle size increased before stemming ejection occurred. It appears that at the larger particle size the interstices between the grains are large enough, although the overall void ratio does not vary significantly. This condition allows the pressure to disperse more rapidly through the stemming material. This "pressure relief" minimises the pressure at the stemming column as the confining medium is expanding and the stemming column to borehole contact is decreasing thus maintaining the stemming intact for a longer period.

A series of laboratory scale trials were then planned to investigate the effect of the energy transfer to the confining rock mass. The effect of the explosive gases as the "prime mover" of the fragmented material was tested by enclosing the explosive charge in a steel tube cast into a concrete block. In this series of tests a crack pattern around the explosive charge with several large cracks radiating to the surface was observed, and this is similar to that reported by other researchers. Thus the shock part of the energy formed the crack network ready for the gas pressurisation action.

The effect of the pressurised gases on fragmentation and movement was measured in a series of experiments where the gases were allowed to escape to atmosphere through various sized orifices above the blasthole collar. It was shown that the quality of fragmentation, K_{50}, would decrease by 50%. from a no stemming case to a fully confined situation and a corresponding increase in fragment movement or burden velocity of 25% was observed. So the "gas pressure is the prime mover" statement in the literature was found to be correct and the amount of confinement or energy transfer to the confining medium plays an important part in rock fragmentation and burden movement.

The effect of different stemming types on the blast properties was then investigated. Stemming types used ranged from sand to various sized limestone to complete confinement by cementing the charge in the blasthole. The effect of some stemming, irrespective of its type, appears to form a resistance barrier long enough for energy to be transferred to the confining medium and do useful work. This was shown by a sharp change in the blast performance from no stemming to very weak confining stemming followed by a gradual change as the confinement quality increased. Stemming materials of different particle size showed a significant decrease in stemming ejection velocity as the stemming particle size increased. Again indicating a better transfer of the energy to the confining rock and less wasted energy in ejecting the stemming from the blasthole.

The true test of any laboratory study is in the field where some of these findings were tested in actual production blasting conditions. A series of tests were carried out in a hard rock quarry where the explosive charging in the front row of holes were modified and compared to existing practice. The energy transfer was measured by monitoring the front row burden velocity. When the stemming particle size approached this 1:10 ratio (particle size to blasthole diameter) significantly higher burden velocities were obtained. A burden velocity increase of 25% was measured when the stemming particle size to blasthole diameter ratio was 1:3 for a blasthole diameter of 76mm.

Another important aspect of increasing the stemming particle size is the time the stemming material remains in the blasthole after the explosive has been initiated. When the stemming particle size to blasthole diameter ratio was greater than 1:10 no stemming initial movement was detected and no stemming ejection occurred from the blasthole during that critical period. In the case of a much smaller ratio, approaching 1:100, as early stemming movement was detected and stemming ejection velocities up to 100 metres/sec were measured.

The quality of stemming used in blastholes has long been ignored and an "any stemming will do" attitude has prevailed in the mining industry. When environmental factors are important the use of crushed rock stemming material compared to drill cuttings has been shown to reduce air blast and also flyrock. However, the benefits of the crushed rock stemming on the blast performance has been over looked as the environmental factors prevail. The industry belief that "crushed rocks form a bridge" hence act as a better form of stemming has been quantified with fragmentation quality, K_{50}, reductions in the order of 50%, in laboratory tests, and burden velocity increases in the order of 25%, in a quarry operation. The quality of stemming in assessing blasting efficiency has been shown to play a significant role and one which should not be dismissed so easily. Confining the explosive gases, which act as an energy accumulator, in the blasthole for as long as possible can produce benefits in the blast performance such as fragmentation and burden velocity, and also play an important role in the reduction of flyrock and air blast.

CHAPTER 7

REFERENCES

Anon., "Stemming.", 1987, AECI Tech. Bull., Series 2, No.46, June.

Anon., "Water as a Stemming Material - Its Use in Western European Countries.", 1970, Colliery Guardian, Vol. 218, No.5, pp247-253.

Armstrong L.W., Moxon N.T. and Sen G.C., "The effects of confinement on fragmentation and movement.", 1993, 4th Int. Symp. on Rock Fragmentation by Blasting, Vienna, Austria, 5-8 July, pp353-360

Baranov E.G. and Kovalenko V.A., "Effect of the Detonation Parameters of Explosives on the Explosion Energy Distribution of a Borehole Charge.", 1975, Soviet Mining Science, Vol 11, No6, Nov-Dec, p 665-669.

BHP Research Newcastle Laboratory, 1989, Internal Reports.

Brinkmann J.R., "An Experimental Study of the Effects of Shock and Gas Penetration in Blasting.", 1990, 3rd International Symposium on Rock Fragmentation by Blasting, Brisbane, Australia 26-31 August, pp 55-66.

Britton R.R., Gozon J.S. and Skidmore D.R., "Explosive energy-rock fragmentation correlations for two experiments.", 1989, International Journal of Surface Mining 3, p 163-166.

Cameron A.R. and Torrance A.C., "The Underwater Evaluation of the Performance of Bulk Commercial Explosives.", 1990, 6th Annual Symposium of Explosives and Blasting Research, Orlando, Florida, USA, pp 27-42.

Chapman D.L., 1899, Phil. Mag., 47, 90

Chiappetta F.R., "Blasting Theory.", 1989, First High-Tech Seminar on State-of-the-Art Blasting Technology Instrumentation and Applications, Orlando, Florida, USA, May 15-19.

