

AN INVESTIGATION INTO COAL DAMAGE DURING BLASTING

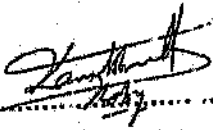
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A thesis submitted to the Faculty of Engineering, University of the Witwatersrand, Johannesburg, in fulfilment of the requirements for the degree of Doctor of Philosophy.

Johannesburg, 1997

DECLARATION

I declare that this is my own, unaided work. It is being submitted for the degree of Doctor of Philosophy in the University of the Witwatersrand, Johannesburg. It has not been submitted before for any degree or examination at any other University.

Signed 

Dated this 20th day of November 1997

ABSTRACT

This thesis has investigated ways of exerting control over coal breaking during blasting. Its ultimate objective has been to optimise the use of explosive energy in blasting so as to ensure reduced production of coal fines in the comminution process.

The investigations reported were conducted on laboratory models at a scale of a controllable energy to enable the development of a theoretical background for the thesis. However, the bulk of the work was undertaken insitu in surface and underground coal operations. The research was conducted in three phases. The first phase dealt with the approach to modelling the primary crushing mechanism which generates the coal fines during the blasting operation. A laboratory apparatus to simulate blast-generated dynamic loading was devised. Dynamic coal crushing test conducted on a set of over 150 samples of coal showed an exponential relationship between the comminution energy and the subsequent coal damage. A comminution index (Comi) characterising the ability of coal to generate fines was derived.

The second phase of the research concerned the field investigations. This had the following double aim: validating the hypothesis of the influence of the dynamic loading of the shock waves and that of the inherent discontinuities imbedded in coal on fines generation; and quantifying the problem of fines in the blasting techniques currently used in the local mining industry. Three sites were used, namely Kleinkopje of AMCOAL, Greenside of Goldfields and ATC, Tavistock of JCI.

The investigations undertaken at Kleinkopje (surface mining operation) involved the determination of discontinuities by logging inherent fractures appearing on 75 diamond drilled cores of coal and the monitoring of 76 blasts conducted on the sites sampled for discontinuities. These investigations indicated a good correlation between the fines observed in blasted piles and the fracture frequency of the pre-existing fractures per metre of coal.

The work undertaken in underground collieries (Greenside and Tavistock) demonstrated the enormity of the fines' problem in the local mining operations. Approximately a third of the coal broken underground was observed, in 30 blasts sampled and analysed, to fall into the category of fragments of under 6 mm termed 'coal fines', which generate low income.

The third phase of the research dealt with the optimisation of coal breaking during blasting. A series of 100 trial blasts was undertaken with the aim of searching for a blasting technique that enables the generation of fewer coal fines. A number of options were investigated. Trial blasts carried out were evaluated and their results are discussed in this thesis.

The results of the fieldwork undertaken confirmed that fines generation is site specific. However, the decoupling technique used with high VOD explosive products may be successfully employed to reduced coal fines generation in strong coals (few inherent fractures).

DEDICATION

*TO my beloved family
for their love*

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1 INTRODUCTION

1.1 The problem

1.1.1 Position

Coal is a sedimentary rock material arising from the differential compaction of a series of deposited organic and inorganic sediments to form layers (coal seams) of different strength. As a result of the complex conditions of its formation, it generally presents divergent strength properties, which make it difficult to predict its behaviour under load. South African coal is generally hard, free of cleats and has a relatively low methane content compared to European coals (Macgregor, 1983).

The development of mechanised non-explosive breaking technology in recent years has profoundly influenced the local coal mining industry. However, a significant number of operations still use explosives for the primary breaking of coal in underground collieries, while at present, the entire South African surface coal mining industry relies on drilling and blasting. Although the shift to mechanisation is evident in underground mining, explosives will still be used in the industry for a while.

Theoretically, if the breaking processes that occur during blasting were properly understood, the blasts could be designed such as to produce only the most wanted results. However, blasting involves a huge amount of explosive energy imparted to coal in a very short time. It yields products sizes and proportions that defy the blasters' control. Attempting to understand the process has been the subject of research over many years but to date no control has yet been gained over the mechanisms involved in blasting. The tremendous energy released during a blast is still not being effectively utilised in breaking coal to meet the quality expectations.

Frequently coal piles, after blasts, surprise the blasters either by the amount of over- and/or undersize or by the small movement (throw) from the face. More research is still required in this field. The present thesis intends to make a modest contribution by investigating the blasting techniques that may result in less damage to mined coal (i.e. in the generation of fewer coal fines).

1.1.2 Production and marketing of coal

Mining economics require that any industrial operation should meet the market demand. The technology employed and the subsequent quality of the final product ought to satisfy the requirements of the business. In coal mining, most of the products' intrinsic quality characteristics are imposed by nature (ash content, specific gravity, etc.). However, the important value-adding technical quality for marketing is the size distribution of the mined products. Coal produced for export (28% of the total production in 1995)¹ is subject to different size requirements than that produced for the local consumption. The difference encompasses the in-mine breaking technology used, which basically determines the sizing obtained. The in-mine fragmentation achieved is therefore a critical parameter influencing the overall mining cash flow.

Fragmentation is understood as a range of coal sizes delineated by an upper (oversize) and lower (undersize) limit known as called cut-off sizes. These threshold sizes need to be strictly adhered to as they are usually set to correspond to the pay limit outside of which the coal product is not saleable at cost. This concept is fundamental for mine management and the present thesis uses it as a driving idea in the investigation of ways and means of reducing coal damage caused by excessive comminution during

¹ Chamber of Mines Statistical Tables, 1995

blasting operations.

Drilling and blasting in coal mining is perceived to be associated with high levels of undersize and/or oversize despite having the advantage of low cost. Oversize do not pose a serious threat to productivity as they can be broken into smaller fractions during downstream operations after blasting. The undersize are the major concern of any coal mining operation. In addition to the loss of earnings, they constitute a potential danger for the personnel (explosions and airborne dust).

1.1.3 Coal fines and why they need to be reduced

In the last three years, South African coal production has been boosted by the termination of the country's isolation from the world and the rising market price. Increased demand for the product has triggered a trend towards excavating greater volumes of coal. However, the high mining rate could lead to rapid depletion of reserves. The production costs in the local coal industry are low (Chadwick, 1994), but the mining efficiency is equally low. The extraction ratio of coal is low partly because a valuable proportion of reserves is left behind in the mines in the form of useless fines and dust. Breaking coal to fines denotes a lack of control over the breaking process and constitutes hence a poor management of the resources. Even though some fines can be beneficiated, they are always subject to market penalties that subsequently impact negatively on the revenue. The primary breaking process used in the mine (drilling and blasting) is usually the major generator of coal fines. It is therefore important that more research be undertaken so as to acquire a better insight into the process and to gain control over the crushing mechanisms responsible for fines.

1.1.4 Actual loss of income through fines generation

Mining standards in the local coal industry set the lower cut-off size for mined coal at 6 mm (Thorpe and Pedroncelli, 1986). Coal sized between 25 mm and 6 mm is highly marketable. The < 6 mm fraction is termed duff, undersize or simply fines. Records indicate that on average, 30 % of coal broken underground falls into the category of fines. Approximately 25 % fines are produced in surface mining operations. Such poor performance is a cause for concern since coal reserves are not renewable. No good reason can be given to justify such a waste of opportunities for profit. A survey done by Delaporte and Heizmann (1995) at two AMCOAL collieries illustrated better the impact of coal fines on the mine's cash flow. A reduction of 1 percentage point in in-pit coal fines was seen to improve the mine revenues by R 2 million at Kleinkopje and R 1,5 million at Landau per annum.

1.2 Scope of the thesis

Problems of explosive-induced fractures still have two basic aspects requiring a lot more research. Firstly, there is the performance of the explosive charge in a confined chamber (blasthole) and the relevant conditions favourable to effective transfer of energy from the explosive to the medium. Secondly, there is the rock engineering consideration of the control of the initiation and extension of fractures in the confining medium and the subsequent damage induced; this depends on the heterogeneous nature of the medium and the interaction between the explosive and the rock, for instance, coal.

Coal broken up into numerous small particles and abundant fine particles is understood to be the result of multiple fracture initiation causing the

crushing. This always arises from an uncontrolled complex state of stress distribution generated by the combined effects of the interaction between the explosive reaction and the coal mass. The problem has been investigated in a number of trial blasts reported in this thesis. The research aimed at investigating ways of controlling crushing so as to reduce coal losses through fines. The bottom line was to seek a basic understanding of the breaking mechanism and the occurrence of damage, i.e. to quantify the relationship between the blasting parameters (energy input) and fines generation and then to look for blasting techniques which would reduce fines in coal blasting.

1.2.1 Coal damage caused by blasting

In coal mining, multiple forms of damage can be induced in coal by blasting work (i.e. damage to mined coal by comminution, damage to the coal remaining in place after excavation, etc.). This thesis is concerned solely with damage to coal in the form of fines generated by blasting. In general, two types of blasting operations induce this damage: blasting work for overburden (in surface mining) and that for coal production (both in surface and in underground mining).

1.2.2.1 Blasting for overburden removal

Blasting work aimed at removing the overlying ground above coal is undertaken in surface coal mining. These operations, as well as the heavy-duty machines used to handle blasted rocks above the coal seam (shovels, dozers, draglines and trucks), induce damage of various extents in coal. In essence, most damage is induced by the explosive energy that is released directly into the coal, when the explosive charge in the hole drilled in the overburden comes into contact with coal in the seam. Due to

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the concentration of the charge at the bottom of the blasthole, huge amounts of energy are imparted directly to coal. The low compressive strength of coal causes it to sustain profound in-situ damage, ranging from macro to micro fractures. Investigations of the simulated dynamic shock loading of coal under controlled conditions were attempted as part of this thesis. The response of coal was evaluated and the extent of the resulting fracturing was assessed. The small-scale laboratory work involved tests in perspex models. The results are analysed and interpreted in this thesis and the relationship between the in-hole explosive coupling (given partition of shock and gas energy phases in the blast) and the damage observed is discussed.

1.2.1.2 Production blasting in coal

Mining of hard coal requires the use of explosives placed in drilled holes within the coal mass. Blastholes are drilled with relatively small diameters to enable the use of reasonable powder factors. The higher the energy the bigger the resulting zone of comminution. Sustained damage in coal is dependent on the radial distance to the axis of the blasthole during detonation and on the inherent discontinuities within the coal burden. It is worth noting that coal is a versatile material with a number of innate weaknesses such as bedding planes, cleats and cross-cleats, and other types of flaws. Its response to the loading stresses of blasting is generally complex and difficult to predict. To the author's knowledge, research in this field has not yet yielded a successful model for predicting the behaviour of coal under the dynamic loading of blasting stresses (shock and gas). This thesis intends establishing an empirical relationship, if possible, between the coal fines generated, the inherent fractures and the characteristics of the explosive energy input. The thesis acknowledges that the in-hole loading geometry of the charge has a great influence on the overall

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performance of the explosive. The partition between the effects of the shock and gas energy phases of the detonation determines the initiation and extension of the blast-induced fractures. The investigation therefore revolves around this parameter. The research methodology adopted distinguishes the effects of the two energy phases in terms of their contribution to fines generation in blasting. It is assumed that the shock energy is solely responsible for quantifiable coal crushing.

Given the above background, the objectives of the thesis are outlined as follows:

- to assess the levels of in-mine coal fines generated through the use of the current blasting technology in the coal industry,
- to seek an insight into comminution in blasting,
- to highlight the need for proper evaluation of the in-situ strength properties of coal, including the prevailing inherent fractures which are the key to the partition of the explosive energy (shock and gas) responsible for blast results, and
- to arrive at an understanding of the blasting technology needed to minimise in-mine coal fines generation in order to improve the cash flow of mining operations.

1.2.2 Overview of the thesis

The objectives listed above are dealt with in the nine chapters of this present thesis, which are summarised here. The first chapter describes the overall purpose of the thesis. It outlines the main objectives of the research done which revolves around the search for ways and means of engineering a technology for blasting coal to produce fewer fines. The chapter covers market related considerations to pinpoint the importance of sound drilling and blasting practices for optimum coal mining. Efficient

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operations are needed to satisfy the increasing demands of the market and therefore ensure maximum profit. It ends by outlining the scope of the work and summarising the approach taken.

The second chapter reviews previous research on rock and coal breaking. The literature consulted pinpoints the site-specific nature of fines generated in blasting. The need to reconsider the definition of a site's strength parameters is clearly highlighted. The thesis favours the dynamic coal and explosive reaction parameters. It is found that no reliable prediction of coal breaking is possible as long as control over the process cannot be guaranteed. More research is justified. The investigations reported in this thesis are intended to contribute to the scientific effort in this field.

Chapter 3 presents the basic hypothesis of the model studied in this thesis. It highlights, on one hand, the results of careful observations of coal fines in blasted piles in an effort to locate their original position in coal burdens prior to blasting. On the other hand, it examines the contribution of the individual mechanisms responsible for coal fines generation (primary and secondary crushing). It identifies the driving mechanism that determines the producible fines and upon which the global model is based. It is shown that it is the intensity of the shock wave pulse impacting on the blasthole walls which dictates the extent of the crushing and the configuration of fractures around the blasthole. The model presented is based on a simulation of the onset and propagation of shock pulses observed in laboratory. The description of the theoretical background on the behaviour of coal under dynamic loading is completed in chapter 4 that deals with the criteria of optimisation.

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Chapter 4 describes the comminution model envisaged. It presents the background of the test methodology adopted in order to assess the dynamic response of coal to blast loading stresses in the laboratory. It sets out the concept of the 'appearance function' which is used to define the comminution index (whose units are MJ/kg/% fines, meaning powder factor per percent of fines generated) which is characteristic of the ability of coal to resist dynamic crushing. The concept is illustrated in the laboratory and in-situ investigations presented in the subsequent chapter.

Chapter 5 presents the apparatus designed for laboratory investigations. It also describes the techniques used in the small-scale-simulated blasting work undertaken. The dynamic coal crushing test conducted on a set of more than 150 samples prepared from coal blocks collected from different collieries resulted in an empirical exponential relationship being found to relate the comminution energy and coal fines generation. This relationship may be used as guideline in predicting the comminution behaviour of coal.

Chapter 6 concerns the field investigation of fines generation conducted in surface mining operations. This investigation was aimed at validating the fundamental hypothesis of the thesis described in Chapter 3. The chapter 6 reports on the geotechnical work undertaken which involved 75 diamond drilled cores considered as a representative sample in the study of the in-situ discontinuities in coal of the three operating pits at Kleinkopje Colliery. It also presents the technical parameters of blasting work done involving 76 blasts conducted in the areas sampled for inherent fractures. Coal fragmentation obtained after the blasts is discussed and compared in terms of the fines generated. A correlation between the fines generated and the inherent fracture frequency per meter observed in different coal horizon is attempted. A good linear relationship is found to exist between

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the pre-existing fractures and the fines in blasted coal piles. The blasting work in surface operations was followed by an intensive investigation of the generation of fines in underground collieries, which is presented in the next chapter.

Chapter 7 deals with the investigation in underground operations. Trial blasts were conducted at two pilot collieries of the Witbank coalfield. The aim of this project was firstly to determine the extent of the fines generation problem in underground mines, and secondly to investigate the ways of possibly developing a remedial blasting technology, a blasting technique which may lead to the generation of fewer coal fines. This chapter presents the blast designs used in the trials and the discussion of the fragmentation results obtained. The underlying idea of the technique that is suggested in the present thesis is to achieve some control over the process of coal breaking by altering the in-hole geometry of the explosive charge. By reducing the explosive charge outer confinement the blasthole one may increase considerably the comminution index ($ComI$), defined as the powder factor per percentage of fines generated, therefore improve the fragmentation. The idea was basically proved through successful results of trial blasts that are presented and discussed in this chapter. Their interpretations and theoretical background are developed in next chapter.

Chapter 8 is aimed at placing into context the discussion of the results obtained from the investigations presented in chapters 6 and 7. It focuses on the optimization and control of the blast-induced fractures in industrial blasting operations. It is also intended at outlining the contribution of the present research and pinpointing the subsequent recommendable practical measures for optimizing coal blasting.

Finally, Chapter 9 finally presents the conclusions of the thesis and indicates the areas of future work in this field.

2 BACKGROUND

2.1 Introduction

A literature review intended to give a background understanding of the theory of comminution in blasting evolved over the years is presented. It focuses on the results of previous efforts to control fines generation for optimum mining.

In the South African coal mining industry, great progress has been made towards the mechanisation of coal loading and transportation in collieries to improve productivity. However, very limited work has been done to gain insight into the breaking process in coal blasting. Blast results are frequently a surprise for the blasters. No effective control can be exerted over them so long as the in-situ strength and breakability of coal remain a mystery to blast designers and practitioners. Some years ago, COMRO attempted an evaluation of the various mining methods used for coal getting in terms of the size of the respective coal products generated (Thorpe and Perdroncelli, 1986). However, only mechanised coal breaking methods were considered and limited efforts were made to study the fundamentals of the breaking processes. Other attempts to investigate the optimisation of coal breaking in blasting were made by individual mines such as Kriel, Bank, Kleinkopje, Goedeheop, to mention only a few. Once again, not only are the reports confidential, but also the scope of the work was narrow, being mainly intended to solve a particular operational problem. In any case, coal sizing was not considered as the criterion for optimisation.

Blasting is the cheapest industrial means of rock breaking. It produces a range of product sizes (under-, mid- and oversize) whose proportions

depend on the blast design and its implementation on site. In coal mining for instance as said early, oversize do not pose a problem as they can be reduced, at reasonable cost, to the most wanted size range, namely the mid-sizes. The undersize are a problem in any operation. They are commonly classified as duff or fines (< 6 mm) and coal grits (< 500 microns). They have a negative influence on the mining cash flow since in some cases they may not be recoverable representing loss of resources or they may be subjected to penalties on the market. Airborne coal dust (< 80 microns) is a health hazard for mine workers, in addition to the permanent danger it carries of underground explosion (Landman, 1993).

2.2 Damage in blasting

Blasting uses an explosive charge that is set to produce an instantaneous chemical reaction (detonation) in a purposely-drilled hole. After the detonation, a pressure wave front of approximately 1 000 MPa and a subsequent high temperature up to 3 000 degrees are immediately generated. These conditions occur within a time frame of 1 to 2 microseconds (Whittaker, Singh and Sun, 1992). The pressure front created (shock waves) is characterised by its pulse that impacts on the coal of the blasthole walls, subjecting it to severe dynamic loading stresses. Given the low compressive strength of coal, the impact initiates multiple fractures in the coal burden that cause the breakage observed after the blast. The dynamic effects of the shock stresses (impact, propagation, reflection and interaction with coal) produces a fracture pattern in the coal burden (fragmentation), ranging from over-crushing to splitting and spalling, etc. Whittaker et al. highlighted that the static considerations used in previous research into the mechanics of rock fragmentation (Allsman, 1960; Kumao, 1961; Langefors, 1963; etc.) are inappropriate. One would rather use the

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dynamic fracture properties that are more suitable for study of rock breaking mechanisms since the onset, growth and fragment formation from the blasting induced cracks occur in a dynamic process (Whittaker, Singh and Sun, 1992).

The mechanics of fragmentation in blasting involve two basic parameters, namely the energy involved in producing fractures and the resistance to fracture of the material known as its toughness (Outcherlony and Zongqi, 1983 and Isakov and Sher, 1983). Fractures and the subsequent damage induced are initiated by the energy of the shock waves and completed by the energy of the gaseous products of detonation.

The speed of propagation of the reaction in the detonating explosive, also referred to as the rate of the thermo-chemical explosive reaction (Du Plessis, Wallace and Tipping, 1987), is proportional to the in-hole rate of liberation of energy to the material during the blast. It determines the partition of the available explosive energy between the two phases of shock and heave. It thus dictates the parameters of the interaction between the explosive and the rock (coal for instance), during the blast. It therefore controls the blast results (the size distribution of the resulting material fragments).

The rate of reaction is directly related to the in-hole velocity of detonation (VOD). The VOD is hence a measure of the rate of liberation of the detonation energy. It is a window into the partition of the explosive performance during the blast (Lownds, 1986). Depending on conditions of the detonation reaction (ideal or non-ideal), the energy within or behind the reaction may be a determinant coal breaking and fragmentation setting component in the blast. However, the VOD accounts for the time elapsed

to reach the peak shock stress corresponding to the maximum dynamic pressure loading in the medium.

Kutter and Fairhurst (1971) ascertained that the more the detonation products are allowed to expand before interacting with the rock (coal for instance), the lower the peak stress transmitted to the medium. If an air gap is allowed between the charge and the confining blasthole walls, the shock waves have less crushing effects as they are attenuated before impact. Consequently, the violence of the stresses imparted to the medium will be reduced, thereby decreasing the amount of fines generation. Blasting techniques based on this concept are commonly used in mining as well as in civil engineering work to prevent damage to the remaining excavation. They are known as controlled blasting techniques (pre- and post splitting, buffer, cushion and smooth blasting techniques).

The in-hole VOD influences the magnitude of the peak pressure impacting on the coal of the blasthole walls and thus determines the extent of the crushing zone. The profile of the radial stresses developed in coal (see Figure 1) may be represented by a single instantaneous pulse similarly to the trace of the recorded strain measured by Brinkman (1989) shown in Figure 2.2. In an earlier work, the author (Kabongo, 1985) applied the law of comminution to show that the ratio of the amplitude of the peak stress of the blast pulse to the characteristic strength of the medium is directly proportional to the fragmentation produced. The theoretical work referred to above is corroborated by field investigation undertaken by Brinkman (1989). He observed an exponential trend profile of the loading stresses in the burden of a blasted quartzite rock decreasing with the increasing radial distance (in time). In addition, he reported a trend of finer fragmentation found to correspond with the high VOD shots (high peak stress) and a

decreasing fragment size with the increasing VOD and shock energy.

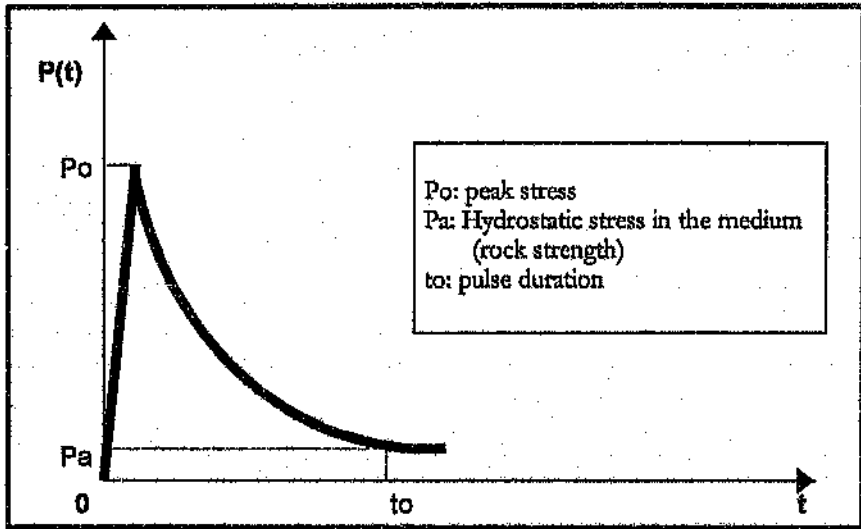


Figure 2.1 Schematic representation of a detonation pulse
(After Kabongo, 1985)

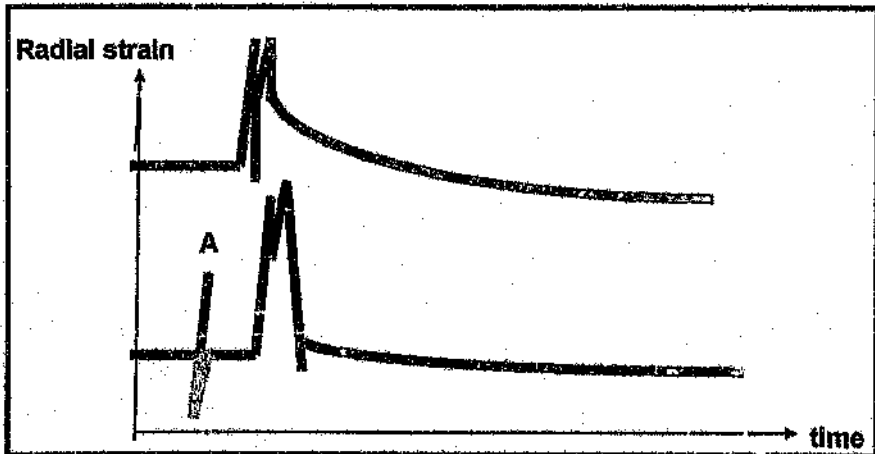


Figure 2.2 Typical radial strain trace recorded by Brinkmann (1989)
Noted that A is an electrical noise.

2.2.1 Fundamental concepts

The present concepts used to describe the breaking mechanisms in blasting are based on a set of assumptions made to simplify the analysis of events. Whittaker, Singh and Sun (1992) alluded to the rock breaking processes being reduced to a plane phenomenon occurring solely in the direction normal to the axis of the blasthole. Fragmentation is thus considered to be two-dimensional despite the fact that, in reality, it is a complex three-dimensional process involving a multi-parametric interaction between the explosion characteristics, the anisotropy and the free surface effects (Isakov and Sher, 1983). Furthermore, it is assumed that the diameter of the blasthole is always smaller than its length and that the hole is far away from free boundaries. In addition, it is also assumed that detonation occurs simultaneously along the entire length of the blasthole and that the change in effects along the blasthole axis is negligible. Given such a hypothesis, the understanding of the mechanism of the onset, propagation and dissipation of fractures and/or damage in blasted material gained over the years of research can be summarised as shown below.

2.1.1.1 Dynamic of shock waves

Upon detonation, within microseconds, the hole is filled with gaseous detonation products at high pressure and temperature. The instantaneous pressure momentum generated is imparted to the medium, exerting on the hole walls radial compressive stresses beyond their resistance. The medium, for instance, coal is superdense and over-crushed in the zone immediately surrounding the blasthole. In hard rock, the crushed zone (see Figure 2.3) is estimated to extend from the borehole to an area five to seven times the hole radius (Kutter and Fairhurst, 1971). The extent of the comminution zone created around the blasthole is still a subject of

controversy. A new school of thought opposes Kutter's point of view that tends to minimise the extent of the zone. It is now believed that the extent is rather large, up to 20 times the blasthole radius or more, depending on the blast conditions. Nevertheless, Whittaker, Singh and Sun (1992) report that the fracturing in this zone is due to granular cracking, micro-cracking, differential compression of the particles, the matrix of the rock and other forms of plastic deformation. Da Gama and Nelson (1971) showed that normal detonation of chemical explosives generates an extremely high rate of loading stresses that usually induce conditions of brittle behaviour rather than plastic flow. However, this zone of crushed rock is of no use in practice. The over-breaking done here, in fact represents a loss of the explosive energy and is generally associated with the unnecessary damage to excavations which compromises safety in the working environment, e.g. fall of ground from the hanging wall in underground mines or from the highwall in opencast mines. Any optimum blasting technology will therefore seek to minimise such occurrences.

2.2.1.2 Quasi-static gaseous products expansion

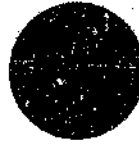
No direct relationship has been established between the gas energy phase of the detonation and the generation of fine particles in blasting. The energy of gas comes into play sometime after the dynamic action of comminution around the blasthole has been completed. When the crushed zone is in place, the radial and tangential cracks are initiated in the coal burden. The gaseous products of explosion fill the cavities of fractures opened up by the previous dynamic phase. It is a relatively long-lasting action (flat slope) compared with the shock effects (steep slope) as the rate of gas expansion along the isentrope is quite slow (see Figure 2.4).

Dynamic phase

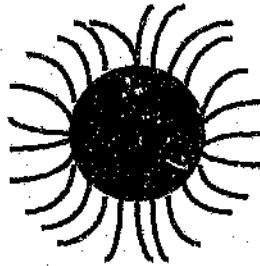
(a) Crushing



(b) Crushing and fracture in non-linear zone



(c) Radial cracks from elastic wave



Quasi-static phase

(d) Expansion of cavity. Extension of non-linear crushed zone. Growth of radial fractures.

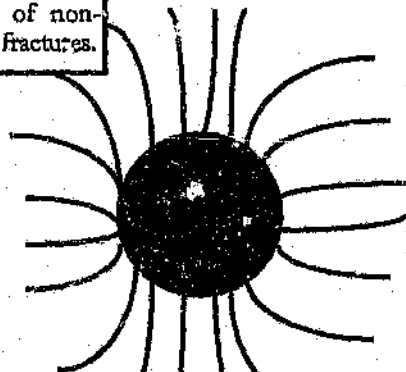


Figure 2.3 Consecutive stages of the fracture process of a fully contained explosion. (After Kutter and Fairhurst, 1971)

It is during this expansion of gaseous products that the host of the rock breaking work is done (Lownds (1986); Brinkman (1987); Britton, Konya and Skidmore (1984), Sarracino (1993)).

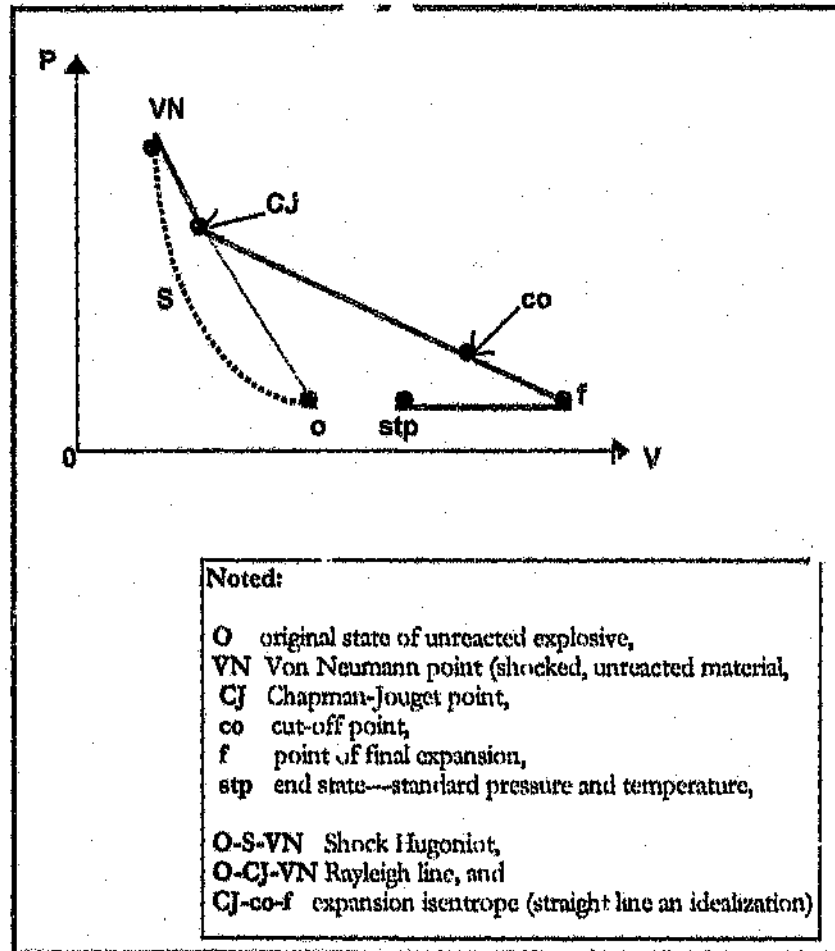


Figure 2.4 Path of a detonating explosive in the P-V plane
(After Sarracino, 1993)

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The action of the gas is termed 'quasi-static' as opposed to the purely dynamic short-lived effects of shock. The gas infiltrates fissures and the pressure extends the network of radial cracks and enlarges the fracture zone. Langefors and Kihlstrom (1963) contended that the gas phase contains the greatest energy share of the explosion. A limited amount of fracture onset has been attributed to this phase by some scientists (Brinkman, 1986) but the matter is debatable. At this stage, this speculation cannot be confirmed because of the lack of proper techniques to separate the effects of the two phases in a blast. Even the technique used by Brinkmann (1987), referred to as the state-of-the-art stainless steel blasthole-lining, can only allow the venting of the explosion gas from the blasthole which enables therefore to isolate the work performed by the shock phase in the rock. But the assessment of the gas effects is only possible by differentiation of the breaking result obtained from a lined blasthole from that of a normal blast (with both phases of shock and gas included). Such procedure of assessment of the gas effects' share in setting the blast outputs may be inaccurate and misleading since it is difficult to produce two identical blasts.

2.2.1.3 Coupling effects

Breaking in blasting depends on the energy partition between the phases of shock and gas. Mel'nikov (1940) found that the distribution of explosive energy between shock and heave could be effectively controlled by a gapped charge in the blasthole. He ascertained that by reducing the initial pressure of the detonation products, and increasing the duration of their action on the rock, the blast energy is redistributed and there is a reduction in the losses, i.e. in inefficient over-crushing of the rock (Mel'nikov *et al.*, 1971).

Britton, Konya, and Skidmore (1984) investigated the primary mechanism involved in breaking rock with explosives at a laboratory scale. They varied the decoupling ratio from 1 to 3, calculated both the explosion shock energy and the gas pressure, sieved and weighed the broken rock materials obtained. They observed that as the decoupling ratio increased from 1 to 3, the mass of the breakage decreased by 29% and the explosion pressure also decreased by 28%. The normalised shock energy remained constant. They concluded that the breakage, stigmatised by both the newly created surface areas and the dislodged inherent fragments by the gas pressure in blastholes, decreases with the increasing decoupling ratio, while the shock energy remains constant. The implication is that the constant stress waves or the shock energy does not contribute significantly to varying the rock breakage in blasting and therefore the primary force by which explosives break the rock is the gas pressure.

In follow-up work, Konya, Britton and Lukovic (1987) backed up their previous investigations by looking into the fragmentation generated by decoupled charges, both in laboratory tests and in the field. Their results are summarised in the plots of Figures 2.5a, 2.5b, 2.5c, and 2.5d. These results show that blasts in which the in-hole coupling is 100 % never achieve optimum breaking results, in terms of the total energy imparted to the rock, the total volume of broken rock and the fragmentation obtained. With regard to fines generation, it was observed that it increases with the increasing degree of the in-hole coupling. These investigations were concluded showing that the first step towards optimising blasting would be to determine the most suitable decoupling ratio under the blast conditions considered. This corresponds to the generation of a maximum energy release within the detonation, hence a better rock fragmentation. Indeed, it is not possible at present to isolate completely and accurately

the effects of shock from those of gas. That is to say, the assessment of the respective roles played by each energy phase during the blast is still a matter of controversy. However, it is well established that induced cracks in rock material are initiated by the effects of shock energy and that the enhancement, extension, coalescence and fragments' formation are due to the action of the explosion gas products.

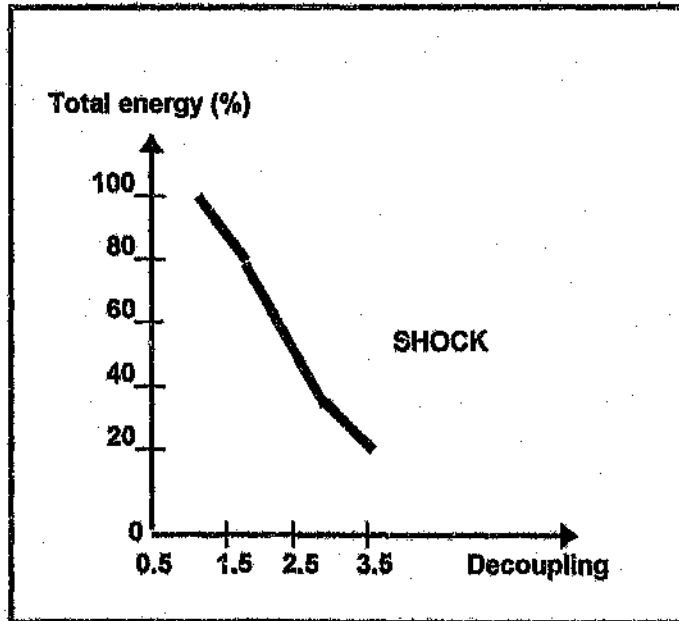


Figure 2.5a Total energy vs decoupling ratio for dynamite (shock energy) (After Konya, Britton and Lukovic (1987))

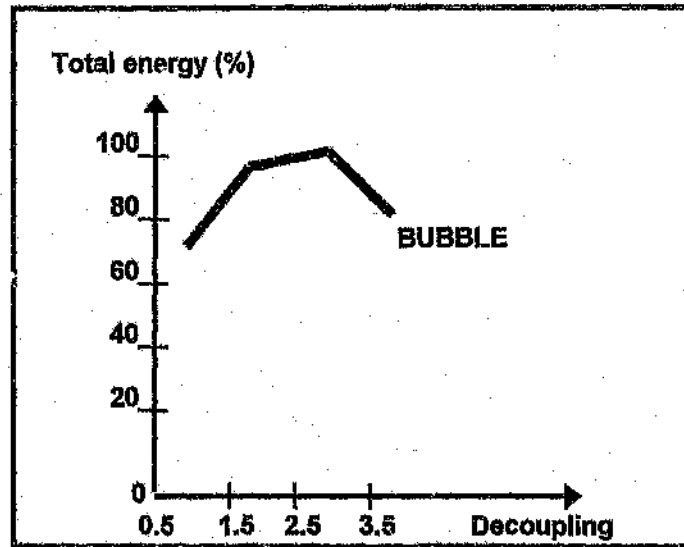


Figure 2.5b Total energy vs decoupling ratio for dynamite (bubble energy) (After Konya, Britton and Lukovic (1987))

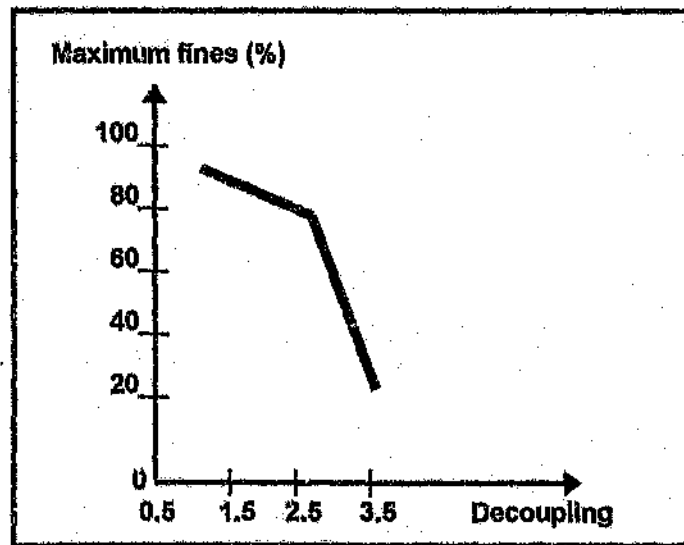


Figure 2.5c Percentage of maximum fines vs decoupling ratio (After Konya, Britton and Lukovic (1987))

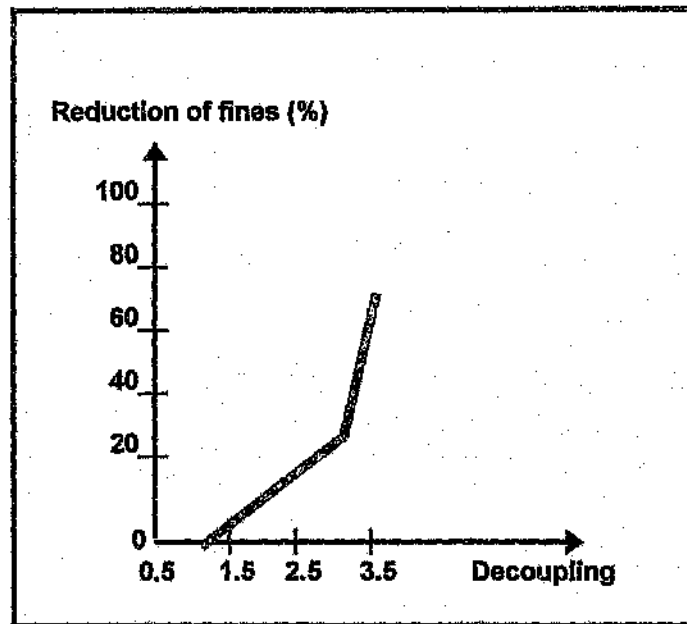


Figure 2.5d Percentage of reduction in fines vs decoupling ratio
(After Konya, Britton and Lukovic (1987))

2.2.1.4 Coupling effects and induced fractures at the blasthole bottom

A particular mode of fractures has been observed in laboratory blasting work by a number of scientists. A so-called 'umbrella fracture' was reported to develop at the bottom of blasted holes in perspex models. This conical crack pattern was observed to increase with an increasing gap at the bottom of the hole. Schultz (1972) reported this finding in a work undertaken at the University of the Witwatersrand in Johannesburg, in perspex cubes of 50 mm x 50 mm x 50 mm which were charged and blasted using a low-energy detonating cord. He attributed the formation of this fracture pattern to the tightness of the fitting of the cord that had not

been pushed right to the bottom of the drilled hole. Seligmann (1972) also observed that the shape and size of the umbrella was not influenced by the distance between the bottom end of the hole and the bottom of the model, thus excluding the effect of wave reflection. He postulated, after careful underground observation, that the shape and size of the umbrella were influenced considerably by the conical shape left at the bottom of the hole by the shape of the drill bits used. By carefully rounding the bottom of the hole, it was possible to eliminate the umbrella shape since sharp corners at the bottom of the hole were perceived to act as stress raisers. The theory of channel effect that was proposed by Johansson and Persson (1970) was used here to explain the mechanism of formation of this conical fracture as follows: The detonation front would cause a high-pressure air plug to be accelerated by the instantaneous momentum of the reaction. This plug would impact on the rock at the bottom of the hole, intensifying localised fracturing which develop to form this fracture pattern.

Cunningham (1970) also reported on conical fractures observed in rock at the bottom of blasted holes underground. He asserted that their formation was due to the impact of the detonation shock head on the bottom of the hole, in addition to stress intensification at the corners of the borehole. He observed that the angle of this conical fracture was related to the shear angle of the material. He thus concluded that the mechanism of formation was shear failure under the high shearing stresses of the distortion pulse generated in the material by the impact of the shock wave on the bottom of the blasthole.

The contribution of Anderson, Winzer and Ritter (1984) was based on investigations of the time-history of the principal strains generated in rock by cylindrical explosives charges, using strain gauges bonded in the

borehole to monitor the strain-time in bench blasting. It was postulated that although very complicated strain histories occurred in the affected rock, the shear strains possibly due to shear waves were likely to influence fractures in rock breaking. These reports highlighted the contribution of shearing to blasting fractures and the associated damage. Although the conical fracture phenomenon has been comprehensively dealt with, more investigations are required to investigate shearing stresses and the extent of their contribution to blast damage.

2.2.1.5 Crushing in blasting

Quian Liu and Katsabanis (1993) acknowledged the difficulty of separating and thus measuring directly the peak shock pressure and the gas pressures in the crushed zone formed around a blasthole in blasting. Instead, they chose a theoretical approach to study the crushing effects of stress waves. Their report indicated that the generation of fines, quantified in terms of mass percentage of the rock blasted, increased with increasing explosive density in fully loaded blastholes. Their experimental work concluded that fines increase significantly with increasing powder factor. The implication is that low-density explosives may be the key to the problem of reducing coal fines in industrial blasting.

2.2.2 Induced damage in coal

Chiappetta (1988) found coal fines generation to be site-specific. He observed that in surface mining of coal, fines originate from various sources that can be classified according to the specific cause of damage, as follows:

1. huge blasts involving large burden/spacing,
2. drilling and blasting into coal,

3. Insufficient backfill of drilled hole in coal,
4. poor decking (use of small charge),
5. poor initiation and explosive performance,
6. bad location of primer,
7. water in borehole or water saturation above or below the coal seam,
8. poor timing
9. dipping and rolling of seams, and
10. Structural geology of the overburden and coal.

Among these parameters, those related to the explosive energy input and its distribution in the coal seam are critical, he says. The worst case of induced damage results from blastholes that are drilled through the coal. In this case, the explosive energy is released directly into the coal that consequently sustains tremendous damage.

2.2.2.1 Drilling and blasting in coal

Coal is subjected to direct dynamic loading. The shock stresses generated by the detonation create an important zone of crushed coal around the blastholes. Expanding gaseous products filter through the fines to produce huge clouds of coal dust associated with heaving of the fragmented coal. The thick column of dust and fumes polluting the environment can be detrimental to human health. The lack of an adequate free breaking surface may result in the heaved coal not moving away fast enough from the face and re-directing gas stresses into the hanging wall (in underground mines) or the highwall (in surface mining) to cause back-breaking. This can be hazardous since deep open fractures in the hanging wall or highwall lead to unsafe working conditions (falls of ground). It is therefore recommended, in surface blasting, to stop overburden drilling above the seam so as to leave

a safe buffer of rock burden to contain the bottom fracturing without damaging the coal in the seam (Coleman, 1994).

2.2.2.2 The problem of drilling through coal

In general, most surface coal operations have difficulty in implementing the recommendation to stop drilling above the seam. Black coal cuttings are always regarded as a signal to the driller to stop the machine. Unfortunately, at this stage, the drill bit is already deep into the coal. Chiappetta (1994) advocated the backfilling of a portion of the borehole drilled into coal. The problem has always been to assess the depth of the hole in the actual coal and the amount of backfill to use to enable effective damage control.

2.3. CONCLUSION

It can be seen that the outgoing compressive shock pulse is responsible for the zone of crushed coal set around blastholes in detonatic Fines generation in blasting can be altered by varying the parameters of the interaction between the in-hole explosive performance and the properties of the medium. In coal blasting where optimisation means reducing the generation of fines, decoupling of the explosive charge would appear to be the way to solve the problem. The most suitable decoupling ratio needs to be determined from optimisation investigations on site.

Fines generation is mainly associated with the comminution resulting from the shock energy phase. Since the VOD of an explosive charge is directly proportional to the shock energy available, increasing it in a blast may enhance the generation of finer particles despite the better breakage it

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may produce. Here again, the decoupling technique is recommended for use to reduce the peak stress (impacting on rock (Bligh, 1974; Mel'nikov et al, 1978 and Hagan, 1979) to mention only few).

The review of previous work has also shown that little research work has been done by way of modelling fines generation in soft rock, for instance coal. It is not sufficient simply to state that coal fines are site-specific. There is a pressing need to study the behaviour of coal subjected to the dynamic loading such as that created by blasting. Conditions responsible for coal damage require comprehensive research as to determine their relation with the fines generation under several site conditions. This will enable adequate modelling of the breaking process therefore optimising the mining operations.

The model must be sensitive to the energy partition between shock and heave. The present thesis attempts a contribution to such modelling by relating the peak strain/stress, the material strength properties and the in-hole charge geometry to the observed breakage in blasted coal.

3 HYPOTHESIS OF THE THESIS

3.1 Basics of modelling fines generation in coal blasting

The main theme this thesis evolves from analysis of the breakage around a blasthole in detonation, as shown in Figure 3.1. Two aspects are considered: the path of the explosives' breaking energy moving through a coal burden and the pattern of the induced damage in the material.

The explosive energy that breaks coal burden is imparted to the latter in two consecutive phases. They are, the "dynamic phase" caused by the action of the generated shock waves which propagate at a supersonic rate into the coal burden and the "quasi-static phase" of the action of the gaseous products resulting from the decomposition of the explosives. These two phases induce two different breaking patterns that create damage of various natures and extent in the coal.

The dynamic phase imparts the breaking energy to the coal instantaneously, mostly by impact. The impulse of the shock waves impacts on the coal of the blasthole walls. The coal is thus loaded dynamically well beyond its resistance and therefore deforms beyond recognition. It is shattered around the blasthole in an annular zone of super-crushed (ultra) fines. The next breaking phase results from the high pressure and high temperature of the gaseous products formed by the detonation reaction. These gases find their way through the network of induced fractures created by the shock phase, extending and widening them to complete the fragmentation. This second phase does not play any considerable role in the generation of coal fines and is thus of little interest to this study.

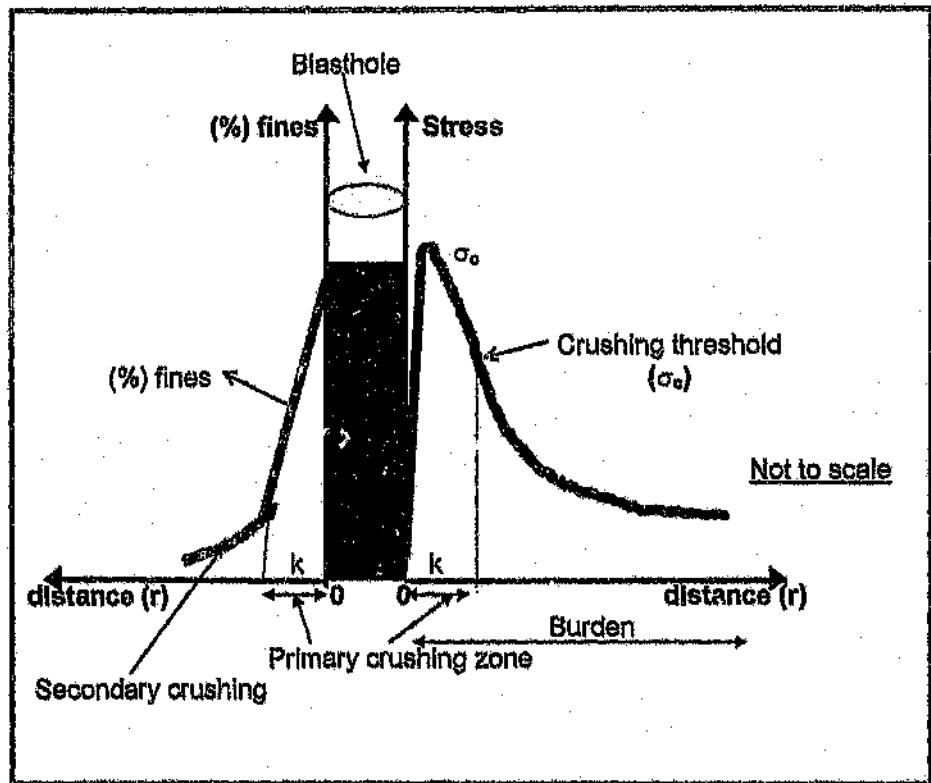


Figure 3.1 Distribution of breaking energy and coal fines around a blasthole in detonation (After Kabongo, 1995a)

3.1.1 Shock energy imparted to coal

The shock energy imparted to coal during blasting depends on the available energy contained in the explosive, the drilled hole diameter and the in-hole confinement. The available energy is measured by the wh Lownds (1986), quoting Cooks (1958), referred to as the Absolute Strer... Value (ASV) of the explosive or its Relative Strength Value (RWS) when compared to Al FFO.

Considering the blasthole during detonation, as illustrated in Figure 3.1, the coal breaking energy develops from the initiation point in the blasthole where it is suddenly increased to the peak following the instantaneous reaction of combustion. The rising time of this energy has been hypothetically assimilated to a linearly increase of pressure and gas products volume within this very short time frame. The peak stress (pressure) σ_0 shown in Figure 3.1, corresponds the highest stress developed at the interface of the explosive and the coal in a tightly coupled blasthole (coupling ratio = 1) or at the interface of the explosive and air (coupling ratio < 1).

Assuming that the conditions of the combustion reaction characteristic of the detonation is complete, this peak energy is directly proportional to the explosive density (ρ) and the square of the velocity of detonation (D). This peak energy may be predicted using a detonation code such as IDEX or CIPEX (Lelper and Du Plessis, 1990) or simply approximated by using the conventional formula for detonation pressure:

$$P_d = \frac{1}{4} \rho D^2 \quad 3.1$$

where P_d is the dynamic pressure, ρ is the density of the explosive and D is the VOD. The peak energy E_0 in coal can be calculated from the equation:

$$E_0 = P_d V \quad 3.2$$

where V is the volume of gases produced per unit mass of explosive.

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After its peak, the energy front enters a different medium and its increasing trend changes. Its prevailing conditions now depend on the dynamic resistance offered by the medium to the energy flow. If an air gap is provided between the explosive charge and the coal, the resulting energy attenuation will be proportional to the decoupling ratio. Johansson and Persson (1970) referred to this condition as resulting from the channel effect of an external air gap on blast output. This may be explained to say that, if the explosive is tightly fitted in the blasthole, then the peak shock pulse impacts directly on the coal, causing an important zone of crushing. Thereafter, it decays rapidly as it propagates through the coal burden, creating various fracture patterns ranging from the primary crushing zone (coal fines), to a network of axi-radial fractures (fracture zone) and a zone of no fractures at all (Kutter and Fairhurst, 1971). Kutter justifies the zone of no fractures to say that the energy front that originally propagated in the burden at supersonic velocity (shock waves) ends up travelling as a sonic and subsonic harmless wave front. But, if on its way, it impinges a free face, this outgoing compressive wave will be partly reflected back into the burden as a tensile wave package, fracturing coal backwards from the free face by spalling effects.