Cole R.H., "Underwater Explosions.", 1948, Dover Publications, New York, New York.

Cook M.A., "The Science of High Explosives.", 1971, ACS Monograph Series, R.E. Krieger Publishing Co. Inc., Huntington, New York.

Coulson J.M., Richardson J.F., Bockhurst J.R. and Harber J.H., "Chemical Engineering. Volume 2 : Unit Operations.", 1985, 3rd ed., Pergamon Press, New York.

Curran D.R., Seaman L. and Shockey D.A., "Dynamic Failure of Solids.", 1987, Physics Reports 147, No 5 & 6, pp 253-388.

Field J.E. and Ladegaard-Pedersen A., "The Importance of the Reflected Stress Wave in Rock Blasting.", 1971, Int. J. Rock Mech. Min. Sci., 8, pp 213-226.

Fourney W.L., Dick R.D. and Simha K.R.Y., "Model Study of Crated Blasting.", 1988, Rock Mechanics and Rock Engineering, V21, n3, pp183-205.

Guilfoyle K., 1991, "Safe and Efficient Blasting in Surface Coal Mines", Training Manual, ICI Australia Operations

Gurney R.W., "The initial velocities of fragments from bombs, shells and grenades.", 1943, BRL Report 405.

Hagan T.N. and Duval M.B., "The Importance of some Performance Properties of Bulk Explosives in Rock Blasting.", 1993, Rock Fragmentation by Blasting, Rossmanith ed., Balkema, Rotterdam, pp 387-393.

Hagan T.N. and Just G.D., "Rock Breakage by Explosives - Theory, Practice and Optimisation.", 1974, Int. Sac. for Rock Mechanics, 3rd Congr., Denver, Col., Part B, pp 1349-1358.

Hardesty D.R. and Kennedy J.E., "Thermochemical Estimation of Explosive Energy Output.", 1977, Combustion and Flame, Vol. 28, No1, p 45-59.

ICI Explosives, "Safe and Efficient Blasting Course", 1993

Johansson C.H. and Persson P.A., "Detonics of High Explosives.", 1970, Academic Press, London, England.

Jouguet E., 1906, "Micanique des explosifs.", J. Math., 1905, 347, 6, Paris

Just G.D. and Lamont G., 1979, "Stemming of Blastholes in Mining Excavations.", Proc. Aust. I.M.M., No.269, March, pp 7-15.

Kolsky H., "Stress Waves in Solids.", 1953, Oxford University Press, London, England.

Konya C.J. and Davis J., "The Effects of Stemming Consist on Retention in Blastholes.", 1978, Proc. Explos and Blasting Tech., New Orleans, La., Feb 1-3, pp102-112.

Kopp J.W., "Stemming Ejection and Burden Movements from Small Borehole Blasts.", 1987, US Bur. Mines 9080, 18p.

Kristiansen J., Kure K., Vestre J. and Bergqvist I., "An Investigation of Heave and Fragmentation Related to Explosive Properties.", 1990, 3rd International Symposium on Rock Fragmentation by Blasting, Brisbane, Australia 26-31 August, pp 83-90.

Mohanty B., "Energy, Strength and Performance, and Their Implication in Rating Commercial Explosives.", 1981, Proceedings of the 7th Conference on Explosives and Blasting Technique, Phoenix, Ariz., USA, 19-23 Jan, p 293-306.

Nilsson F., "Crack growth initiation and propagation under dynamic loading.", 1984, 3rd Conf. Mech. Prop. High Rates of Strain, Inst. Phys. Conf. Ser. No.70, pp 185-204.

Otuonye F.O., Konya C.J. and Skidmore D.R., "Effects of Stemming Size Distribution on Explosive Charge Confinement : A Laboratory Study.", 1983, J. Mining Engineering, August, pp 1205-1208.

Postupack C., 1989, "Application and Economics of Overburden Casting - Can it Work for You?", First High-Tech Seminar on State-of-the-Art Blasting Technology Instrumentation and Applications, Orlando, Florida, USA, May 15-19.

Preece D.S. and Taylor L.M., "Spherical Element Bulking Mechanism for Modelling Blast Induced Rock Motion.", 1990, 3rd Int. Symp. Rock Fragmentation by Blasting, Brisbane, Aust., 26-31 Aug., pp 189-194.

Snelling W.O. and Hall C., "The Effect of Stemming on the Efficiency of Explosives.", 1912, US Bureau of Mines, Technical Paper 17.

Taylor J., "The Resistance of Stemming : Dynamical Investigations.", 1945, Trans. I.M.E., Vol. 105, pp 51-74.

Taylor J., "The Resistance of Stemming : Statical Investigation.", 1938, Trans. I.M.E., Vol. 56, pp 153-165.

Udy L.L and Lownds C.M., "The Partition of Energy in Blasting with Non-Ideal Explosives.", 1990, 3rd International Symposium on Rock Fragmentation by Blasting, Brisbane, Australia 26-31 August, pp 37-43.

Worsley P. and Nixon F.T., in Zaburunov S.A., 1990, "Blasting Mind Over Matter.", E&MJ, April, pp 20-25.

Zaburunov S.A., 1990, "Blasting Mind Over Matter.", E&MJ, April, pp 20-25.

Zakharova I.G., "Energy Losses during an Explosion.", 1984, Combustion, Explosion and Shock Waves, Vol 20, No3, May-June, p 343-346.