The Figure 3.1 above depicts the relationships between, on one hand the fines generation and the radial distance in the coal burden and, on the other hand the breaking stresses that corresponds to the explosive energy and the radial distance in the coal burden. These dependence curves highlight a significant discontinuity in the respective decreasing trends of fines and that of the breaking stresses. It may be worth noting a trend of rapid decrease in generated fines compared to the decreasing trend in the explosive energy around the blasthole. This trend persists until at some stage where despite the stresses being lower than the crushing threshold

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some fines could be still found in the coal broken (Figure 3.1). This is to say that fines were observed to dwell beyond the imaginable zone of high shock stress concussion in the coal burden. This behaviour of fines suggested that the explosive energy is not the sole parameter controlling fines creation in blasting. There should be an additional mechanism which, alongside with that of the explosive energy, interferes in the creation of fines during blasting. The question in this thesis has been to identify this mechanism and prove its relevance with regard to fines generation in blasting.

The concern on this question elucidated in this thesis was sparked by observations made from several blast results analysed for fragmentation. It observed that the function of the distribution of fines along the axi-radial direction of the blasthole does not decay regularly up to zero fines as it would, if it was dependent solely on the shock energy. Some localised coal fines of various proportions are always found in piles of blasted coal beyond the primary crushing zone, denoted *k* in Figure 3.1. The existence of such type of fines is evidence of a separate crushing mechanism responsible for their occurrence. The hypothesis in this thesis is that these fines result from the interaction between the tailing shock pulse degenerated into elastic stresses travelling in the coal burden and the inherent discontinuities in the material (cleats, in-born open or filled fractures, and other flaws).

Previous research done by the author with the intention of investigating whether the pre-existing fractures in coal have an influence on coal fines generation (Kabongo, 1995a) ascertained that the basic mechanism creating these localised fines in coal involves shearing stresses. They are generated at the tips or lips or both, of the in-born fractures, depending on

the direction of application of the stress. The background of this hypothesis is explained below.

3.1.2 Location of fines in coal burden

Investigations into the source of the fines that were observed in coal piles in relation to the coal burden prior to blasting provided the following basic understanding. It was observed that coal fines generated in blasts of the same characteristics were always variable. Despite the shock energy parameters being kept identical, the amount of generated fines was still seen to vary. In-depth analysis of the blast results showed that the fines originated from three major sources in a coal burden: the pattern of the blastholes centre-to-centre lines, the free face (bottom of the pile) and scattered fines. These types of fines were attributed respectively to the impact of the shock waves on the blasthole walls, the casting dynamic of the face coal (face velocity and impact on the ground) and the inherent fractures.

3.2 Mechanism of crushing around the blasthole

Blasting in coal involves complex loading stresses that vary in magnitude as they travel through coal. The stresses are initially compressive at the interface of the explosive and the coal. They evolve as shearing and tensile stresses associated with the differential loading of the induced and the pre-existing fractures, hoops stresses, and are finally reflected from the discontinuities' boundaries. Coal is always considered a soft and brittle rock material. Its response to such a complex stress situation depends on the rate of loading, which is always a function of the explosive's shock energy release. The latter is in turn dependent on the VOD. The higher the VOD, the bigger the zone of shock energy released therefore the higher

the loading rate of the coal.

Since the intensity of the loading stresses in coal is variable, it is important to optimise their damaging effects throughout. Lizotte and Scroble (1994) reported that the annular zone of over-crushing corresponding to the inception of energy transfer to coal (peak stress) extends around the holes, with a radius not exceeding five times the hole diameter, and generates less than 0.5% of ultra-fines. It is worth noting that purely compressive stresses are limited to this zone which absorbs most of the blast shock energy. The induced radial fractures are initiated at the border of this zone. They are caused by the hoop tangential stresses associated with compression and are responsible for the abundant network of radial cracks formed around the powdered coal zone. Small-size coal particles are yielded (undersize). They are designated as coal fines as long they are sized as minus 6 mm.

Moving further inside the burden, the rate of attenuation of the breaking energy that is dependent on the coal's internal structure, dictates the size of the resulting particles. The crack density decreases. Some single cracks continue growing in preferential stress directions. The energy driving the fracture tips decays progressively through the discontinuities until it dies out. Along their paths, loading stresses interact with inherent fractures to induce differential loading of the fractures features (tip, lips and filling) involving shearing. It is contended in this thesis that the shearing induced in pre-existing fractures is the secondary source of coal fines generation that takes place during the blast. Its effects are studied in next paragraphs, considering three configurations of impacting loading stress on a hypothetical fracture.

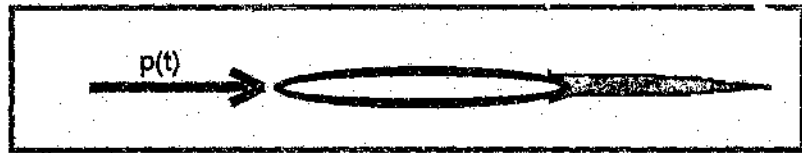


Figure 3.2a Effect of blasting stresses impact along a radial fissure

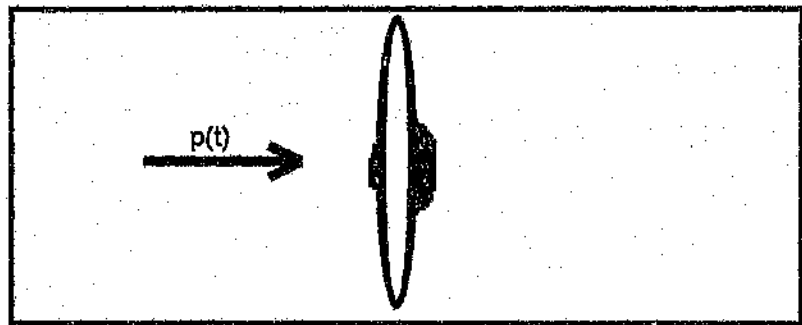


Figure 3.2b Effect of blasting stresses impact along a tangential fissure

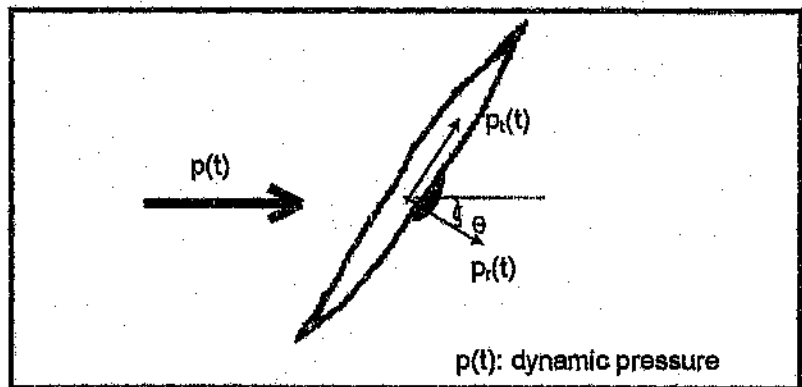


Figure 3.2c Effect of blasting stresses impact along an angled fissure

3.3 Modelling of coal fines in blasting

3.3.1 Primary crushing

Primary crushing of coal essentially depends on the action of the shock waves. It is directly proportional to the brisance¹ energy of the explosive charge which is defined by Lownds (1986) as being the sum of the potential shock energy that is carried through the medium shocked and the strain energy around the borehole behind the shock. The coal of the blasthole walls is the most affected by the dynamic impact of the loading shock waves. The coal is thus over-crushed by these dynamic effects to the extent that is proportional to the peak shock energy imparted to it within a very short time (i.e some microseconds). The relevance of the shock energy contribution into the creation of the fragmentation observed after a blast has been and is still a subject of long-standing debates. However it is commonly agreed by blasting scientists that the this form of energy is responsible for creating the fines and the network of cracks around the blasthole. The instantaneous loading of coal by shock waves occurring in blasting may be modelled in a simple and easy-to- perform test based on the fundamental principles of comminution.

Von Rittinger (1867) established that the energy consumed in a comminution process (A) is proportional to the area of newly developed surface (S).

$$A \sim S \quad (3.3)$$

Kick (1885) found that the work required in breaking is proportional to the reduction in volume (V) of the sample in comminution.

¹ Brisance is the measure of the ability of an explosive to break or shatter material by shock. This characteristic is mostly used for military explosives.

$$A \sim V$$

(3.4)

Bond (1952) came up with the above two theories together to say that the work input is proportional to the new crack tip length, and equals the work input to generate the product minus that represented by the feed. What he said here is that since the surface area of a unit volume of material is inversely proportional to the diameter, the crack length in unit volume is considered to be proportional to one side of the area and therefore proportional to the square root of the diameter. Clearly he established that the energy consumption in comminution is proportional to both the volume and the area of newly created surface.

$$A \sim V, S$$

(3.5)

The general differential equation of all these comminution theories is of the following form:

$$dA = -C \frac{dX}{X^n} \quad (3.6)$$

where A is the energy consumption,

C is a proportionality coefficient,

X is the sample dimension, and

n is the exponent.

- If $n=1$, the equation (3.6) describes the Kick's theory:

$$A_k = -C_k \int_D^d \frac{dX}{X} = C_k \ln \frac{D}{d} \quad (3.7)$$

where $C_k = K_1 \sigma^2 V$;

with K_1 being the coefficient of measuring units,

σ the strength characteristic of the rock,

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- V the rock volume being broken,
 D the average diameter of the rock lump before breaking and
 d the average diameter of rock after breaking.

$$\text{Thus: } A_R = -K_1 \sigma^2 V L n \frac{1}{d} \quad (3.8)$$

- If $n=2$, the equation (3.6) describes the Rittinger's theory:

$$A_R = -C_R \int_D^d \frac{dX}{X^2} = C_R \left(\frac{1}{d} - \frac{1}{D} \right) \quad (3.9)$$

where $C_R = K_2 \sigma \varepsilon \Delta V$;

with K_2 being the coefficient of measuring units, and

ε the coefficient of surface energy consumption per unit area.

$$\text{Since } V = \frac{\pi D^3}{6} n \text{ before or } V = \frac{\pi d^3}{6} m \text{ after the blast} \quad (3.10)$$

with n being the number of lumps before the blast,

m the number of lumps after the blast

then:

$$A_R = K_2 \varepsilon \left(\frac{\sigma \pi d^3}{6} m - \frac{\sigma \pi D^3}{6} n \right) = K_2 \varepsilon (\pi d^2 m - \pi D^2 n) \quad (3.11)$$

but $\pi D^2 n = S_0$ is the area of the rock surface before the blast and

$\pi d^2 m = S_1$ is the area of the rock surface after the blast,

then: $\Delta S = S_1 - S_0$ is the area of the newly developed surface,

therefore:

$$A_R = K_2 \varepsilon \Delta S \quad (3.12)$$

The quantity $\frac{1}{\varepsilon}$ is called the Rittinger number which characterises the

maximum area being newly developed by the stroke of a unit energy on the crushing medium.

- Lastly, if $n=1.5$, the equation (3.6) describes the Bond's theory.

$$A_B = -C_B \int_D^d \frac{dX}{X^{1.5}} = C_B \left(\frac{1}{\sqrt{d}} - \frac{1}{\sqrt{D}} \right) \quad (3.13)$$

$$\text{where } C_B = K_3 \lambda 1.35 \pi^{1.52} V^{1.83} \quad (\text{Boukharov, 1972}) \quad (3.14)$$

with λ being the bulk characteristic of the rock which characterises its strength and surface energy consumption per unit area, therefore (3.13) becomes:

$$A_B = -K_3 \lambda V \Delta S \quad (3.15)$$

The conditions and limitations of usage of the above law of comminution have been a subject of contention until nowadays. However, Hukki (1975) showed that the actual relationship between energy and particle size is a composite form of the three laws. He recommended the use of Kick's law in the crushing range of about 10 mm diameter; that of Bond in the crushing range of the conventional rod-mill and ball-mill and the Rittinger's law for the fine grinding range of 0.01 to 1mm (Wills, 1989). Tackling the same question, Boukharov (1972) quoted that the total energy consumption by rock in blasting including the energy consumed in the rock deformation and that consumed in the formation of new surface may be assessed from the following equation of Rehbinder:

$$A = \frac{\sigma^2 V}{2E} + \frac{s \Delta S}{V} \quad (3.16)$$

In which σ is the strength characteristic of the medium,

E is the Young modulus,

s is the coefficient of surface energy consumption per unit area,

ΔS is the area of newly developed surface, and

V is the volume of the medium considered.

It may be showed that the equation (3.16) is in perfect agreement with the results of Hukki's investigations. In blasting, the rock fragmentation observed in much piles is always created by effects of two distinct breaking actions. They involve the primary breaking by compression which results in crushing and liberation of new free surfaces, and the liberation of the in-situ inherent fragment block formed by the intersection of sets of inherent discontinuities Scott, Lutombo and Kleine (1993). Depending on the distance from the blasthole and the predominant action in the burden, the breaking equation (3.16) may be skewed either toward the law of Kick or that of Rittinger as shown in the following development.

Considering the equation (3.16), If $\frac{\Delta S}{V} \ll 1$ the equation transforms into the law of Kick's. This means that the fragmentation will be dominated by big lumps formed mostly by the phase of expanding gas products which favours the liberation of inherent blocks in a fractured rock medium, i.e. coal. However, if $\frac{\Delta S}{V} \gg 1$ the equation transforms into the law of Rittinger. In this case the fragmentation will rather be fine mostly due to the compressive shock action of the blast.

Practically speaking, the determination of the energy consumption in blasting is a very complex problem. While the exponential coefficient n in the general equation of comminution (3.6) has been related to the radial distance in the burden and the prevailing law of energy attenuation in the medium, the main complication in modelling comminution in blasting remains the determination of the coefficient of proportionality C . This coefficient $C=(C_R, C_K, C_B)$ depends on the properties of the medium which is neither homogeneous nor isotropic. Boukharov (1972) pointed out that the failure to determine the C coefficient which ought to take into account all feature characteristics of rock medium, has left scientists with no other alternative than to resort to experimentation.

The experimental laboratory test suggested in this thesis consists of dropping a hammer weight on a coal sample placed at the bottom of a mortar, as shown in Figure 3.3, so as to generate conditions of dynamic loading similar to those occurring in blasting. Despite the small scale of the test and the eventual discrepancies due to the size-effect since cores are extracted from the coal of the seam, it is assumed that the fundamental physical behaviour with regard to coal strength properties remains the same. It is also assumed that the crushing, therefore the generation of fines that occurs as a result of the loading energy imparted to the coal through the impact of the weight is only dependant on the incident shock waves. It is not influenced by the reflection of shock waves that may occur from the ends of the coal samples.

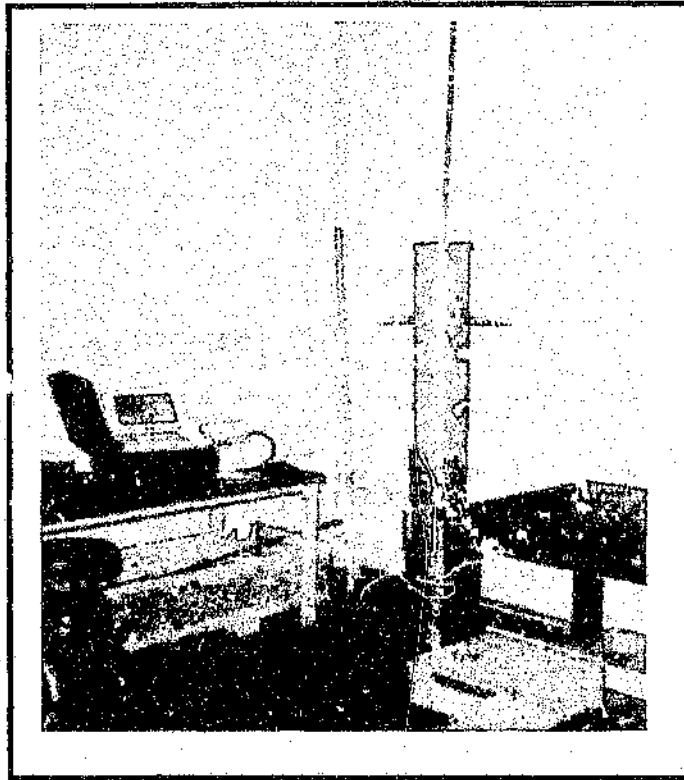


Figure 3.3 Set-up for the dynamic impact test
(After Kabongo, 1995b)

3.3.2 Secondary crushing

Beyond the shock zone, sonic stresses interact with inherent discontinuities in the coal to generate additional fines. The discontinuities may be in-born micro- and meso-fractures or flaws such as grain boundaries. They are centres for the nucleation or extension of cracks which, depending on the spatial location and the activating stresses, can contribute to either fragmentation (coal lumps) or fines.

3.3.2.1 Fracture in a radial direction

In a fissure oriented in a radial direction, the outgoing radial loading stress $p(t)$ shown in Figure 3.2(a) is transmitted wholly or partially, according to the nature of the fracture (open or closed) and the properties of the fill material, to the extreme tip of the fissure. It then drives the tip further inside the coal splitting it into upper and lower fragments (coal lumps). This is typical of the behaviour of layered coal when the blasthole is drilled perpendicular to the bedding planes. Breaking takes place preferentially along the layers' planes of weakness and very limited fines are generated. The fracture mechanic, in this case, is that the blasting pulse pressurises the crack, widening it by expansion and extension until the driving stress at the tip falls under the critical limiting value called 'stress intensity factor' that is characteristic of the coal toughness and its mode of failure. It is worth to underline that in terms of the mode of loading with respect to the crack surface, three types of failures modes may be distinguished. They consist of the mode I that is characterised by the stress intensity factor K_{Ic} occurring in material loaded in pure tension. The mode II is characterised by the stress intensity factor K_{IIc} that occurs in material loaded in pure shear along the plane of the crack. The third type is the mixed or combination of modes I and II that is believed to occur very often in mining related rock fracture problems (Outcherlony, 1974). Rossmanith (1994) clarifies these definitions by considering an infinite rock mass containing a small crack to define the mode I fracture as occurring when the loading force (tensile) is applied to the crack lips resulting in extending the crack tips. However the mode II occurs if a shear stress is applied and a combined modes I and II occurs if both type of stresses are applied. The critical stress intensity factor K_c is understood to correspond thus to the stress singularity at the fracture tip that causes failure.

3.3.2.2 Fracture in a tangential direction

The radial outgoing loading pressure $p(t)$ impacts on the lip of the fissure. Depending on the nature of the fracture and that of its fill, it is either stopped at the interface (barrier) of the two materials (coal and fill, air or void) or it loses some energy here but manages to cross over with reduced magnitude (Kutter and Kulozik, 1994). At the moment of impact, the momentum of the propagating pressure produces compressive stresses which, depending on their magnitude, may generate some coal fines locally (see Figure 3.2(b)).

3.3.2.3 Fracture randomly oriented

Assuming that no loss of energy occurs at the point of interaction (see Figure 3.2c), the radial explosion stress $p(t)$ will be decomposed into two components, $p_r(t)$ and $p_t(t)$, respectively along the radial and the tangential directions. The component $p_r(t)$ is essentially compressive, whereas $p_t(t)$ is tensile. The effects of the component $p_r(t)$ are opposed to the compressive strength of the coal which once again, according to the ratio of magnitude of the loading stress to the coal resistance, will sustain crushing and fines generation as in the case shown in Figure 3.2b. The tangential component $p_t(t)$ involves a combination of tensile and shear. Its effects will contribute towards driving the fissure's tip deep into the coal along the direction of the fracture at the angle θ , as shown in Figure 3.2a. It will cause some extension of the tip until the magnitude of its driving pressure goes beyond the intensity factor, which is characteristic of the toughness of the material. The angle θ is known as the friction angle that determines the direction along which the fracture's lips may develop shearing stresses. Depending on the stress transmitted and the coal's shear strength, additional shear crushing, controlled by the mode II fracture stress intensity factor K_{II} , may develop.

3.4 Conclusion

Two basic crushing mechanisms contribute towards the generation of coal fines in blasting: shock-driven primary crushing and secondary mechanisms dependent on the inherent fractures. Much controversy has been going on about which is the greater contributor to the generated fines observed in coal piles after blasting. However, the concern in this thesis is rather on the mechanism that may be controlled so as to reduce fines generation in mining operations. The shock-generated primary crushing appears to be controllable. It may be controlled through the rate of the explosive energy release in the blasthole that is a characteristic of in-hole charge performance in-situ. The higher the explosive strength in a tight-fit coupled blasthole, the bigger the annular zone of fines created around the hole. The dynamic loading of the mechanism can be simulated in an energy-controllable comminution laboratory test, as demonstrated in the following chapters. The secondary crushing mechanism is dependent on the pre-existing discontinuities (in-born fractures, flaws, etc.) which may be difficult to assess. The hypothesis formulated in this chapter was tested in a series of investigations reported on in the subsequent chapters.

4 BLAST-SIMULATED DYNAMIC LOADING OF COAL

4.1 Introduction

The mining of coal in South Africa, both in surface mining and in some underground mines, makes use of explosive energy to extract coal from the seams. This mode of breaking generates damage that is mostly caused by the instantaneous loading of coal by the high stresses generated by the detonation. Whatever size distribution the blast is designed to produce, the huge amount of energy released within a short time during the blast subjects coal to sudden loading which results in it fragmenting into three size groups: over-, mid- and under-sizes. The amount of energy and the time frame of its release make investigation into the breaking process very difficult. To date, no reliable sensor, capable of withstanding such conditions, is commercially available, and so this thesis avoided direct measuring of blast energy in coal. However, it has decided to model blast loading conditions of coal in a simple comminution test that is dealt with in this chapter.

4.2 Dynamic behaviour of coal

Coal breaking in blasting is governed by the stress-time parameters of the blast energy involved and the dynamic resistance to fracture opposed by the material. The relationship between the energy imparted to coal and the time may be represented by a single pulse propagating through the burden and determining the corresponding zones of coal deformation, as shown in Figure 4.1. Two distinct zones that are created in the coal burden are known as the stable zone in which coal is not affected by the blast effects and the failure zone where the blast has caused damage (Kutter and Fairhurst, 1971). The extent of the damage depends on the energy attenuation properties within the burden. These are dependent on the in-

situ strength of the coal, i.e. the internal structure and the inherent discontinuities (Zipf and Bieniawski, 1989).

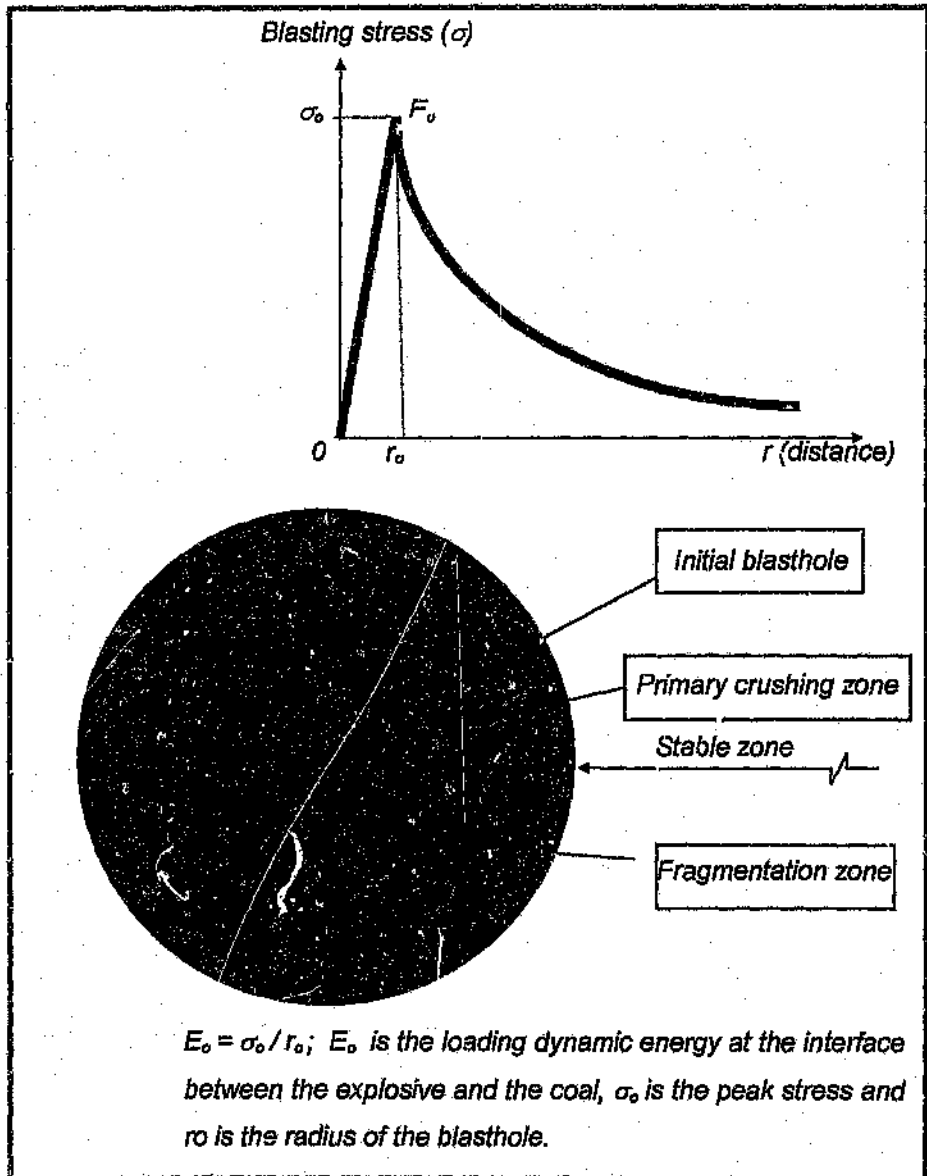


Figure 4.1 Dynamic loading energy and subsequent coal response

The failure zone has two sub-zones: the crushed and the fragmentation zones. The crushed zone is characterised by the high magnitude of the loading energy, corresponding to the steep part of the function of the energy history in the radial direction. This is evident in the high rate of loss of the breaking energy consumed during the creation of the high density of cracks characteristic of the comminution zone around the blasthole. The fragmentation zone consists of coal with particle sizes above the cut-off size considered for fines. The rate of energy loss through coal is reduced but the stress level is still some way above the strength resistance of coal. Consequently, it breaks into bigger fragments. The damage criterion hence formulated is that coal is only damaged in a blast if the stresses generated are higher than the critical stress σ_c and if their loading time is shorter than a critical pulse time τ_c characteristic of the strength of the particular coal. This criterion may be used as a guideline for in-situ blast optimisation, i.e. to choose the most suitable explosive and blast geometry parameters (type, charge diameter, decoupling ratio, burden, spacing and timing).

4.3 Interaction between explosives and coal

Like any other industrial comminution processes, blasting is inefficient. Usually, the energy available in the explosive is not used effectively in coal breaking. The rate of energy release in a blast needs to match the physical properties of the coal in order to enable the optimum breaking to occur (Udy, 1994). Effective rates of energy release in coal breaking depend on the explosive performance characteristics inherent to its composition i.e. the dynamics of pressure-volume (see Figure 4.2 (Lownds, 1992)) and the attenuation properties imposed by the material composition i.e. the function stress-time (Rhevesky, 1978). The achieved degree of interaction between the explosive and coal may be elucidated by a recorded signal

trace of the function of stress-time (stress-distance) also known as the function of the breaking-history developed in the coal burden during the blast shown in Figures 2.1 and 4.1. It highlights the decaying amplitude of the blast stresses travelling through the coal burden. This breaking-history information, together with the coal breakage observed in blast piles should provide an excellent basis for an insight into the comminution process which has taken place. Unfortunately, it is not possible to capture reliable information in-situ because the current range of sensors cannot withstand the high magnitude of the energy involved. The approach in the present research involved laboratory investigations simulating the blast-dynamic loading. The energy partition in blasting was modelled by altering the falling height of a hammer mass on a coal sample placed at the bottom of the stamp mortar apparatus devised for the simulation. This apparatus is presented and its functioning explained in the next chapter. However, in the fieldwork undertaken, it was not possible to vary the explosive material. Instead, the partition of the breaking energy input into coal was varied through changing the in-hole geometry of the explosive charge. This resulted in variable amount of shock and heave energy, depending on the decoupling ratio, being imparted to coal during the blasts. The reasoning in these field trials was based on the simplified model of energy partition in blasting proposed by Lownds (1986). He illustrated that the energy released by an explosive to the blast may be considered equal to the area underneath its isentrope characteristic curve (the function of energy delivery from the shock Hugoniot point of the detonation) as shown in Figure 4.2.

It is worth reminding that theoretically, two important energy stages may be distinguished in any detonation. They are on one hand, the adiabatic compression of fresh explosive material that causes a rise of pressure,

The energy imparted to the blast is delivered through the expansion of the detonation products along the characteristic isentrope curve illustrated in the Figures 4.2 and 4.3. Along this isentrope, the expanding detonation products perform work losing therefore internal and kinetic energy to the benefit of the various forms of energy in the blast namely shock wave and wasted energy.

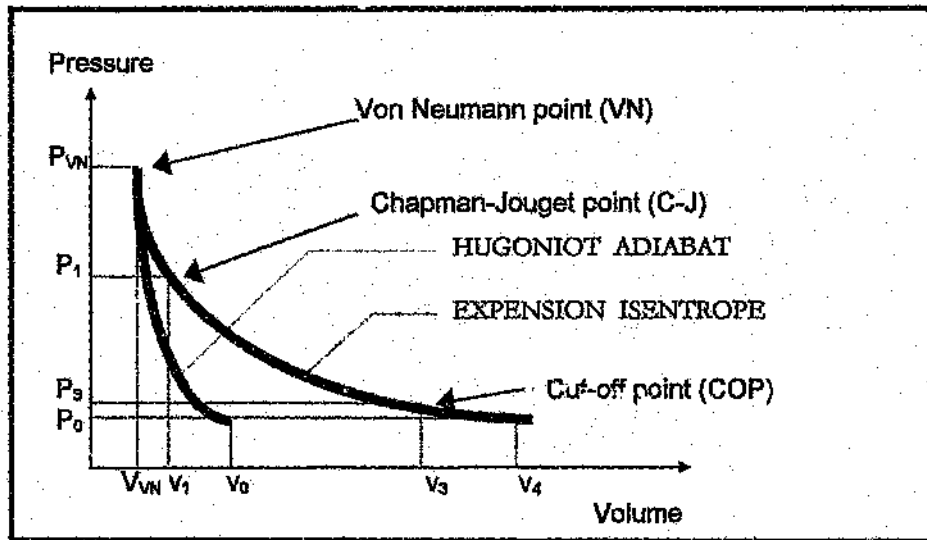


Figure 4.3 Illustrative P-V diagram of a shock and reaction Hugoniot

The physical meaning of different stages of detonation shown in Figures 4.2 and 4.3 is briefly presented in the above list of major points of the reaction path in the pressure vs volume plane:

0 (P_0, V_0): explosive material at the initial stage of atmosphere conditions,
(before the initiation),

- 4 (P_0, V_4): expanded gas products at the final stage of atmosphere conditions (after escaping at the free face),
- 1(P_1, V_1): C-J point corresponding to the peak energy available to the blast,
- 2(P_2, V_2): energy at the interface between the shock and gas phases, dependant on the in-hole axial and longitudinal confinement of the explosive charge, and
- 3 (P_3, V_3): the cut-off-point, which means the limit of the effective breaking energy available to the blast.

4.3.1 Comminution energy in coal blasting

The shock energy is responsible for the comminution and the network of induced fractures in coal (primary crushing). It also influences the extent of the secondary crushing, depending on the geology of the coal (Kabongo, 1997). The second breaking stage which involves the effects of expanding gaseous products may also generate additional crushing by shearing and other complex mechanisms dealt with earlier. Otherwise the generated gas sweeps through the zone of fines previously set around the blasthole, infiltrate into the cracks and extend them to complete the fragmentation and so move the fragments away from the solid coal.

The total energy availed to the blast by an explosive charge is called the "Absolute Strength Value" (ASV), or "Relative Weight or Bulk Strength" (RWS or RBS) when compared in terms of weight of volume of ANFO, (Cook, 1958). It is transferred to coal partly in the form of shock energy input (E_{si}) and partly in the form of heave or gas energy input (E_{gi}). Part of the E_{si} is consumed effectively in breaking the coal (E_{se}) and the other part (E_{io}) is wasted in various forms such as heat and loss at the free face. Thus one can write the following equation:

$$E_{si} = E_{se} + E_{lo}$$

4.1

Assuming that the energy losses are kept constant through use of similar blasting conditions (same coal i.e. physical properties assumed identical, same explosive charge and accessories, same blast geometry) it can be written that

$$E_{si} = F(E_{se})$$

4.2

where F is a linear function relating E_{si} to E_{se} .²

The law of comminution applied here would mean that the effective shock energy imparted to coal which is responsible for causing the comminution zone, is proportional to the coal fines generated therein. This effective shock energy (E_{se}) can be measured through the fragmentation observed in the coal pile after the blast. But, since the observed fragmentation corresponds to both the comminution energy ($ComE$), and the fracture energy ($FracE$), considering that all losses of energy at the free face have been accounted for in the E_{lo} of equation 4.1 above, it may therefore be stated that:

$$E_{se} = ComE + FracE$$

4.3

$ComE$ is measured through the generated fines observed in the blast pile. Similarly to the above, the following function can be established:

$$E_{se} = f(ComE)$$

4.4

² This is a first approximation of the practical reality observed in blasting. Increasing the energy input in the rock either through the powder factor or the use of a higher energy explosive results in an increase in rock breaking (more fines and reduced coarse fractions) as shown in Figure 6.6. This clearly shows the existence of a proportional relationship between the energy input and the strain energy in the rock that is responsible for breaking. The nature of this relationship is a subject for controversy. Nicholls and Duvall (1963) ascer that it is a "some function of the ratio of the characteristic impedance of the explosive to that of the rock". In the absence of any contrary argument this thesis assumes it linear for the sake of convenience.

where f is called the "appearance function of comminution" which is a characteristic of the coal type and the blasting conditions considered. It can therefore be deduced that f is a function of the input shock energy and the dynamic of its propagation through coal. That is to say, f is a material function.

4.3.2 Modelling of comminution in blasting on a reduced scale

The intention was to simulate the loading conditions of the shock waves generated in blasting at a controllable energy level. The difficulty in assessing the breaking process in field blasting due to the huge energy scale involved was overcome by devising a laboratory test to enable the prediction, control and optimisation of comminution in coal. The test technique allowed an evaluation of the fines generated from coal samples submitted to dynamic loading stresses by the free fall of a hammer mass in a mortar, as described in the next chapter, to measure the coal's dynamic crushability. The Coal Dynamic Crushability (CDC) is a measure of its ability to resist dynamic crushing stresses.

4.3.3 Coal dynamic crushability (CDC) test

The idea here was to analyse the complex phenomenon of coal response to the blast-generated stresses through a series of discrete simulations of shock and gas actions in processes that may be controllable. The impact stamping mortar used was a steel mass of 20 kg. The hammer was dropped in free fall on a coal core sample from an indicated height. The height of the fall could be varied to create different levels of impacting gravity energy. The kinetic energy input into the coal sample was determined and related to the effective breaking energy (comminution

energy, ComE) assessed from the magnitude of the strains recorded on the core coal sample. The strain deformation history was recorded from each sample by strain gauges bonded onto the core. The fragments of broken coal were collected and sieved to assess the resulting size distribution and in particular the fines generation. The technique simulated the effects of shock energy in blasting relatively well, despite the limitations of energy size and loading times involved. Fair correlation was observed between the results of the laboratory and field blasting tests, as discussed in the next chapters.

4.4 Background to the experimental work

The dynamics of the shock waves released in a coal blast are dictated by the rate of detonation of the explosives used. During a blast the coal mass is shocked, pressurised, heated up and subsequently damaged (Kabongo, 1995b). The observed damage ranges from coarse, well-fragmented coal to fines that end up in the waste heap. The challenge in commercial coal mining is to minimise the fraction of unusable fines in favour of the saleable fragmentation sizes so as to optimise the economics of the operation.

The approach to modelling blasting stresses adopted in this thesis was to use mechanically created shock waves in coal samples. As in previous work presented by Kim and Mc Carter (1993), the breaking stresses in core-drilled coal samples were generated by impact loading in the apparatus described in the next chapter.

4.4.1 Dynamic strain measurement

Core specimens of coal were examined with the naked eye for innate

meso-fractures. In some cases, thin polished sections were prepared for electron microscope assessment of the inherent micro-fractures (see Figure 4.4). Foil strain gauges were bonded onto the core samples in a half-bridge resistive circuitry to measure strain in both the longitudinal and radial directions (Figure 4.5). Strain recording was triggered automatically when the loading shock stresses in the coal at the bottom of the guiding steel tube of the mortar reached a pre-set sensitivity level. The strain signal was amplified in a double-channel sensitive amplifier before being recorded. The recording system consisted of a high-speed blast wave storage oscilloscope coupled to a 486 laptop PC for data capturing and using appropriate software (see Figure 4.6). The test is user-friendly and may be easily performed either in the laboratory or on site. It may well be the answer to Holmberg's (1993) recent call for simple and easy-to-use on-site techniques for predicting blast results in order to make optimisation decisions on site.



Figure 4.4 Inherent micro-fractures in coal seen at the Electron



Figure 4.5 Foil strain gauges bonding on coal core samples for test

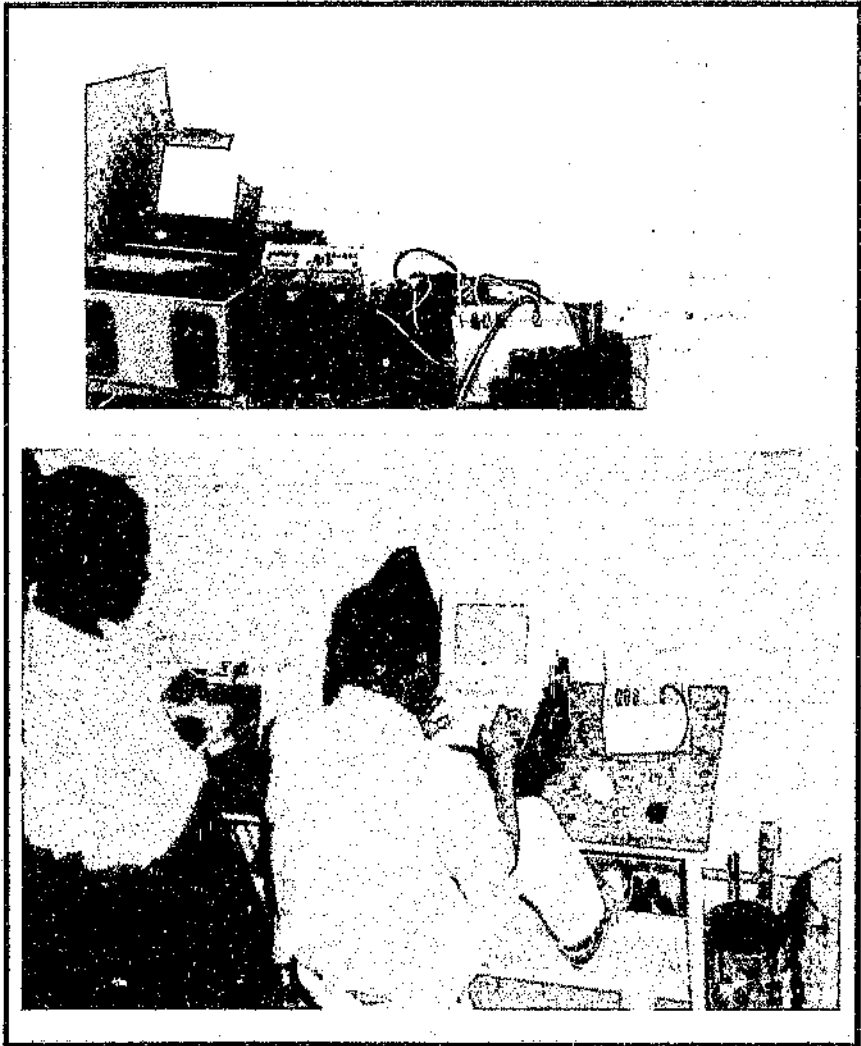


Figure 4.6 Recording equipment used to capture strain variation in tested coal samples

4.4.2 Critical stress

Classic uniaxial compressive strength tests were conducted on a number of coal samples from the same coal blocks. Ultrasonic waves were propagated through the coal to evaluate its physical and mechanical properties.

The dynamics of the impact stresses responsible for damages were deduced from the traces of the strain history (oscillogram) recorded. Typical signal records are shown in the next chapter. The traces were analysed on the assumption that the dynamic behaviour of the coal material obeys the elastic theory, i.e. induced stresses produce proportional strains. This implied that at failure (interface of the elastic and plastic zones), the Hooke's equation could be applied as follows:

$$\sigma_c = E_d \delta_c \quad 4.5$$

where σ_c is the critical stress,

E_d is the dynamic modulus of elasticity and

δ_c is the critical strain observed.

4.4.3 Pulse energy

The pulse energy responsible for developing the observed strain and the resulting damage was assessed from the equation of Steverding (1976) as being proportional to the square of the critical breaking stress (σ_c) and its duration τ . The integration of the area underneath the strain signal was used to measure the strain pulse energy:

$$W = \int_0^{\tau} \sigma^2(t) dt \quad 4.6$$

In view of the dynamic mode of the impact (accumulated kinetic energy proportionally transferred to coal according to its properties), the total energy imparted to coal was assessed from Kim and Mc Carter relation:

$$W = AE_d V \int_0^t \delta^2(t) dt \quad 4.7$$

where W is the coal-breaking energy,

A is the cross-sectional area of the impact,

E_d is the dynamic elasticity modulus of coal,

V is the velocity of the weight at impact,

δ is the strain reading recorded in the coal and

t is the current time.

4.5 Concluding remarks

The primary crushing mechanism yields most of the reducible coal fines. The present thesis aimed at investigating ways of predicting, controlling and optimising fines generation in blasting and sought therefore to establish a relationship between the input energy, the loading time and the resulting sizing of broken coal. This chapter has laid down the theoretical background on which the discussion in the next chapter is based. Thus, the next chapter presents the laboratory investigations done at the University of the Witwatersrand and the main results obtained.

5 LABORATORY INVESTIGATIONS

5.1 Objectives and similarity criteria

The Investigations were aimed at providing an insight into the process of coal breaking in a dynamic loading mode. Understanding the process is the key to controlling the mechanism and to predicting the end-results. The qualitative blast work done in laboratory at the Department of Mining Engineering of the University of the Witwatersrand used a transparent continuum medium (perspex blocks) to provide an insight into the creation of the fracture zones around the blasthole during detonation. The quantitative trials attempted modelling the full-scale in-situ blasting through a dynamic crushing test based on the following physical similarity criteria:

1. Similarity of density: The coal samples used in the trials were collected from various mines of the country namely Kleinkopje, Greenside and Tavistock, Arthur Taylor Colliery. The density of coal in the models could be considered similar to that in-situ, except that the modelling of natural fractures in coal could not be guaranteed (due to size effects).
2. Similarity of geometry: Cylindrical explosive charges in detonation involve cylindrical expansion of blast stresses around the blasthole. This induces a compressive loading in coal confined around the blasthole similarly to that in the test model. The test model used coal samples core-drilled to 42 mm diameter, approximately 105 mm long (length of 2.5 times diameter), with an average density of 1.450 g/cc. These were used to model the coal burden between consecutive blastholes in a blast (multiple-holes blasting).
3. Similarity of the breaking mechanism: The loading conditions

responsible for breaking were scaled down to controllable levels in the model, while maintaining the dynamics of the impact loading characteristics of the blast-generated shock stresses. Mechanically generated shock waves were imparted to the coal specimen through the impacting mass hammer falling freely from a given height. The free fall of the hammer produced measurable dynamic compressive loading stresses that broke the samples. The kinetic energy input responsible for the breakage was determined. The dynamic strain gauges bonded on the samples recorded the stress history in the specimen during the loading. The ratio of the energy input to that effectively used in coal breaking called the "comminution energy efficiency" (CEE) Index was determined.

4. The third theorem of similarity that was postulated by Borovikov and Vanyagin (1995) states that "two bodies prepared from the same material similar before the application of external forces, remain similar after the application of forces that are distributed over surface (volume) in a similar manner". The theorem adds, "the values of such forces per unit area (volume) in both bodies are identical". According to this theorem, it may be considered doubtful that fragmentation in the model is similar to that in full-scale blasting due to the effects of inherent fracturing, but the fines generated in the model are indisputably similar to those generated by full-scale shock waves.

5. Techniques used and results obtained

5.2.1 Blasts in perspex models

Laboratory blasting work undertaken involved detonating small charges of PETN explosive in small holes of increasing diameters (2.4, 3.0, 4.0 and 4.5 mm). The idea was to vary the in-hole coupling ratio of the explosive

charge in the blast so as to investigate the effect on induced fractures resulting from the blast. Changing the in-hole coupling ratio amounts to modifying the partition of the blast energy between its component 'shock' and 'heave' phases. The test blasts were carried out in perspex cubes (50 mm x 50 mm x 50 mm). Perspex was chosen to allow visual assessment of the results. The cubes were pre-loaded bi-axially (static loading in two directions, vertical and horizontal loading) on an AMSLER hydraulic press to simulate the underground conditions of coal in-situ confinement around the blastholes. The blasts were fired through the free face as shown in Figure 5.1a. The distribution of the blast-induced fractures around the blastholes was assessed (see Figure 5.1b). The perspex cubes were later sliced along the major principal stress direction to measure the extent of the longest crack induced.

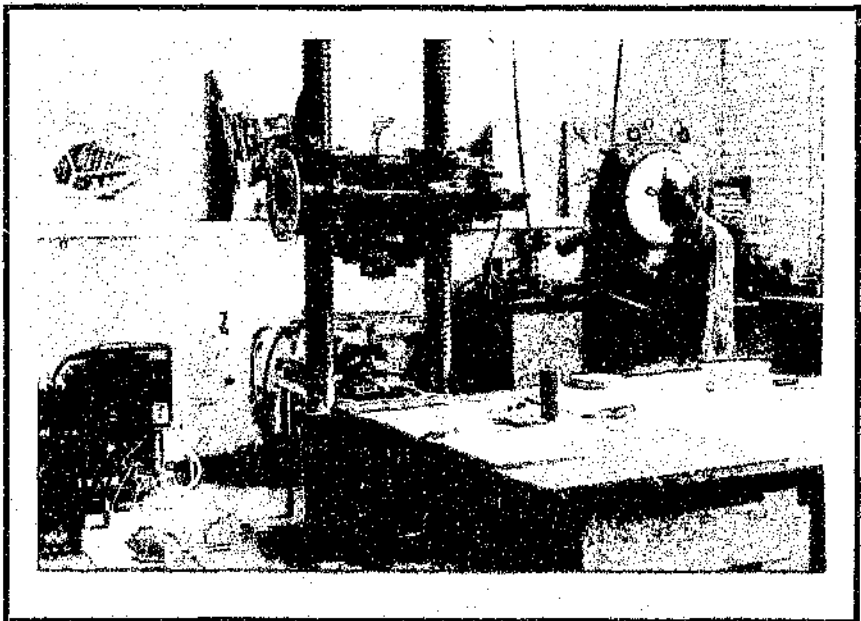


Figure 5.1a Laboratory blast set-up for investigating induced fractures

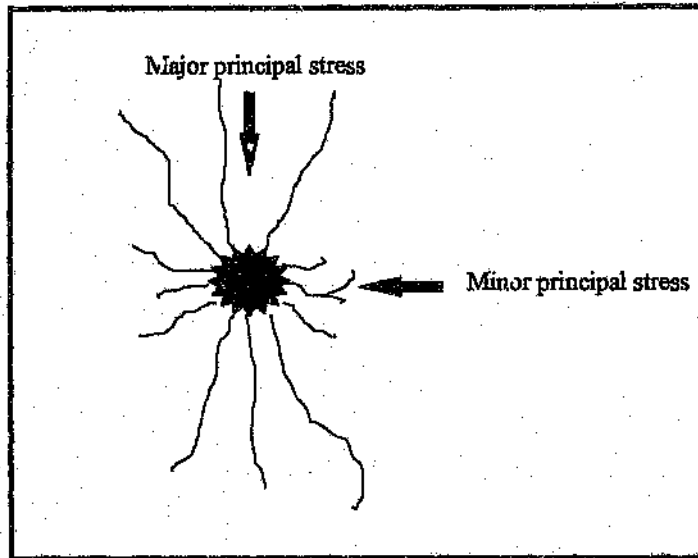


Figure 5.1b Typical fracture pattern induced by blasting a confined
medium

The results obtained are summarised in Table 5.1. They are summarised in the graphic of Figure 5.2 that is supported by the pictures in Figures 5.3 to 5.5. These results demonstrated that increasing the annular gap between the explosive and the blasthole walls (increasing the explosive charge decoupling, in other words, reducing the in-hole coupling ratio) produced the decreasing of the effects of the energy phase of shock waves in coal. This led therefore to an extension of the induced fractures subsequently due to the compensatory increase of the gas phase effects. The consequence of the reduction of the contribution of the shock energy was evidenced by a clear reduction in the extent of the resulting zone of crushing around the blasthole. It was noted, however, that the extent of the overall zone of the induced fractures was increased. This was evidence of

the redistribution of the total energy available to the blast which occurred between the two phases of shock and gas. The loss of shock energy enhanced the gas phase, resulting in the observed extended fracture zone (Kabongo, 1995c).

Table 5.1 Effects of decoupling on fracture extension in perspex models

Hole diameter (mm)	Explosive diameter (mm)	Decoupling ratio (Δc)	Fracture extension (mm)
2.4	2.4	1	16
3.0	2.4	1.56	24
4.0	2.4	2.78	28
4.5	2.4	3.52	20

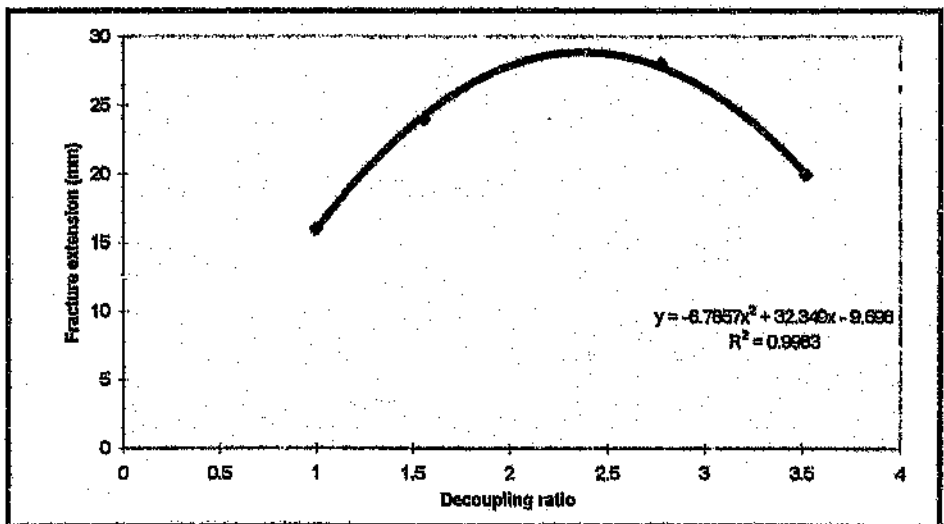


Figure 5.2 Dependence of induced fracture extension on the charge decoupling ratio

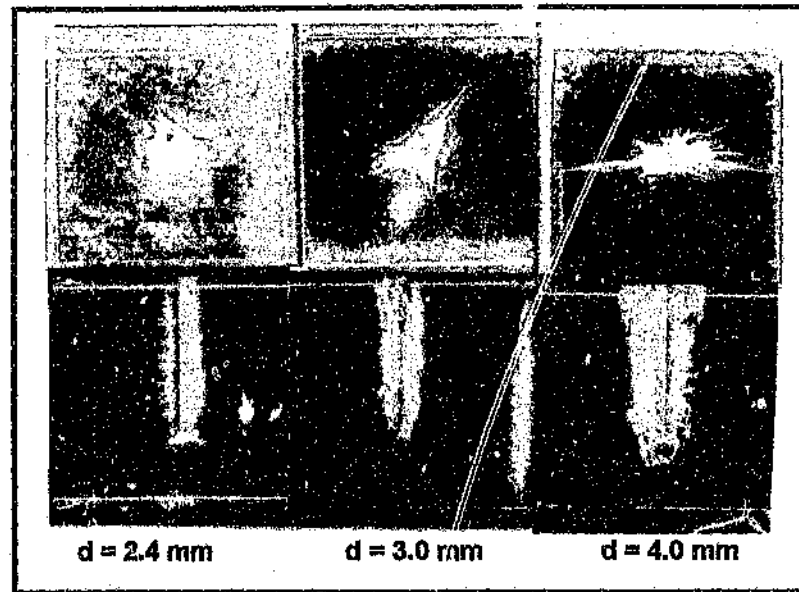


Figure 5.3 Effect of decoupling on primary crushing and fracture extension

Note: For $d = 4.5 \text{ mm}$, induced fractures decreased (see in Table 5.1)

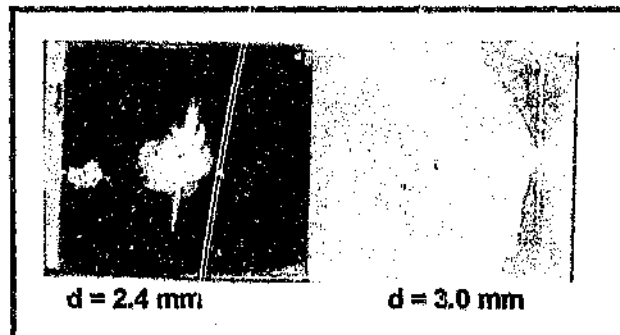


Figure 5.4 Effect of decoupling when the ratio of axial to lateral loading exceeded one

Note: Beyond $d = 30 \text{ mm}$, fractures could not be contained in the cubes.

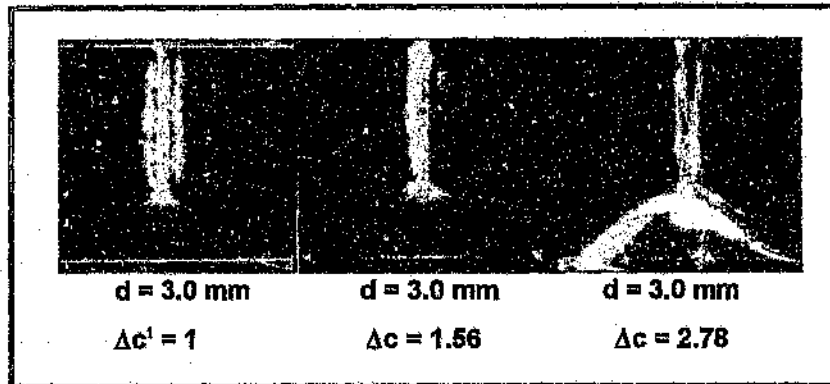


Figure 5.5 Blasthole bottom fracture

Note: In this investigation the variation of the decoupling ratio was obtained by increasing the gap between the explosive end and the bottom of the blasthole.

These results confirmed that the shock phase in blasting is responsible for determining both the density and the extent of radial fractures in the zone immediately around the blastholes. Fines are a result of the higher density of fractures occurring at the interface between the explosive and the material at the blasthole walls. They are generated through the brutal transfer of the energy of the pulse of strain waves to the resisting material during the dynamic impact which is characteristic of the detonation. The higher the energy released in the shock phase, the deeper the extent of this primary zone of comminution, resulting in more fines being generated.

The results of blasts in perspex models also showed that the phase of the expansion of gaseous products influences the final length of the induced

¹ Δc stands for decoupling ratio.

fractures (zone of fractures). This phase determines the final size of the material particles, that is the resulting fragmentation of the blast, and the conditions of the remaining excavation after the blast, or control of the overbreaking (Kabongo, 1995c).

5.2. 2 Extrapolation of these results to coal blasting

The outcome of the described investigation implies that the decoupling of the explosive charge in blastholes results in attenuated impacting shock waves pulse, which reduces the primary crushing and hence limits the generation of excessive coal fines. This means that the energy responsible for comminution in coal blasting can be regulated through altering the degree of the in-hole coupling of the explosive charge. This finding was further pursued in field investigations aimed at devising techniques for reducing fines generation of coal blasting. Another alternative proposal for controlling the generation of fines in coal blasting would be to use an explosive formulation with low shock energy. One would have to be careful not to tamper with the detonability of the explosive since this could be compromised by efforts to reduce the shock energy component. The last field of contention is the critical density of the explosive and the possibility of malfunctions. It can give rise to problems of in-hole desensitisation and misfires.

5.2. 3 Blasthole bottom fractures

Another observation made during the laboratory investigations concerned the induced fractures extending at the bottom of the blasthole. They were also observed to increase with the increasing decoupling ratio. This implies that high coupling at the bottom of the blasthole, in addition to highest energy concentration due to bottom priming, bred favourable

conditions for heavy localised crushing. However, this crushing is limited in extent since the highly compressive shock stresses generated would attenuate rapidly over a short distance from the source. Thus the reduced gas energy phase generates shorter fractures, avoiding excessive damage to coal.

When blasting the overburden to expose coal (surface coal mining) using large diameter holes, the fracturing which occur at the bottom of the blastholes may be an important source of damage to coal if adequate rock burden is not left above the seam. Also, accidental drilling through the coal seam commonly occurs. In both cases, Chiappetta (1994) recommends that some backfilling of the blastholes be done prior to charging. However, the major practical problem always encountered involves the quantification of the required backfill since the length of the hole drilled into the coal is often difficult to assess. Normally the operator stops the drill's advance only when black powdered coal cuttings appear on the surface. At that stage, the bit is already some distance into the coal.

5.2.4 Physical and mechanical properties of coal

Coal samples were collected from two pilot collieries, Greenside and Tavistock, Arthur Taylor Colliery, both mining the coal from the Witbank coalfield. Sampling was conducted respectively in the No 1, 2 and 4 Seams at Tavistock and in the No 2 and 4 Seams at Greenside. Each seam was sampled at the top, the middle and the bottom to give fairly representative specimens. They were core-drilled at Wits to 42 mm or diameter and further prepared for tests to determine both their static and dynamic strength properties.

The tests conducted in the laboratory included the uniaxial compressive strength (UCS), the velocity of propagation of ultrasonic waves through coal, and the dynamic impact test for coal crushability (DICC).

The DICC test was performed using the apparatus devised in-house at Wits shown in Figure 5.6.

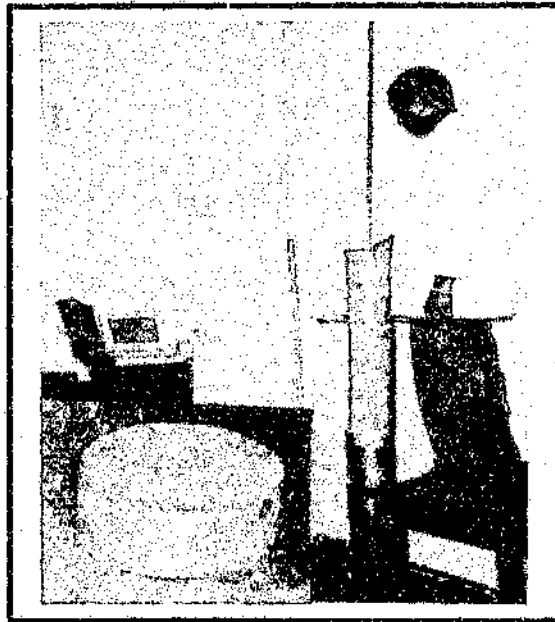


Figure 5.6 Coal crushing stamp mortar for the DICC test

5.2.4.1 Uniaxial compressive strength test

Coal samples were prepared to the standards required by the International Society of Rock Mechanics (Bieniawski, 1979) with a ratio of core length to diameter of 2.5. Where possible at least three samples were considered for each sampled horizon in a seam (i.e. top, middle and bottom). They were submitted to an increasing load until failure occurred, in order to

determine the resistance to static compression. The characteristics assessed included the density (ρ), the compressive strength (σ_c), the dynamic elasticity or Young modulus (E_d), and the Poisson's ratio (ν), the velocity of propagation of both the body (compressive component V_p) and the surface (shear component V_s). The results are summarised in Table 5.2.

Table 5.2: Physical and mechanical properties of coal

Colliery	Seam	ρ (kg/m ³)	σ_c (MPa)	E_d (GPa)	ν	V_p (m/s)	V_s (m/s)
Greenside	2	1 400	39.7	64.0	0.27	2 174	1 480
	4	1 450	12.6	66.6	0.29	2 465	1 350
ATC	1	1 400	28.7	101.7	0.20	2 736	2 000
	2	1 580	25.9	146.5	0.31	3 162	1 985
	4	1 430	15.0	56.5	0.29	2 403	1 296

Greenside Colliery mines coal from the No 2, 4 and 5 Seams. Tavistock, Arthur Taylor Colliery mines the No 2 and 4 Seams. The hardest coal tested was observed to come from the No 2 Seam at Greenside. This coal had a compressive strength of approximately 36 MPa in the direction parallel to or along the beddings, and 43.4 MPa across or perpendicular to the beddings. The Tavistock coal had a compressive strength of 27.9 MPa along the bedding planes. The coal from the No 5 Seam is the softest at Greenside. It is mechanically mined underground using continuous miners or roadheaders. Currently, the No 1 Seam is not mined at Arthur Taylor,

although it is still mined at some other Tavistock collieries. The general trend observed is that the deeper the seam, the higher the compressive strength and hence the harder the coal. However the in-situ strength situation may differ due to variations in the fracture frequency.

The strength anisotropy, defined as the ratio of the strength across to that along the beddings, is approximately 1.2 (see Appendix A). The ratio of the elastic modulus in two directions is around 1.6. This implies that the coal of this seam can be expected to transmit stresses relatively better across rather than along beddings. More breakage in the plasto-elastic zone around the hole can thus be expected in this favourable direction due to the stress effect being more effective, rather than in the opposite direction where additional energy venting through discontinuities leads to rapid attenuation of pulses (along beddings). This effect was clearly apparent in single-hole blasts performed in layered material. The experiment highlighted the effect of the spacing of bedding planes on the blast results (Kutter and Kulozik, 1990). Closer spacing of beddings hampers the propagation of blast energy; the blast effects are confined to the blocks around the blasthole, resulting in a poor network of fractures that do not penetrate into the coal.

The coal of Seam 4 can be considered as soft. The average compressive strength is 12.5 MPa at Greenside and 13.5 at Tavistock along the bedding planes 12.6 MPa at Greenside and 16.6 MPa at Tavistock across the bedding planes. The beds are hardly distinguishable, as calcite appears to fill all the cleats. The strength anisotropy in coal from Greenside is around 1.1, indicating that even breakage around blastholes can be expected. Tavistock coal presents a higher anisotropy ratio of 1.3. This pointed towards an uneven distribution of breakage around holes.

5.2.4.2 Dynamic Impact test for coal crushability (DICC)²

The test is aimed at assessing the behaviour of coal subjected to dynamic loading. It assumes that the coal strength and the explosive energy that is input for breaking are vital parameters controlling the breakage and the generation of coal fines. The relationship linking these parameters is therefore the key to the optimisation of the blast effects and final outputs, in particular the fragmentation.

Early research used the static failure recorded from models gently loaded in a conventional UCS test as criterion in attempts to predict the blast results and blastability of rock materials (Fogelson, Atchison, and Divall (1959), Allsman (1960), Kumao (1961), Langefors (1963), etc.). This thesis supports the work done by Kou and Rastan (1992) which considered that since the release of the breaking energy into material during blasting is a dynamic process, it is advisable therefore to use dynamic parameters of failure. The DICC test uses on the one hand, the measurement of the velocity of propagation of ultrasonic waves in coal that is recorded using the equipment shown in Figure 5.7. On the other hand, it uses the data of size distribution of particles produced by the free fall of the hammer mass onto coal samples in the stamp mortar of Figure 5.6. Data measured are used to determine an energy/breakage relationship called the "appearance function" which is characteristic of the coal material. The function of energy/breakage obtained may be used to predict the comminution (primary crushing) in coal in-situ, and the subsequent fines generation

Dynamic elastic properties of coal

The dynamic elastic properties were determined by passing ultrasonic

² Crushability is defined as the ability of a coal material to generate fines.

waves through the coal samples and monitoring the propagation time. The time data were used to compute the velocity of propagation of the body or p-waves (V_p) and the surface or s-waves (V_s). These were used to determine the dynamic elasticity modulus (E_d) and the Poisson's ratio (ν) using the equations (Rzhevski and Navik, 1971):

$$E_d = V_p^2 \rho \frac{(1+\nu)(1-2\nu)}{(1-\nu)} \quad 5.1$$

$$\nu = \frac{V_p^2 - 2V_s^2}{2(V_p^2 - V_s^2)} \quad 5.2$$

where ρ is the density of the coal.

In the present investigations, CNS transducers were used with a Pundit p & s unit coupled with a Tektronix double-beam oscilloscope of type 565. The equipment is shown in Figure 5.7 and the data recorded are presented in Table 5.2.

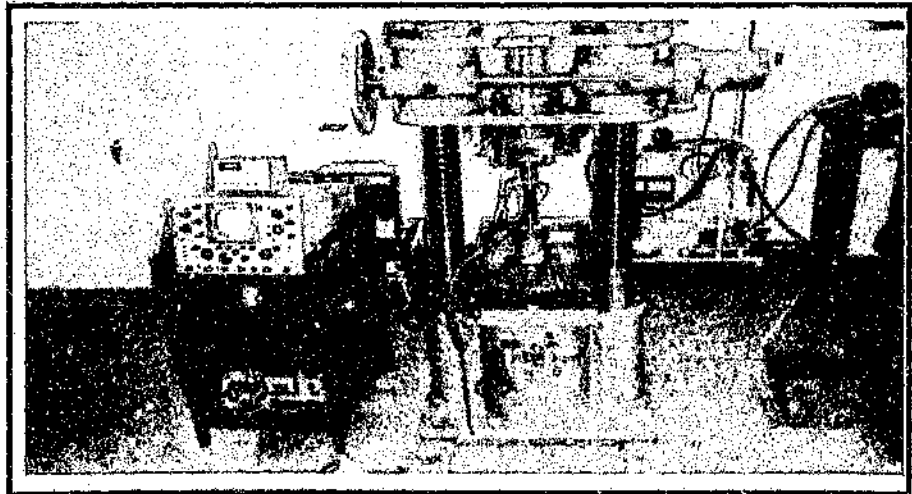


Figure 5.7 CNS/Pundit equipment used to measure the velocity of propagation of body and surface waves in coal samples

Evaluation of the efficiency of energy transfer

Like several other industrial comminution processes, blasting is inefficient. Wills (1988) stated that only one per cent of the total energy consumed in conventional grinding is used in actually creating the new surfaces. Nielson (1995) quotes Morrel et al. (1992) who have asserted that industrial crushers can effectively use only up to 20 per cent of their energy input. Brinkman (1987) showed that even if the use of explosives results in 20 per cent of the explosive energy being transmitted to the material as elastic strain energy, part of it will still be consumed in the generation of micro-cracks which do not contribute to the apparent breakage. The importance of the comminution energy efficiency in any attempt at defining the appearance function of any material is thus underlined.

In this thesis the efficiency of comminution energy is measured through a parameter called 'comminution energy efficiency (CEE)'. This parameter is defined as the ratio of the input energy to that effectively consumed in coal breaking termed 'comminution energy (ComE)'. The energy input may be the kinetic energy that is accumulated in the hammer at the impact (laboratory conditions), or the chemical explosive energy in a real blast.

The comminution energy was determined from the strain-history signals recorded on loaded coal samples using dynamic strain gauges. Two foil gauges of 20 mm length and 120 ohms resistance were carefully bonded to each sample in a half-Wheatstone bridge. The gauges were connected so as to feed information to a sensitive recording system consisting of a double-channel amplifier and a high-speed wave recorder, triggered automatically by the strain level in the coal. The signals recorded (see a

typical sample in Figure 5.8) were downloaded into a 486 laptop PC for analysis, using the High Speed Data Recording (HSDR) and the Digital Signal Processing (DSP) software³.

Comminution energy

The basic hypothesis of the test is that the behaviour of coal material under load is elastic (Hooke material) within some limits of load and deformation, depending on the coal. The strain is then proportional to the stresses developed. Beyond this elastic domain, the comminution law applies, i.e. the breakage observed in coal is proportional to the energy consumed in its generation.

The strain signal recorded during the dynamic loading of the coal sample, e.g. Figure 5.8 embodies significant information pertaining to the coal breaking history. The first wavelength of the recorded strain signal is characteristic of the impulse that generates the majority of the breaking energy. The DSP software used enables the researcher to zoom-in on this wavelength and to single out the part of the signal corresponding to the compressive pulse that generates the comminution (i.e. Figure 5.9). The integration of this part of the signal yields a comminution energy parameter that is termed in this thesis "comminution energy, (ComE)".

The comminution energy, ComE is calculated using equation 4.7 in which the integration limits are restricted to the compressive portion of the signal (t_0 to t_1) as shown in Figure 5.9.

$$ComE = AE_s V \int_{t_0}^{t_1} \sigma^2(t) dt$$

³ HSDR/DSP are trade marks of TLC Software Limited.

A is the cross-sectional area over which the breaking energy is transmitted, in m^2 , E_d is the dynamic modulus of elasticity, in kg/m^2 , V is the velocity of the impacting hammer weight on the sample, in m/s , δ is the strain developed in the sample as a result of the impact or $\delta = \Delta/L$ with Δ being the change in length of the sample in mm and L the original length in m), and dt is the impact elementary time in seconds. The integration limits correspond to the time of coal loading in compression t_0 to t_1 .

This thesis does not bring into the discussion the consideration of elastic energy that may be stored in the coal sample during the process of core drilling because of the careful core drilling methodology employed. Coal samples were drilled using sharp bits, with the rotary coring speed and thrust set so as to produce minimal compression of the coal. Furthermore, the strain recording obtained from a specimen on which sensors were carefully bonded takes into account the global effect of displacements resulting from the impacting and the internal energy.

Coal sizing and fines definition

The fragments of coal collected from the bottom of the mortar were sieved using a vibrating laboratory shaker. Eight sieving stages were used as shown in Figure 5.10. The standard sieving time was fixed at ten minutes. Thereafter the fractions retained in each sieve were weighed and the data fed into the database, a summary of which is presented in Appendix B. The two bottom fractions of -180 and -500 microns are called respectively ultra-fines (U-F) and fines (F). They are used as the basis for comparison of the comminution results of the investigation. However, it is worth noting that in

the local coal mining industry, the cut-off size conventionally used for coal fines is 6 mm (Thorpe and Pedroncelli, 1986).

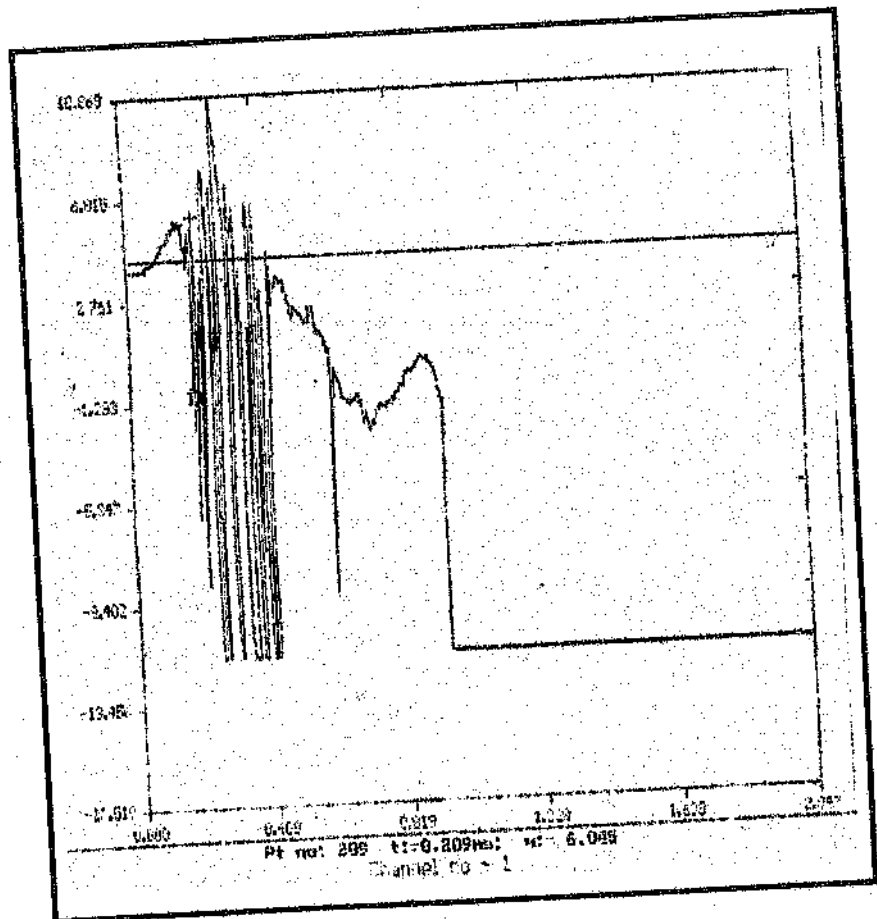


Figure 5.8 Typical strain recording obtained from a DIC test

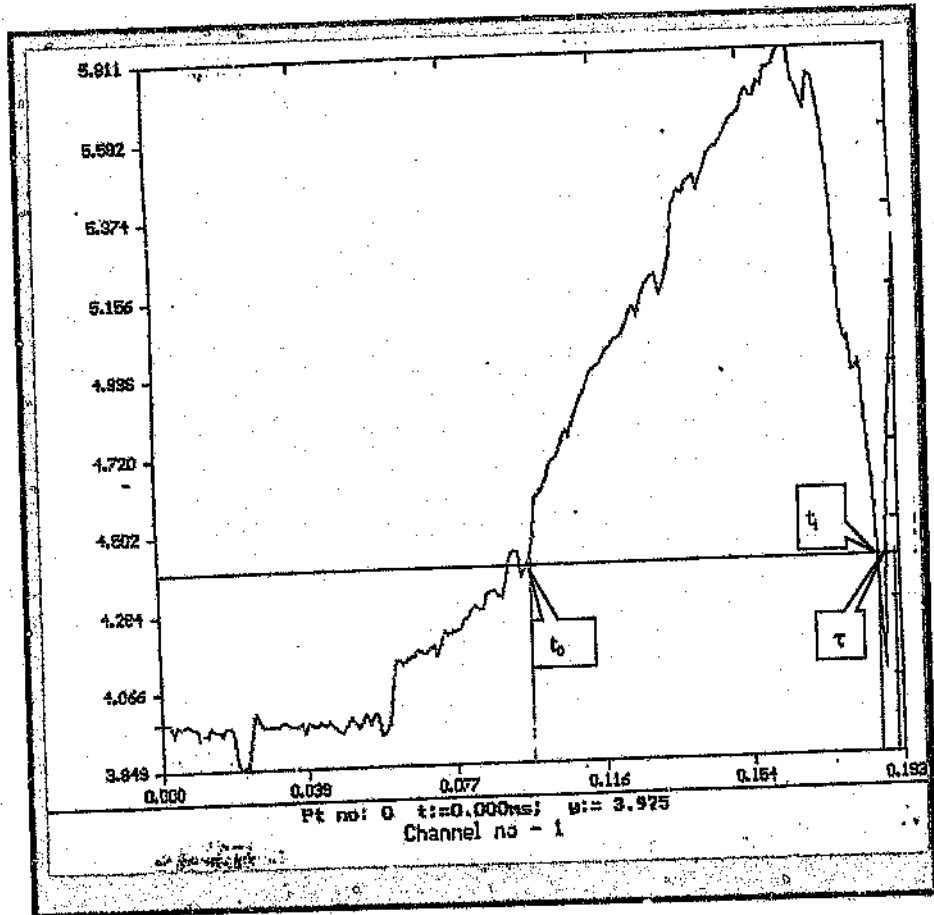


Figure 5.9 Zooming into the strain wave pulse for calculation of the coal breaking energy

In the above illustrative example (Figure 5.9) the time of occurrence of the different loading events are $t_0 = 0.100$ Ms, $t_i = 0.185$ Ms and $\tau = 0.189$ Ms. This means that the time of the compressive loading associated with ComeE was $t_i - t_0 = 0.185 - 0.100 = 0.085$ Ms = 85 μ s.

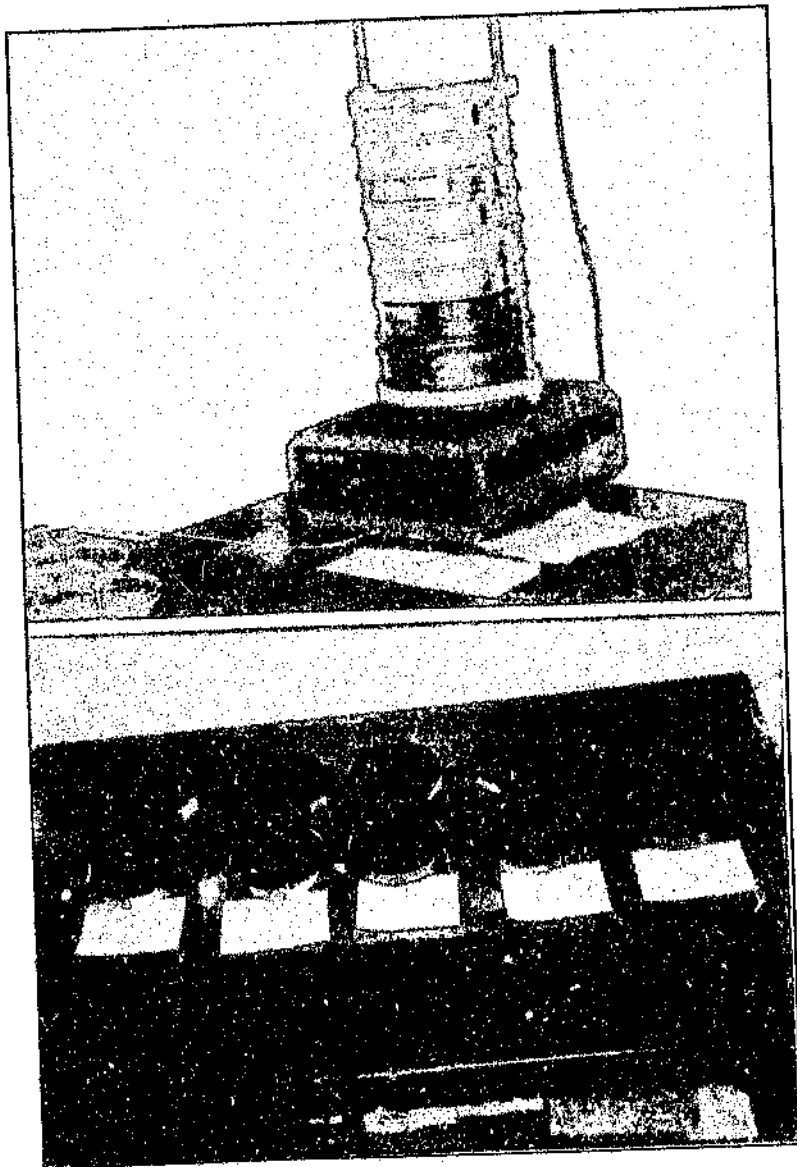


Figure 5.10 Classification of coal fragments for the DICC test

Comminution index (ComI)

A comminution operation always involves three variables, namely the energy, the mass and the resulting size reduction (fragmentation). The parameter used in this project to compare the comminution behaviour of coal is called the 'comminution index' (ComI). It is defined as the ratio of the specific comminution energy, called the 'comminution factor' (ComF), to the percentage of fines generated by the material being comminuted.

Similarly to the powder factor which is the ratio of the explosive mass to the volume or mass of rock broken, the comminution factor is a comparative parameter which indicates the specific comminution energy.

$$\text{ComF} = \text{ComE} / M \quad [\text{J/kg}] \quad 5.4$$

where M is the mass of coal being comminuted.

The specific comminution energy required for generating one percent of fines in a given coal material is then called the comminution index (ComI). It is a characteristic of the coal material and the comminution energy efficiency (CEE) of the process considered.

$$\text{ComI} = \text{ComE} / M / \% \text{ fines} \quad [\text{J/kg}/\% \text{ fines}] \quad 5.5$$

The dynamic crushing of a representative sample of any particular coal can enable one, using this concept, to determine a model of prediction of the required input energy to generate a given level of fines. This may be the required height of fall in the DICC test or the powder factor in an in-situ blast. That is what the laboratory investigations of this thesis were aimed at illustrating.

Laboratory results of coal comminution

Eleven sets of approximately ten coal samples were tested. The comminution data collected are given in Appendix B, but a summary of the results is presented in Table 5.3 below. A 20 kg hammer mass was used to impact coal samples at the bottom of the mortar, thus generating measurable breaking energy. The impact loading time and the efficiency of the energy transfers were determined (see Appendix B). However, the calculated average specific comminution energy per set of coal samples tested (ComF) is given in Table 5.3.

Table 5.3 Compared physical and comminution properties of coal

SAMPLE ⁴	ρ (kg/m ³)	E_d (GPa)	ENERGY INPUT (J)	CEE (%)	ComF (J/kg)	Fines (%)	ComI (J/kg/%fines)
GREENSIDE							
G2B#	1 373	82.5	184	21.5	186	2.03	91.7
G2B//	1 423	75.4	200	30.0	328	2.23	147.3
G4B#	1 443	56.2	190	14.9	121	3.16	38.2
G4B//	1 440	58.3	193	7.0	60	3.30	18.2
G4M#	1 475	72.5	173	17.9	172	4.03	42.7
G4M//	1 440	63.7	176	28.4	126	3.43	36.7
TAVISTOCK							
T1B//	1 419	101	177	22.9	209	2.60	80.4
T1T//	1 384	81.8	216	22.9	221	3.24	68.3
T2M//	1 837	246.4	194	51.4	335	1.60	210.5
T4TM//	1 382	60.3	176	18.8	171	2.62	65.27
T4M#	1 480	60.1	180	20	180	3.28	54.8

⁴ Legend used for sample identification: Mine (Greenside or Tavistock) - Seam No (1, 2, or 4) - Sample location in the coal seam (Top, Middle or Bottom) - Drilling direction (// parallel or # perpendicular to the bedding planes).

The relationships between the coal characteristic physical strength properties and the comminution parameters (comminution energy efficiency (CEE), percentage of coal fines generated, comminution factor (ComE) and the comminution index (ComI)) are graphically represented in Figures 5.11 and 5.12.

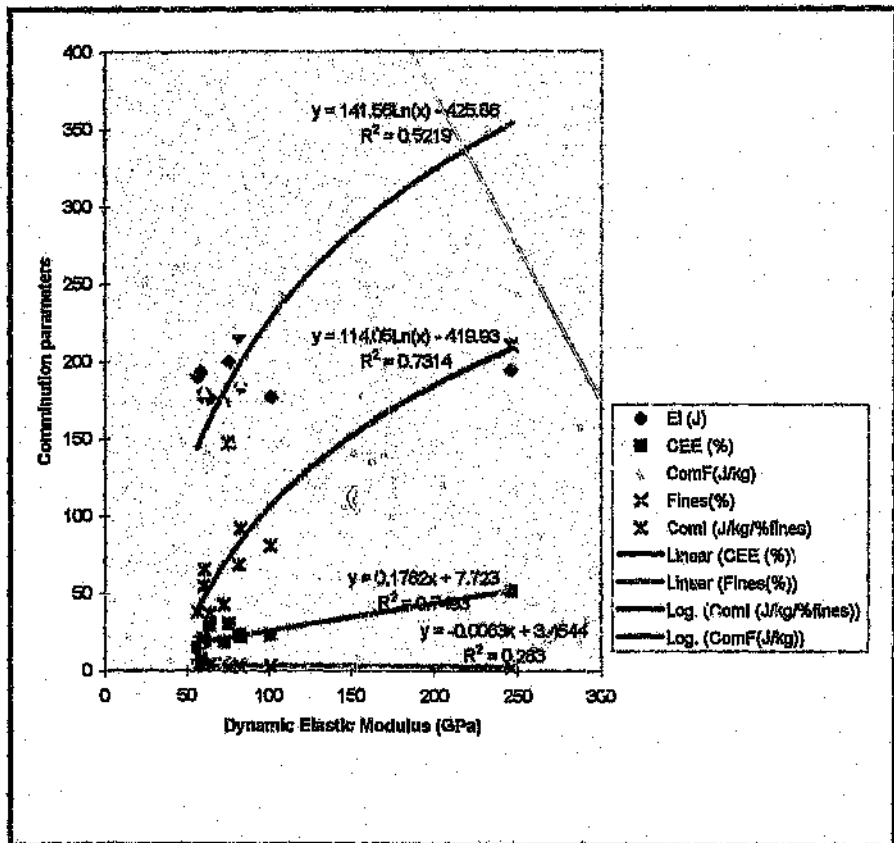


Figure 5.11 Physical vs comminution strength properties of coal

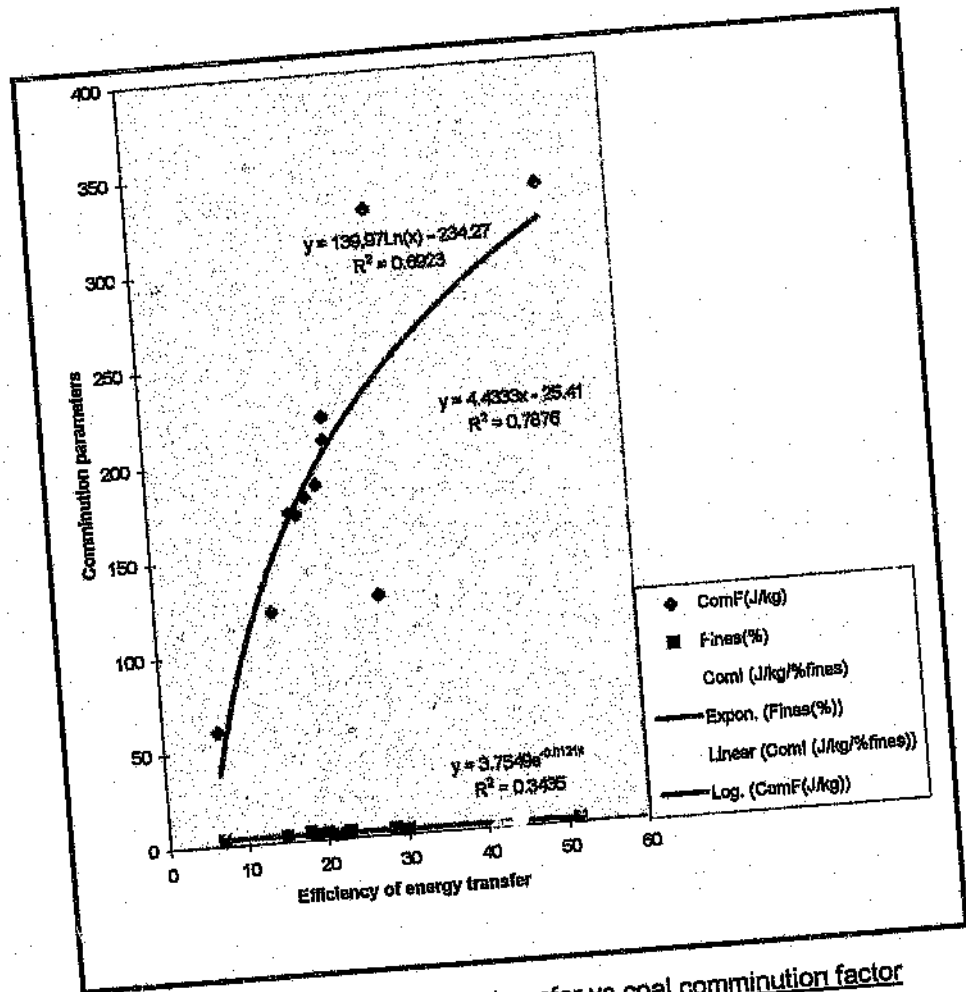


Figure 5.12 Efficiency of energy transfer vs coal comminution factor

It can be inferred from the Figure 5.11 that a poor linear dependence exists between the generation of fines and the strength of coal. However, the trend observed is that the harder the coal, the less fines it generates.

It is also learned that the comminution factor (ComF) is fairly well related to the physical strength properties of coal. The regression coefficient

observed is 52%. This is well below the minimum acceptable level of 58% at a 95 % confidence threshold.

The comminution Index (ComI) is well related to the coal strength. The observed correlation coefficient is 73%. This shows that ComI may be predicted from the strength characteristics of coal with a level of confidence of 99 %. This result confirms that ComI is a reliable parameter that is characteristic of the comminution properties of coals.

The comminution energy transfer (CEE) appears to be well related to the coal strength. A regression coefficient of 74% shows that the harder the coal the more the input energy effectively consumed in the breaking process.

Figure 5.12 depicts a trend of decreasing fines generation with the increasing energy transfer. This is to say that the higher the energy transferred into coal the better the fragmentation, i.e. the lower fines generated. Subsequently hard coals, which according to the result plotted in Figure 5.11 showed to oppose little resistance to the propagation of strain waves, may be expected to generate lesser fines than the soft coals.

In field blasting, the efficiency of energy transfer depends on the in-hole performance of the explosive charge and its confinement in the blasthole. The confinement is understood to have two components, namely radial confinement controlled by the coupling ratio and the coal strength and the longitudinal confinement, which is controlled by the stemming. These parameters play a pivotal role in setting conditions for the final blast results. The longitudinal confinement controls the time of retention of the explosion gaseous products in the coal. It thus has a direct influence on

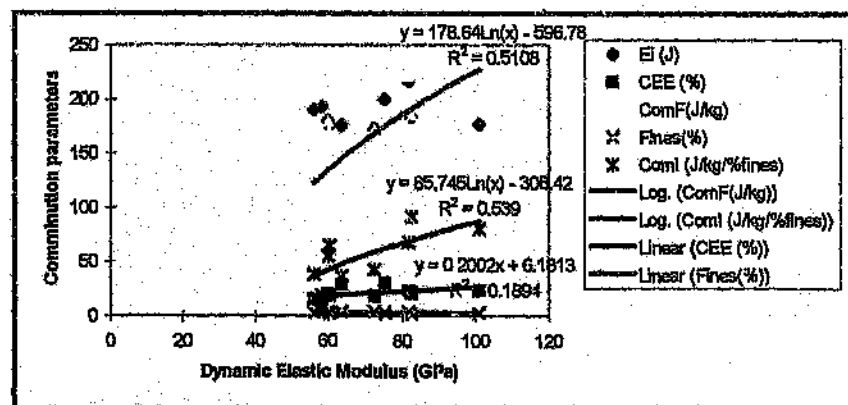
the coal fragmentation. The radial confinement controls the extent of the crushing zone through the coupling ratio. It is therefore around this aspect that the present project devised innovative ideas put forward in this thesis as techniques to reduce coal fines in blasting (optimization of coal breaking).

Discussion of the laboratory comminution results

The results in Figures 5.11 and 5.12 appear to bear a great influence of the sample T2M// whose strength characteristics may be seen as questionable. The considerably high variance of the characteristics of this sample may be attributed to a high degree of contamination of coal of this particular block by some heavy rock material (Intrusions). However, in view of its critical influence on the overall results, it is found worth to evaluate the sensitivity of the model by disregarding. The following change in results are therefore observed:

1. The regression curves of Figure 5.11 and Figure 5.12 are modified as follows:

a)



b)

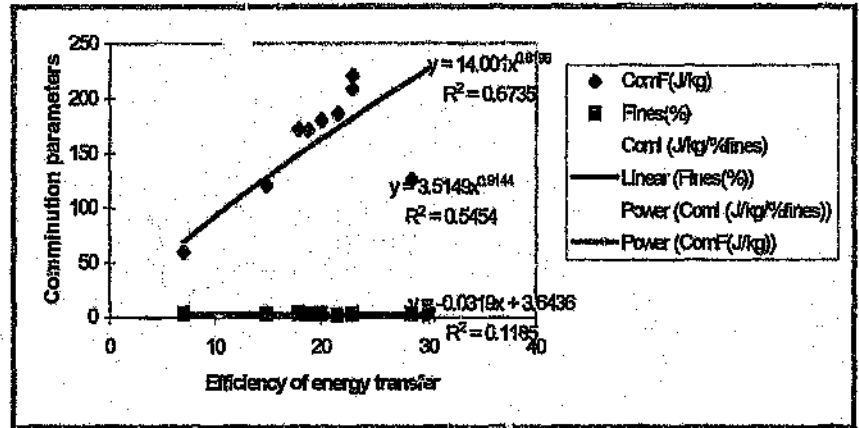


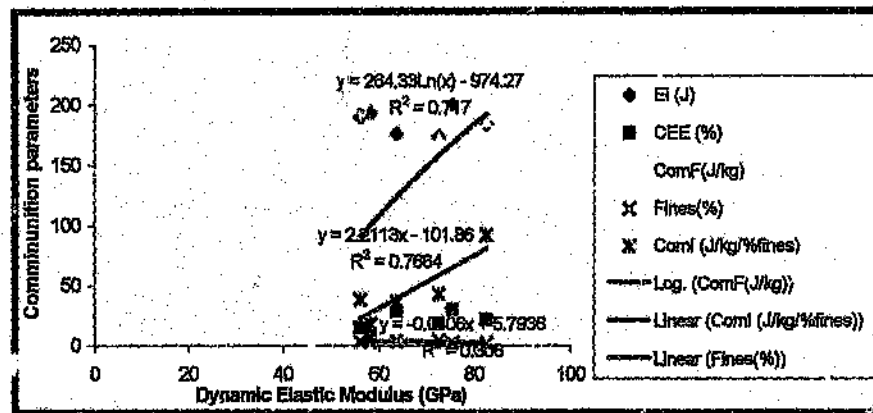
Figure 5.13 Sensitivity analysis of the results obtained (global ... data)

The change observed calls for the following comments:

- The relation between the comminution parameters of this thesis (ComF and Coml) and the Elastic Modulus of coal has not changed (the best-fit function remains a logarithm function). However the confidence level has dropped from 52% to 51% for ComF and from 73% to 53% for Coml.
- A slight change is observed in the relationship between the CEE and the comminution parameters. Coml and ComF are still reasonably well related to the coefficient of energy transfer but the best-fit functions have rather changed to power functions with a respective drop in confidence level of 2% for ComF and approximately 24% for the Coml.

2. The sensitivity analysis was conducted further to probe the validity of the relationship between the defined comminution indexes and the coal strength at the individual mines. Combined coal strength data from the two mines were analyzed individually as shown in figure 5.14-a and -b.

a) Greenside coal



b) Tavistock ATC coal

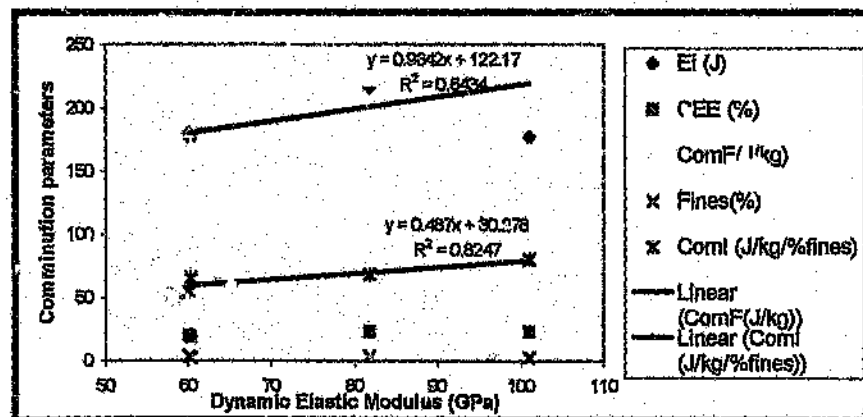


Figure 5.14 Sensitivity analysis of the results obtained (coal data considered per individual mine)

The following additional comments can be made:

- In overall, the comminution parameters defined in this thesis show an acceptable degree of correlation for industrial standards bearing in mind that assumptions made in the conceptual stage of the test are not 100% real. In addition, the strength characterization of coal is particularly problematic due to generic peculiarities of this rock type (the nature of deposition and the natural fractures imbedded).
- It may be seen that the confidence level of the relation between the comminution parameters (ComF and ComI) and the coal strength observed in this research is much more improved when coals are considered according to their original location rather than when they are collectively considered.

5.2.4.3 Modelling of comminution (coal crushability functions)

The crushability curves were plotted to illustrate the relationship between the breaking energy and the generation of coal fines. Coal fragments collected after impact were sieved and the behaviour of the two bottom fractions, termed 'fines' for all the under - 0.5 mm (<500 microns) and 'ultra-fines' for the under - 0.18 mm (< 180 microns) portions, was tested. The size of these fractions is related to the energy properties observed in order to determine the characteristic appearance function of the coal material. The observed results are presented in a series of curves shown in Appendix C. These were found to best fit a theoretical exponential function of the general form shown in Figure 5.15 whose characteristic equation is:

$$\text{ComF [J/kg]} = B e^{[A(\% \text{ Fines})]} \quad 5.6$$

where A and B are coefficients taking into account the breaking conditions.

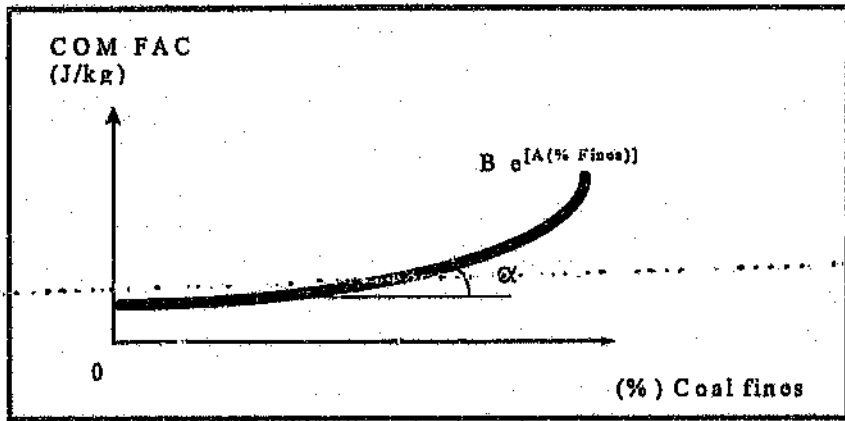


Figure 5.15 General form of the comminution function of coal

From the general form of the comminution function observed, the comminution Index can be deduced as follows:

$$\text{ComI} = \tan \alpha \approx \text{ComF} / \% \text{ fines} \quad [\text{J/kg}/\% \text{ fines}] \quad 5.6$$

This index is further used to compare blasting techniques with regard to the coal fines generated.

5.3 Conclusion

The laboratory investigation has shown that a reasonable good correlation exist between the coal strength properties (dynamic elastic modulus), the energy required to generate fines (comminution energy) and the portion of the input energy consumed in coal breaking process (comminution energy transfer).

6 INVESTIGATION OF THE GENERATION OF FINES IN SURFACE

BLASTING

6.1 Introduction

The model introduced in Chapter 3 asserted that the secondary crushing mechanism is responsible for an important portion of the fines observed in coal piles. The generation of these fines is attributable to the non-homogeneity and anisotropy in coal material, exacerbated by its structure and in-situ strength. The in-situ strength encompasses the effects of the innate planes of weakness, which exert a strong, if not dominant, influence on the results obtained in blasting. The major fracture types in coal are joints, cleats, cross-cleats and bedding planes. In this thesis, fractures refer to all pre-existing planes or surfaces of weakness, whether open, closed, or filled, that divide a mass of coal into segments, each of which is more or less intact. Breakage can easily occur along any of these planes.

This chapter presents the results of the site investigation undertaken to assess the modelled hypothesis given in the previous chapter in order to prove the assertion that generated fines observed in coal piles after blasts are related to the inherent fractures imbedded in the coal mass. The work was conducted at Kleinkopje Colliery, a surface mining operation of the AMCOAL group. The results obtained confirmed the hypothesis as shown in the discussion below.

6.2 Breaking strength of coal

In terms of its geological structure, coal is a sedimentary and stratified rock material made of a number of naturally bound layers (discontinuity planes). It is therefore seldom that the elastic behaviour of laboratory samples, whether cores or bulk samples, is the same as that of the in-situ coal in the

seam. However, over the years, it has been possible to evaluate the strength and blasting behaviour of rock material and coal using standard tests such as the uniaxial compressive (UCS) and tensile strength (UTS) tests. These indices are also used in the classification of a coal's potential for fines generation since they correlate well with the relevant parameters such as porosity, weathering and fissuring (Evans and Pomeroy, 1966). A rule of thumb commonly used is that, for similar mineralogy, the UCS decreases with increasing porosity, increasing degree of weathering and increasing degree of micro-fissuring (Brady and Brown, 1985). Vutukuri, Lama and Saluja, 1974) added that the UCS may also decrease with increasing water content. The tensile strength of rock is generally low and coal is even worse in this regard as its innate fractures result in even less ability to transmit tensile stresses. Although the tensile behaviour of rock is not directly related to fines generation, the walls of fractures (lips) may act as partial reflecting surfaces of stresses, attenuating their propagating energy in an impact and resulting in some fines being generated. In addition, the differential loading of a fracture may generate some shearing at the tips, as illustrated in the model of Chapter 3.

The breaking forces and subsequent damage that constitute an optimum blast in coal depend greatly on the innate weaknesses present (bedding planes and cleats). The cleats are generally meso-cracks at right angles or slightly inclined to the stratification, which appear as main divisions (primary cleats) or secondary divisions (cross-cleats) in coal material. In general, it is observed in South African coalfields that more and better-developed cleats are found in bright coals and that they are widely spaced or even not at all apparent in dull coals. Another natural property of coal is known as slip. This is a crack quite distinct from cleating as it traverses the full thickness of the coal seam and is usually inclined at about 25 to 40

degrees to the vertical. In general, a well-developed slip is advantageous to the extraction of coal whether done by hand, by mechanical means or by explosives but causes major safety problems as it makes the seam (and the roof) inherently unstable.

The orientation and frequency of these natural features (discontinuities) are decisive factors in planning the direction in which coal faces should be blasted. The assessment of fractures is necessary to predict the size of fragments after blasting, as the fragmentation resulting from a blast is very much dependent on the inherent distribution of microfissures, joints, bedding planes, fissures and faults (Maynard, 1990). Lilly (1986) and Cunningham (1987) did acknowledge the merit of including a "discontinuity index" in their respective models, the model of rock blastability and the KuzRam model of prediction of blasting fragmentation. These efforts at modelling blast results unfortunately took only an overall view of fragmentation, the aim being to predict all the oversize and undersize particles generated. There has been no effort to determine the specific influence of the inherent fractures on the generation of fines and this is the focus of the current investigation.

6.2.1 Fracture frequency per metre (FF/m)

This is a simple method of quantifying discontinuities in coal material. The assessment of the parameter consists in counting all the fractures intersected by a sampling line drawn on a core sample, any piece of the extracted coal that has a plane surface, or an in-situ free face of the coal-mass (face of a trench). The average number of fractures counted per running metre of the coal sample is referred to as the fracture frequency per metre (FF/m).

6.2.2 Rock quality designation (RQD)

Rock engineers make extensive use of the concept of RQD introduced by Deere and Miller (1966) to quantify the fractures in a rock mass. The parameter is determined from a core-recovery technique that makes use of all pieces of core sample drilled from the coal seam that are longer or equal to 100 mm. The sum of such lengths when expressed as a percentage of the total length of the drilled core is the RQD index:

$$RQD(\%) = 100 \frac{\sum \text{core lengths} > 100\text{mm}}{\text{Total length}} \quad 6.1$$

The RQD index was found to correlate well with the spacing of discontinuities in hard rock. Priest and Hudson (1966) reported a good correlation, empirically identified, to relate the RQD as an exponential function of the fracture frequency as follows:

$$RQD = 100 e^{-0.11\lambda} (0.11\lambda + 1) \quad 6.2$$

where λ is the mean discontinuity frequency.

6.2.3 Coal grading

Coal is a combustible sedimentary rock believed to originate from the accumulation and deposition of vegetal organic matters during the course of geological history (Evans and Pomeroy, 1973). South African coal deposits are banded with different textures and brightness. The bands run parallel to the seams. Two types of coal are distinguished on the basis of their brightness, namely "bright" and "dull" coal. Mc Carthy (1981) presented a classification of coal according to the grade of bright elements. A summary of this classification is presented in Table 6.1

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below. Mc Carthy ascertained that the general pattern observed in South African coal is as follows:

1. The brighter the coal, the higher the volatile matter and the higher the ranking.
2. The brighter the coal, the higher the fractures and the less suited to blasting it is.
3. The coal ranking, hence fracturing increases with the depth in a seam.

Table 6.1 Coal classification according to brightness (Mc Carthy, 1981)

Designation	Symbols	Composition
Bright	CBB	No dull
Bright with dull bands	CBD	Less than 50% dull
Bright and dull interbanded	CII	50% bright/dull
Dull with bright bands	CDB	Less than 50% bright
Dull	CDD	No bright

6.3 On-site investigation of the effects of fractures on coal fines

6.3.1 Objectives

In order to study the effects of the pre-existing fractures on coal fines generation, a programme of fieldwork was undertaken at Kleinkopje Colliery. The mine is situated at about 13 kilometres south of Witbank and approximately 130 kilometres east of Johannesburg.

6.3.2 Geology and mining at Kleinkopje Colliery

6.3.2.1 Reserves and stratigraphy

The coal reserves of the colliery are part of the Springs-Witbank coalfield, which is the most important one in the country. Of the five seams of the Witbank coalfield, numbered from the bottom up as 1, 2, 3, 4 and 5, only the No 1 and 2 seams are fully developed and mineable. The No 4 seam is also present and it is mined in some pits. The thickness of the seams varies between 2 and 8 m. They consist of various combinations of interbanded bright and dull coal. The No 2 seam is the largest. It is subdivided into a number of distinct coal-quality zones according to the amount of vitrinite (bright coal) present. The bottom part of the seam (approximately 0.5 m thick) contains high-quality coal which is selectively mined ("select") to produce both low-ash metallurgical (LAC) and steam coal. The upper, poor-quality coal is mined as "non-select" to produce steam coal. In areas of very poor quality, the top part of the coal seam is mined as raw coal product to be stockpiled as a future resource.

The No 1 seam has an average thickness of 2.2 m. It consists of high-quality coal which is mined as "non-select" to produce steam coal. The seam contains mainly lustrous to dull lustrous coal, separated from the No 2 seam by an overlying band of sandstone 1 to 3 m thick. Both seams are

overlaid by approximately 35 m of overburden rock, which is drilled, blasted and removed to expose the coal.

6.3.2.2 Mining

The opencast operation mines three pits (2A, 3A and 5W). The coal reserves in each pit are subdivided into a number of production blocks. The strip or cut mining method is used in which cuts 50 m wide are opened and then excavated to expose the top of the No 2 seam. The overburden is drilled, blasted and sidecast by draglines. The sandstone layer between the seams is selectively blasted off to allow coal extraction. The coal is loaded by rope shovels and front-end loaders into bottom-dump trucks which haul the coal to the washing plant or to the temporary dumps.

Overburden drilling and blasting

The mine has four drills allocated to overburden removal (three Bicyrus Erie 61R and one Bicyrus Erie 49R) equipped to drill holes 311 mm (12 1/4 inch) in diameter. The drilling pattern varies according to the thickness and hardness of the sandstone. The explosive used is ANFO augured into dry blastholes. In areas where major water ingress is experienced mainly during the rainy season, HANFO is used. Apparently, considerable efforts are made to control fragmentation and enhance the dragline productivity. Modern initiation systems using both bottom and surface millisecond delays associated with the shock tubes technology is used to ensure better blast results. Explosive charge in blastholes is initiated from bottom and various tie-in techniques are used according to the blast conditions considered. Staggered single chevron (V1) and double chevron (V2) shown in Figure 6.1a,b, and c are currently preferred.

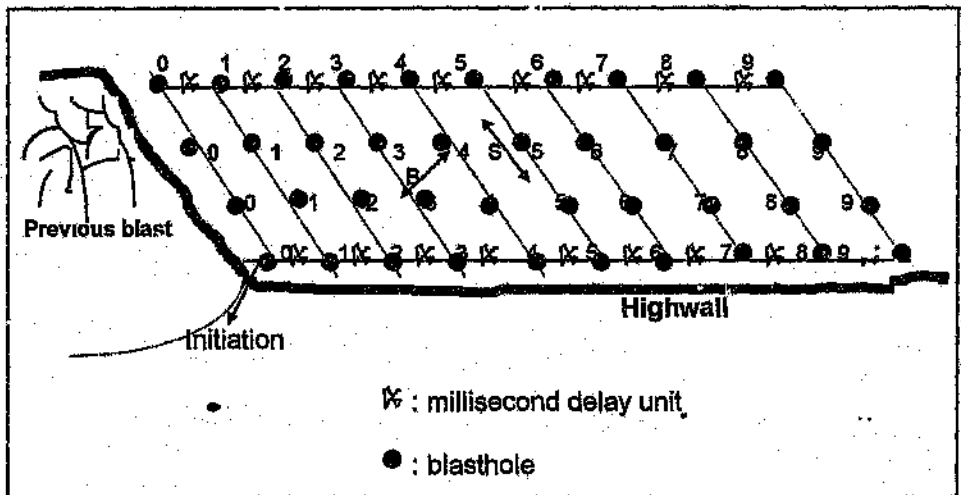


Figure 6.1a Plan view of a typical blasting pattern used at Kleinkopje

Illustration of a V1 tie-in using row by row initiation.

(Apparent burden to spacing ratio of approximately 1:1)

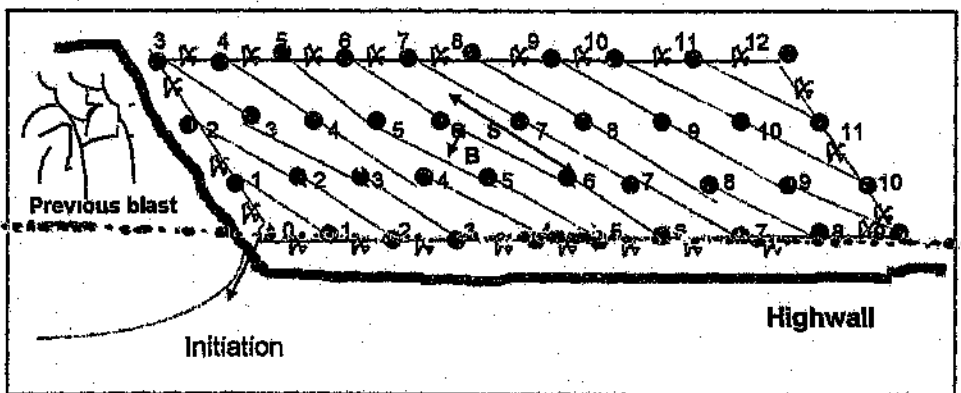


Figure 6.1b Plan view of a typical blasting pattern used at Kleinkopje

Illustration of a V2 tie-in using row by row initiation

(Apparent burden to spacing ratio of approximately 1:3.5)

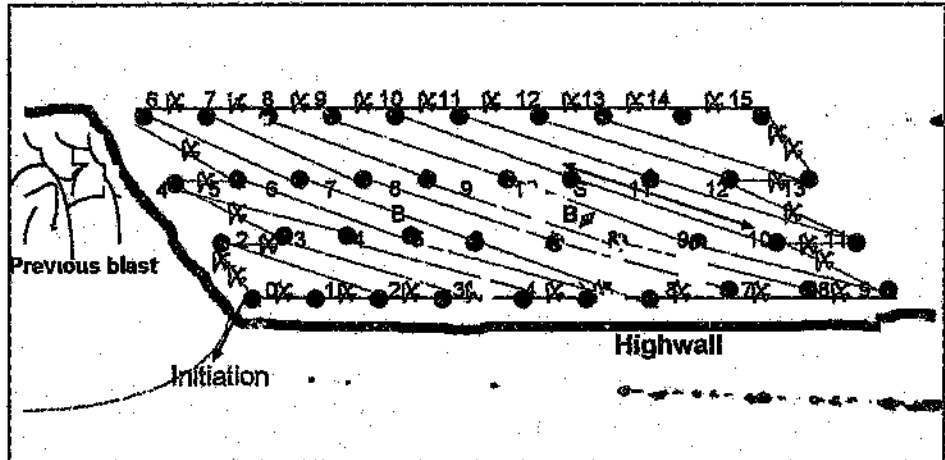


Figure 6.1c Plan view of a typical blasting pattern used at Kleinkopje
Illustration of a V3 tie-in using row by row initiation
(Apparent burden to spacing of approximately: 1:7.25)

Drilling and blasting in coal

The mine operates two Drilltec C25 KL rigs equipped with the normal three-wing drag bits to drill 130-mm-diameter blastholes in coal. The drilling pattern is often square either 4 x 4 m or 3 x 3 m. The fragmentation obtained is obviously a problem, one that this thesis seeks to address. Oversize do not pose a problem. Fines generated are variable, however, an average of 30% of the blasted coal ending up in the marginal category of coal fines (< 6mm). Since the mine's operations are designed to produce coal for export, there is therefore great deal of interest in optimising the blast output by reducing the coal generation.

6.3.3 Methodology of the investigations

The investigation was aimed at establishing a relationship, if there was one, between the pre-existing fractures and the coal fines generated in blasting. Prior to the blasting work, a number of boreholes were drilled in a virgin coal far away from the actual blasting sites. These boreholes were part of the routine programme of coal reconnaissance but the diamond-drilled samples of coal extracted were used to serve the purposes of this project as well. Diamond drilling was organised in all three of the colliery's pits and samples were then taken. The inherent fractures in the coal in the three pits were assessed through examining the core samples collected. The two fracture-definition parameters were utilised, namely the RQD and the fracture frequency per metre (FF/m). From a comparison of the evaluations, a relationship between these fracture parameters was established. This compared well with the previous findings published by Priest and Hudson (1976). Later, blasting work was organised at the sampled sites. It was closely monitored in all three pits and the results are presented and discussed in this chapter.

6.3.3.1 Data collection for the research

Data on discontinuities (fractures in coal) were collected according to a strict *modus operandi* to ensure sound and valid analysis. Drilled coal samples were collected from the diamond-drilled boreholes that were part of detailed prospecting in unmined horizons in the pits. Samples were examined for fracture logging.

Logging of fractures

A total of 75 boreholes were drilled. Cores of 75 mm diameter were recovered. They were logged to assess natural fracturing in the different

coal seams. Of the 75 drilled cores, 24 came from Pit 2A, 29 from Pit 3A and 22 from Pit 5W. The boreholes were drilled in a square mesh of 100 metres from each other to cover a distinct prospection block. Cores were examined immediately after they had been extracted from the barrel. This ensured minimum effects from external factors such as weathering, drying and eventual over-fracturing by desiccation. This also made it possible to distinguish the fractures freshly induced by the drilling operation from the original in-born fractures (In most cases filled with calcite).

Geotechnical logging

It was intended to quantitatively assess the fracture frequency per metre (FF/m) and the rock quality designation (RQD) as shown in Appendix D. The modus operandi of the on-site geotechnical logging was therefore as follows:

1. Any artificial fracture caused by the drilling process or resulting from the eventual handling of the core was discounted.
2. A smooth or somewhat weathered surface with soft coating or in-filling material such as calcite was counted as a pre-existing fracture.
3. Occasionally, part of the core rotated with the inner barrel, resulting in grinding of the fracture surfaces. In this case, the rounded surface was assumed to be pre-existing if it was difficult to decide whether the rounding was the result of a natural or artificial fracture.
4. Where fractures were very numerous (highly fractured), the frequency was determined by dividing the length of the highly fractured core by the average particle size of the fragmented core.
5. Where the core had been broken by handling or by the drilling process (i.e. fresh fractures), the broken pieces were counted as one unit, provided they formed a length of 100 mm as required for the RQD determination.

6. Since tip-to-tip length measurement involves double-counting at each end of a core piece and fully circular length measurement ignores core pieces that happen to have been drilled with a small subtended angle to one discontinuity (ISRM commission, 1978), centre line length measurement was used.
7. Fracture frequency and RQD were established for pieces of core not exceeding 1 metre in length.

Geological logging

This was intended to determine the coal type and grading on the basis of the proportion of bright elements. It was done immediately after the geotechnical logging. Cores were split so as to identify the various coal type zones and the corresponding lengths were measured. The data collected are presented in Appendix D.

Sampling of blasted coal

The blasted coal in muck piles was sampled according to the method described below. In No 2 seam, coal was sampled in both the top cut and the select cut according to the production selectivity requirements (M2T and M2S). Sampling was performed in the three pits immediately after blasts under the following conditions:

1. Samples were collected in line at equal distances 25 metres apart.
2. Since fines on the highwall side are generally greater than on the low-wall side (Slaughter, et al., 1992), two sampling points per sampling profile were considered, one corresponding to the product from the highwall and the other one to that from the low-wall as shown in Figure 6.2a. Where the mining face was narrow, a single central sampling point was considered.
3. Three samples per sampling point were collected from three locations

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corresponding to the top, middle and bottom of the pile using either a P&H shovel, a front end loader or, in some instances, manual shovels. The samples collected were loaded in drums and immediately sent to the preparation room.

4. The M2T and M2S mining horizons were sampled separately.

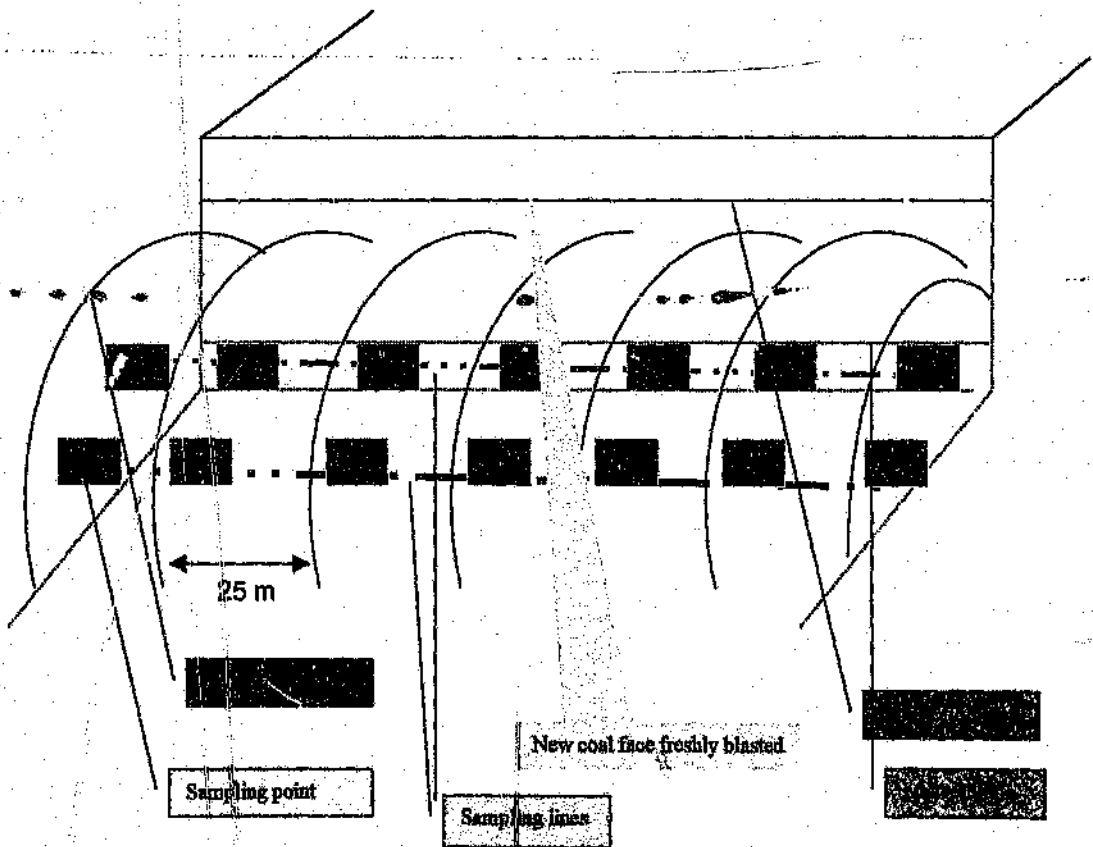


Figure 6.2a Lay-out of coal sampling done after surface blasting

Preparation of sample

Samples of blasted coal were screened and weighed for classification into size fractions (see Figure 6.2b). The screening was done in two streams that differed only in the aperture of the last screening stage (fine screen). One stream had a long slot screen of 0.5 mm x 1 mm and the other stream had a square mesh screen of 0.5 mm x 0.5 mm. The upper screening stages were 100 mm, 70 mm, 50 mm, 32 mm, 25 mm, 6 mm and 3 mm.



Figure 6.2b Screening equipment used for large industrial coal samples

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The long slot screen was used to establish the influence of the shape factor on coal fines. Long and thin fragments, although they lower the value of the coal as much as do square or roundish ones, are sometimes retained on a square mesh screen if they fall across their breadth: (across the longer dimension). To ensure that all the fines (under 0.5 mm) were accounted for, it was important to increase the probability of classification by increasing the chance of settlement in the bottom screen by using a long slot configuration. This configuration also made it possible to compare the results of fines observed from samples collected directly from the pit with those obtained from the routine assessment operations in the plant. Nine screening stages were used with the industrial vibrating screening equipment shown in Figure 6.2c to generate the data presented in Appendix D.



Figure 6.2c Nine screening stages used

6.3.4 Results, interpretation and discussion

6.3.4.1 RQD vs FF/m

Two indices are used to measure inherent fractures in rock and its consequent in-situ strength: the RQD (see section 6.2.2) and the FF/m (see section 6.2.1). These indices are generally used to define fractures of different size. The RQD is used for macro-fractures related to problems of stability of excavations in hard to medium bulky rock in mining and civil engineering excavations, i.e. tunnels, stopes and other underground workings. The FF/m is rather used for meso- and micro-fractures mostly related to medium and soft rock, for instance coal. A correlation was established between these indices in the coal from Kleinkopje, as shown in Table 6.2 and Figures 6.3a and 6.3b. The FF/m increases with the increasing depth of the coal seam. Thus the deepest coal of Seam 1 was observed to be the most fractured, with an average of 23, 30 and 50 fractures per running metre in respectively the 2A, 5 W and 3A pits of Seam 1 (see Table 6.2).

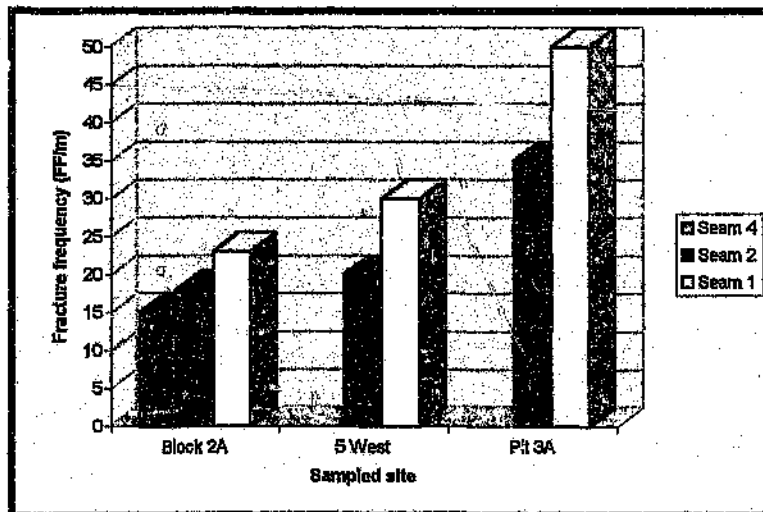


Figure 6.3a Observed FF/m related to the seam depth

Table 6.2 Average fracture frequency per metre and RQD at Kleinkopie

COAL SEAM		BLOCK 2A	5 WEST	PIT 3A
4	FF/m	15	-	-
	RQD (%)	61	-	-
2	FF/m	18	20	25
	RQD (%)	46	43	20
1	FF/m	23	30	50
	RQD (%)	33	25	5

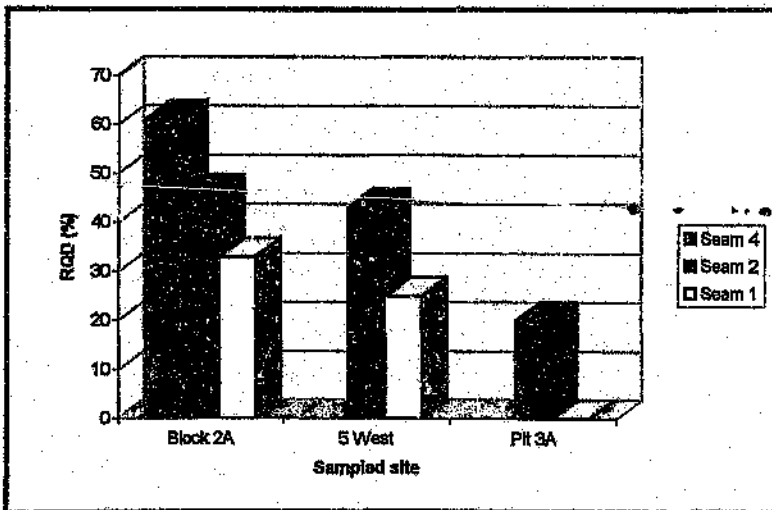


Figure 6.3b Observed RQD (%) related to the seam depth

It is noted that:

1. The trend observed at Kleinkopje of fracture frequency increasing with coal depth can be explained as resulting from the pressure of the overlying weight of soil on the deeper seams (action of gravity). The deeper the seam, the higher the pressure, implying a higher degree of fracturing.
2. The RQD was observed to decrease with coal depth. This trend is similar to that found by Priest and Hudson (1976) to exist between RQD and FF/m in hard and medium rock (sandstone, mudstone and chalk). These authors ascertained that an exponential relationship exists between the RQD and the FF/m of rock materials. The present investigation has led to a linear best-fit whose linearity parameters are given in Table 6.3 below and which has been formulated as follows:

$$\text{In Seam 1:} \quad RQD(\%) = -2.2FF / m + 86.5 \quad 6.3$$

$$\text{In Seam 2:} \quad RQD(\%) = -2.0FF / m + 86.9 \quad 6.4$$

$$\text{In Seam 4:} \quad RQD(\%) = -3.4FF / m + 111 \quad 6.5$$

Table 6.3 Relationship between RQD and FF/m

	SEAM 1			SEAM 2			SEAM 4		
	A	B	(%)	A	B	Δ (%)	A	B	Δ (%)
Block 2A	-3.0	102	95	-2.9	104	96	-3.4	111	98
Block 5W	-1.9	84.3	83	-1.7	77.6	61	-	-	-
Pit 3A	-	-	-	-1.7	78	70	-	-	-
Overall	-2.2	86.5	80	-2.0	86.9	70	-3.4	111	98

Noted: A is the slope of the observed linear function
 B is its intercept
 Δ is the confidence level in percentage.

The above negative linear relationship (Equations 6.3, 6.4 and 6.5) showed the limitations of the RQD as an index for characterising fractures in coal. Fundamentally, coal is a soft rock material that abounds in inherent fractures. Sampling by diamond drilling induced additional fractures which made further use of the RQD inappropriate.

6.3.4.2 Fracture distribution and coal grading

Coal grading is related to the amount of bright elements present. Kleinkopje Colliery groups its coal into two geological types categories: mixed bright and dull/lustrous. The types are easily assessed from the colour of the coal. Any amount of bright elements in a coal qualifies it for classification as mixed bright. Totally black coal, with no bright material is called dull/lustrous. A detailed classification is presented in Appendix D.

Mixed bright coal is generally seen in the bottom part of the thick No 2 seam. It constitutes the colliery's richest reserves since it yields the low-ash coal, which is in high demand. It is used for metallurgical applications due to its high oxidising properties. This coal is blasted selectively and handled with care as it is destined for the export market. Since it is found at depth in the seam, the intensity of fractures is expected to be higher than in the top coal horizons (dull/lustrous). Table 6.4 shows the distribution of fractures observed on site. The trend discerned confirms the expectations: fractures increase with the depth of the coal seam. It is also noticed that, in the same seam, there appear to be more fractures in the dull/lustrous, top coal horizons than in the mixed/bright, bottom horizons. This may be attributed to the difference in stress-transmission properties between the hard rock overlying the top coal horizons and the coal. Coal spreads overburden pressures longitudinally, thus cushioning the bottom horizons and alleviating the fracture induction.

Table 6.4 Coal grading and observed distribution of fractures

Seam	Pit	Fracture frequency per metre	
		Mixed bright	Dull/lustrous
4	Block 2A	15	18
2	Block 2A	19	24
	Block 5W	21	22
	Pit 3	35	30
1	Block 2A	15	22
	Block 5W	-	30
	Pit 3	-	50

6.3.4.3 Coal strength and fracture distribution

The strength parameters of coal in the different pits of the mine were assessed to determine their relationship with fractures and subsequent fines generation. Table 6.5 shows the results of the uniaxial compressive strength (UCS) tests undertaken on coal samples from the colliery. These are compared to the observed fracture frequency in coal. When plotted in Figure 6.4 a poor correlation was observed. This was obviously attributed to the role played by the material filling the fractures (nature and type of filling). Depending on the ratio of filled to open fractures, the compressive strength may vary even within the same coal block. In the case of the present investigation, the filling of the fractures was mainly an intrusion of calcite or some grains of quartz and other minerals in coal.

Table 6.5 Compressive strength vs fracturing in coal

Coal seam		Block 2A	Block 5 W	Pit 3A
4	FF/m	15	-	-
	UCS (MPa)	12.94	-	-
2	FF/m	18	20	35
	UCS (MPa)	18.66	10.5	13.2
1	FF/m	23	30	50
	UCS (MPa)	-	17.4	-

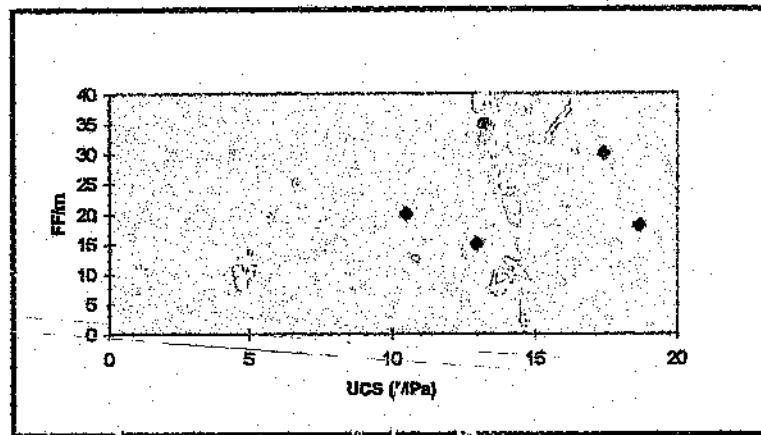


Figure 6.4 Fracture frequency per metre vs coal strength (UCS)

6.3.4.4 Fracture frequency at the highwall

Fractures induced by blasting were assessed in an effort to relate the blast effects and the resulting fractures induced at the highwall. It is common knowledge that blasting induces additional fractures in coal. The interaction of such induced fractures with the pre-existing fractures can lead to further breakage and eventual fines generated (< 6 mm). It was therefore considered important to log, after each blast, every apparent fracture at the highwall in a plane situation (two orthogonal directions referred to as horizontal and vertical). The data on the total fractures observed at the highwall were used to assess the law of fracture increase in relation to the blast parameters. The blast-induced fractures were determined by subtracting the in-born fractures pre-determined from the total fractures observed at the face after blast.

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For technical reasons and safety considerations, the logging of face fractures was limited to one coal horizon in Seam 2. Ten sampling points were considered. Two techniques, namely photography (see Appendix D) and physical counting of fractures at the face, were used to produce reliable results.

The fractures were logged along a sampling line in two directions, vertical and horizontal. The sampling line was drawn on the face and apparent fractures crossing the line were scanned with a digital counter. A summary of the results obtained is presented in Table 6.6 and plotted in Figures 6.5a, b and c.

No significant correlation was found between fractures in orthogonal directions (vertical and horizontal). Although a linear relationship has been established in the graphs below, the confidence of correlation is lower than the acceptable threshold. This result is attributed mainly to the forces generating the fractures in coal. The so-called horizontal fractures are generally a result of the static loading of the coal horizons by the overlying ground, induced pressures resulting from activities on the surface and other geo-tectonic movement to which the earth's crust is subjected (meteorites, etc.). The vertical fractures are mainly due to contraction caused by changes in the weather and to the tangential compensating component of the compressive pressures generated by the above-mentioned loading. However, in terms of fines generation, the horizontal fractures produce fines by the mode shown in Figure 3.2a. The vertical fractures tend to oppose the propagation of the breaking stresses emanating from the blasthole, therefore involving the generation of fines by compression (see model discussed in Chapter 2, section 3.3.2.2).

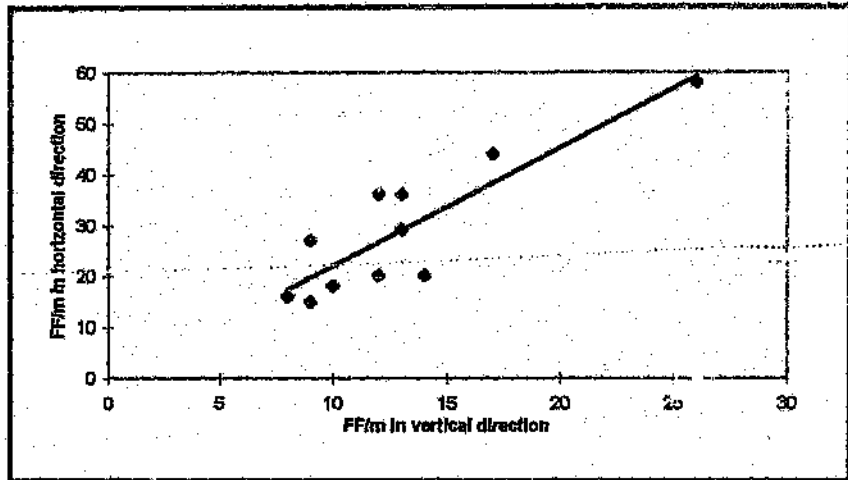


Figure 6.5a Observed relationship between vertical and horizontal fractures in Block 2A

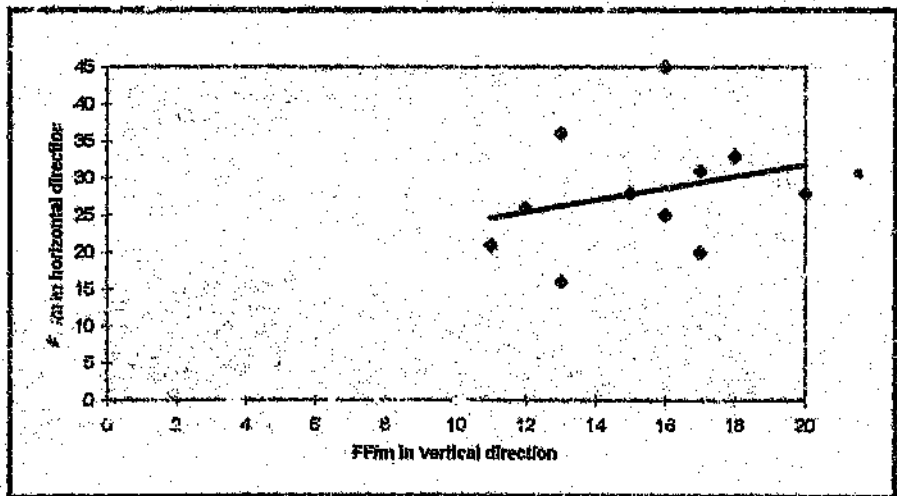


Figure 6.5b Observed relationship between vertical and horizontal fractures in Block 5W

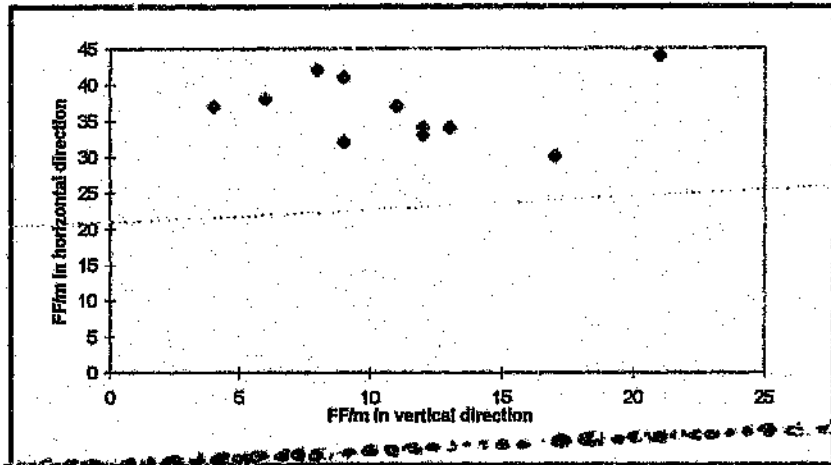


Figure 6.5c Observed relationship between vertical and horizontal fractures in Pit 3A

Fines generation depends on how the fractures are oriented relative to the expansion front of both blast-induced p-waves and s-waves. The interaction of the fracture and these wave components will determine whether the fracture expands in tensile or in shearing mode. However, intuitively one would expect the horizontally oriented fractures to develop from flaws and along the beds (heterogeneous contact) of the various deposited constituents of coal. They are aligned along the path of the breaking stresses emanating from the blasthole and therefore involving more of the fines generation by shearing (see Model discussed in Chapter 3, section 3.3.2.1). The geotechnical work of logging fractures done in the present investigation dealt mainly with this type of fractures. The problem matter in the next few pages evolves around whether these horizontally oriented fractures are significantly related to the coal fines in blasting piles.

Table 6.5 Observed fractures at the highwall

Sampling point	Block 2A		Block 5W		Pit 3A	
	H	V	H	V	H	V
S1	44	17	33	18	38	6
S2	16	8	20	17	34	13
S3	18	10	45	16	42	8
S4	20	14	31	17	37	4
S5	20	12	26	12	33	12
S6	15	9	36	13	32	9
S7	36	13	21	11	44	21
S8	58	26	28	20	30	17
S9	36	12	16	13	41	9
S10	27	9	25	16	34	12
Average	29	13	28	15	37	11

Note: V = vertical
H = horizontal

6.3.4.5 Observed fragmentation in blasted coal

Blasting work undertaken in the areas where coal samples for the geotechnical assessment of inherent fractures were collected was closely monitored and samples were taken to determine the resulting fragmentation. The blast design was kept identical throughout the series of blasts. The samples collected after each blast were weighed and screened for coal sizing. The coal fragments were grouped into ten classes, each of which was weighed to determine the function of the size distribution relative to the blast being considered. Ten equidistant sampling points were identified on the pile. A volume of 0.08 m^3 of coal was placed in a drum to constitute an individual sample. Three samples were systematically collected at each sampling point to correspond to the top, the middle and the bottom of the pile. The size distribution data obtained from the thirty samples from each blast were recorded. The average fragmentation per blast and the respective characteristic curves are given in Appendix E. However, Figures 6.6a,b,c show the overall trend of the fragmentation functions obtained. These figures are plotted from the data of the average fragmentation observed in a number of blasts monitored in different pits of the colliery.

The fragmentation results plotted showed clearly that in general the trend of the function of size distribution of blasted coal is bi-modal in nature. Observed functions were either skewed to the left, towards the modal undersize of $< 25 \text{ mm}$, or skewed to the right, towards the $> 25 \text{ mm}$ oversize. The modal size of 25 mm is far below the mean size of the range considered 50 mm , (i.e. 0 to 100 mm). This implies that the blasts resulted in overbreaking of coal.

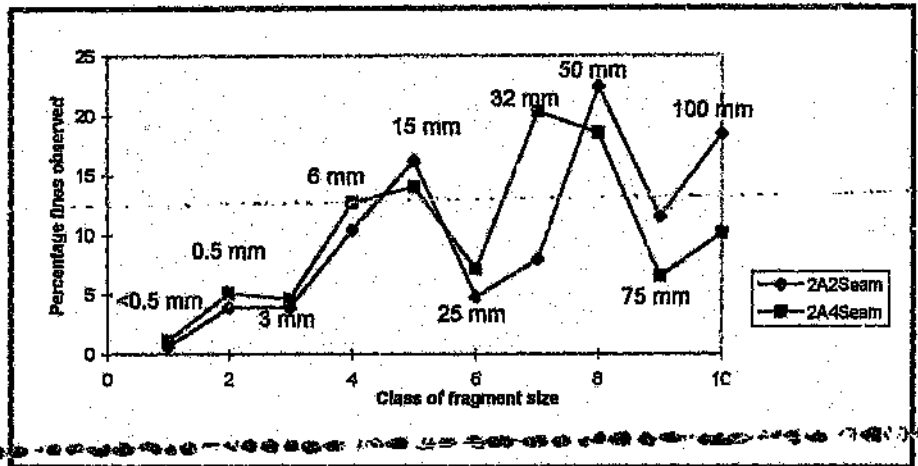


Figure 6.6a Size distributions observed in blasted coal of Pit 2A

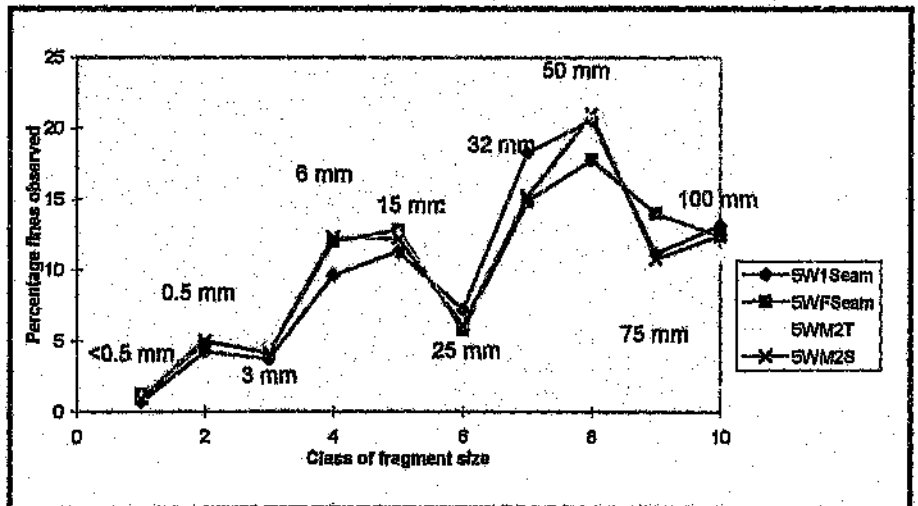


Figure 6.6b Size distributions observed in blasted coal of Pit 5W

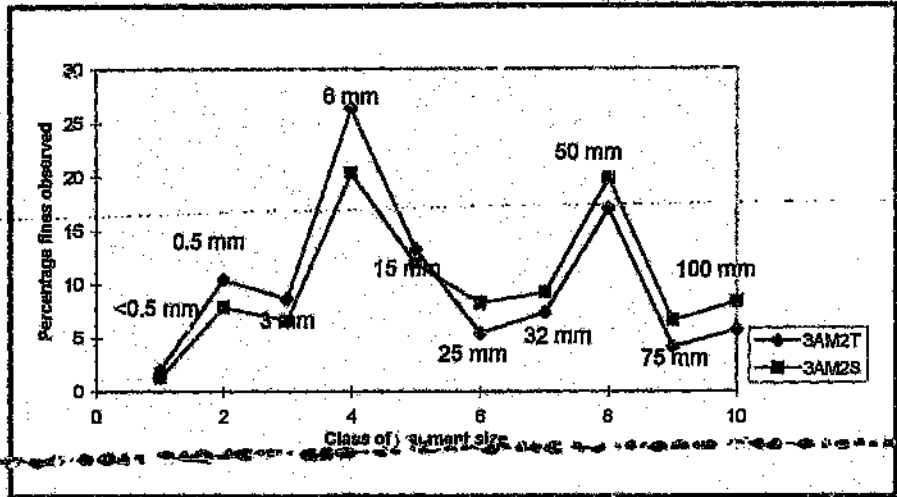


Figure 6.6c Size distributions observed in blasted coal of Pit 3A

Such fragmentation results indicated that more work on blast optimisation was still required to fine-tune the blasting practice at the mine. However, the fragmentation curves highlighted peaks of accumulated fragment size in the intervals of 6 mm to 15 mm in the under-modal class and 32 mm to 50 mm in the class of bigger fragments. A correlation with the fracture frequency was sought and Table 6.7 below shows the prominent peaks of fragment size resulting from the blasting work undertaken for these investigations. The prominent peak in each coal horizon is underlined in the table.

Table 6.7 Observed peak fragment sizes and fracture frequency

Coal horizon		Peak fragment sizes	FF/m
Block 2A	Seam 1	-	23
	Seam 2	15 mm - <u>50 mm</u>	18
	Seam 4	15 mm - <u>32 mm</u>	15
Block 5W	Seam 1	15 mm - <u>50 mm</u>	30
	Seam 2	6 mm - <u>50 mm</u>	20
Pit 3A	Seam 2	<u>6 mm</u> - 50 mm	35

Considering that Seam 2 is mined in both pits of the colliery, it is worth noting the significant relationship between the fragmentation and the fracture frequency. The prominent peak fragment size is shifted from the fragment class of bigger size (50 mm) in coal of Block 2A that has distant inherent fractures, to the class of smaller size (6 mm) in coal of Pit 3A that has a higher fracture frequency. In the coal of Block 5W, where the difference in fracture frequency compared with that of Block 2A is small, the prominent peak does not move from 50 mm. However in coal of Pit 3A, more coal of smaller size is produced, thereby shifting the smaller peak of the fragmentation curve in favour of the fragment class of smaller particles (6 mm). This trend observed may be summed therefore to say that, the closer the fractures, the more coal of smaller size is produced in the blast, hence the fracture frequency has a definite influence on the size of fragments generated by the blast.

It must be borne in mind that blasting involves two complementary energy phases that contribute to the formation of the fragment sizes (shock and

gas). The partition between these phases achieved during the blast determines the share of the resulting particle sizes. At present, it is not possible to accurately delineate the respective shares of the two energy phases in the fragmentation. Nevertheless, the effects of both phases are hampered by the inherent fractures in coal. Thus it is logical that the observed trends should remain valid regardless of which phase is prominent in particular blasting conditions. However, the reducible portion of coal fines is believed to result solely from the effects of shock energy. These fines are set by two different crushing mechanisms, depending on the proximity of the sampling point to the axis of the in-hole explosive. The closer the sampling point to the hole, the higher the expected reducible fines since in this region, primary crushing is the prevailing dominant mechanism of fines generation. The magnitude of the crushing shock impulse decreases with radial distance. Far away from the borehole, the pulse interacts with pre-existing fractures to generate additional fines that are difficult to control and optimise. At such locations, the secondary crushing mechanism is dominant.

6.3.4.6 In-pit coal fines

The conventional practice used in industry to assess the coal sizes generated in mining operations is to collect coal samples from the feed to the washing plant on the conveyor belt. For the present thesis, coal was sampled directly from the blasting faces in order to relate the fines generation to the blasting parameters. This was done to avoid the generation of additional fines that may occur during the intermediate operations prior to the washing plant (loading, transportation or stockpiling). At Kleinkopje Colliery, Delaporte and Heizmann (1995) had ascertained that in addition to the in-pit coal fines, an average of 3.3 % fines was incrementally generated during the loading and transportation

operations.

A. Pioneering efforts to reduce fines at Kleinkopje

Investigations aimed at reducing the in-pit-generated coal fines were conducted at Kleinkopje in 1993. Van Zyl (1993) reported to have achieved good results in improving the overall fragmentation, particularly the fines generation, using the technique of decoupled charge by using cardboard tubes in vertical blastholes to hold the charge of bulk emulsion (Powergel P115) in place. He incrementally varied the in-hole decoupling (diameter of blasthole to diameter of charge), using the ratios 1.3 (108 mm to 83 mm), 1.6 (130 mm to 83 mm) and 1.9 (130 mm to 70 mm). Unfortunately, he used a visual technique to assess the results and therefore no supporting records are available. However, the technique was introduced at the mine and is still being used for coal production.

Delaporte and Heizmann (1995) identified four major factors contributing to the amount of coal fines in mining products. They pertain to the geological conditions, the conditions of the site (pre-mined or virgin area), the blasting factors and the loading/transport operation. From their examination of the in-pit-generated fines, they identified a trend of decreasing coal fines with the drilled depth of the blasthole. The deeper the hole, the fewer in-pit fines are generated. Their finding is acceptable as, for the same mass of charge, a longer hole implies lowering the charge burial depth. This means that either the area of contact between the explosive and the coal is reduced, or the depth of the uncharged collar of the blasthole is increased. The result is therefore a reduction of the specific breaking energy in the coal burden, which translates into fines' reduction. They also confirmed the well-established blasting principle that the more explosive used the more crushing generated. Figure 6.7 below

depicts their observed trend of increasing coal fines with increasing powder factor in coal blasted.

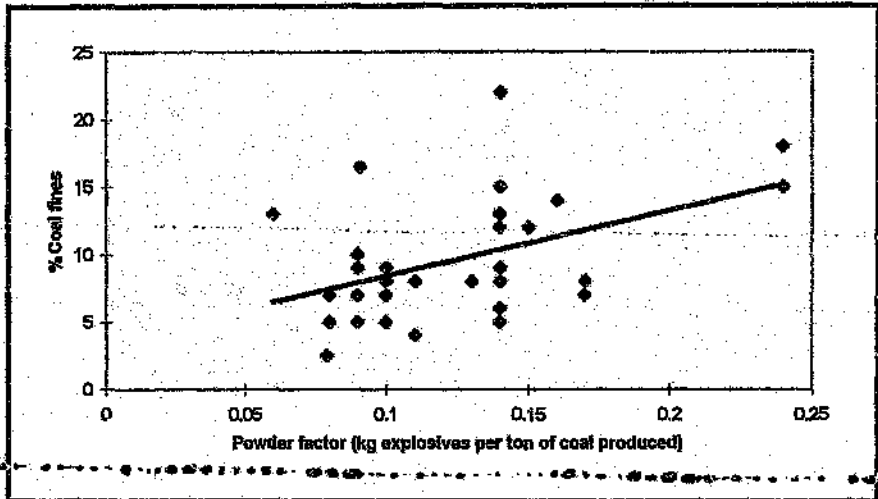


Figure 6.7 Observed coal fines per powder factor (After Delaporte and Heizmann (1995))

B. The present investigation

The trend of increasing fines with increasing powder factor brings about the need to understand the energy mechanisms involved in coal fines generation. This thesis puts forward the hypothesis that the shock energy phase has a dual action, affecting both the primary and secondary crushing mechanisms in coal during blasting. While the influence of the primary crushing mechanism is not in doubt, the present work sought to establish an eventual correlation between the secondary crushing mechanism (involving inherent fractures) and the fines generated in blasted coal.

The fragmentation observed in the blasted coal sampled was studied further to investigate fines generation (Figure 6.8a, b and c). The < 0.5 mm fraction is termed coal fines as in the previous work conducted at the mine and reported above. However, the economic cut-off size recommended by the South African Chamber of Mines is < 6 mm (Thorpe and Pedroncelli, 1986). This definition of fines as < 0.5 mm is usually adopted to avoid market penalties imposed on coal with a high moisture content. Since the < 0.5 mm fractions retain moisture longer and stick to coal lumps, preventing them from drying well, they are discarded from the saleable coal product.

The data on in-pit coal fines resulting from the blasting work undertaken are compared with the fracture frequency in Figure 6.9 and Table 6.8.

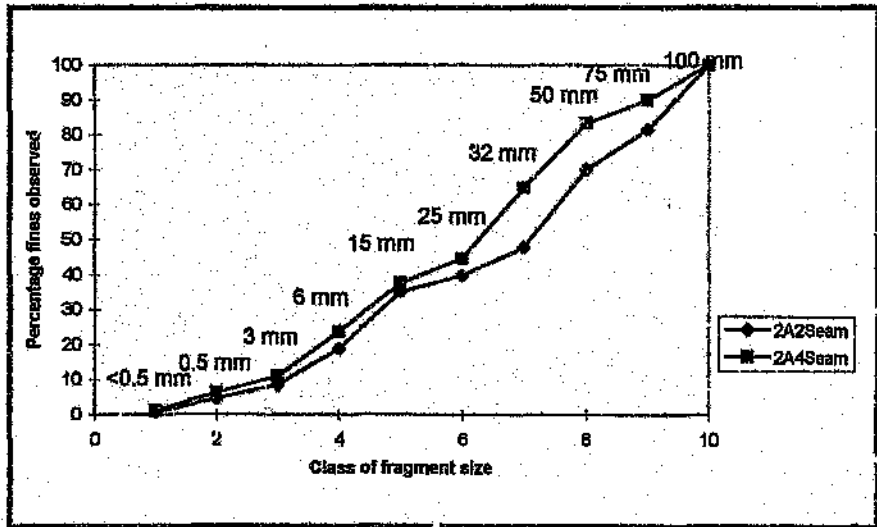


Figure 6.8a Cumulative size distribution functions of blasted coal in Pit 2A

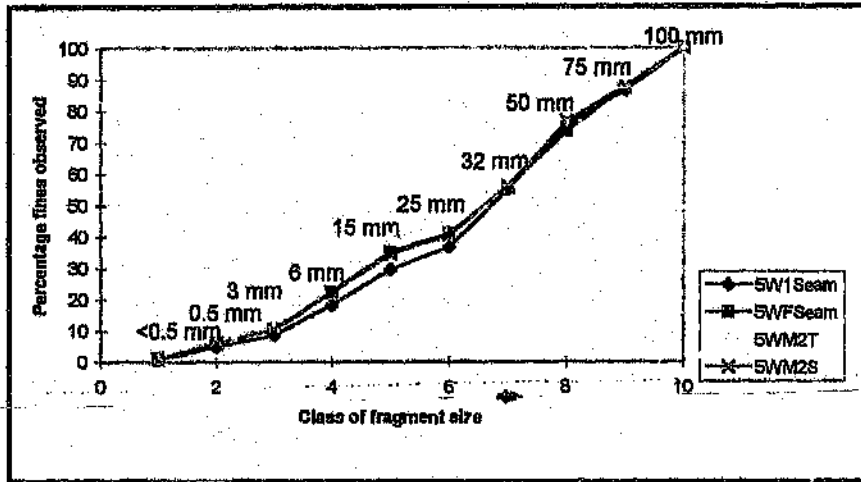


Figure 6.8b Cumulative size distribution functions of blasted coal in Pit 5W

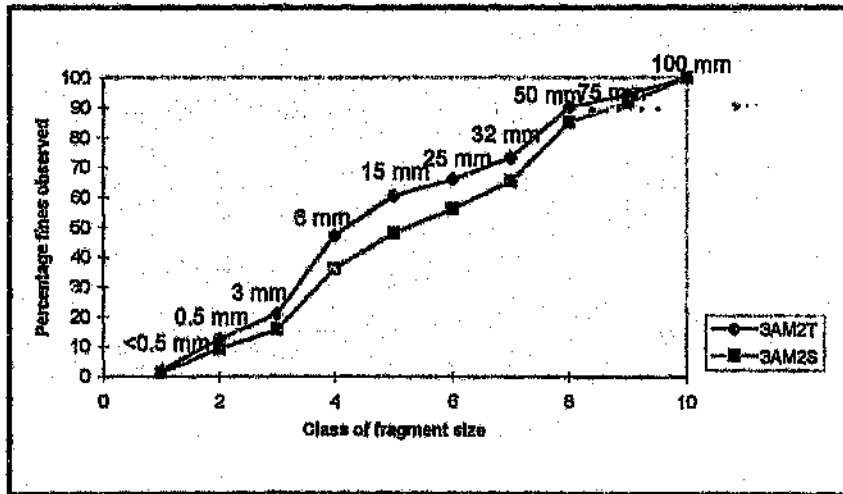


Figure 6.8c Cumulative size distribution functions of blasted coal in Pit 3A

Table 6.8 In-pit coal fines vs fractures in coal at Kleinkopie

Coal seam		% coal fines		FF/m
		< 0.5 mm	< 6 mm	
Block 2A	Seam 2	0.6	8.4	18
	Seam 4	1.1	10.8	15
Block 5W	Seam 1	0.7	8.6	30
	Seam 2 Full	1.2	10.3	20
	Seam 2 Top	1.1	11.2	17
	Seam 2 S	0.9	9.9	23
Pit 3A	Seam 2 Top	2	21	36
	Seam 2 S	1.3	15.7	35

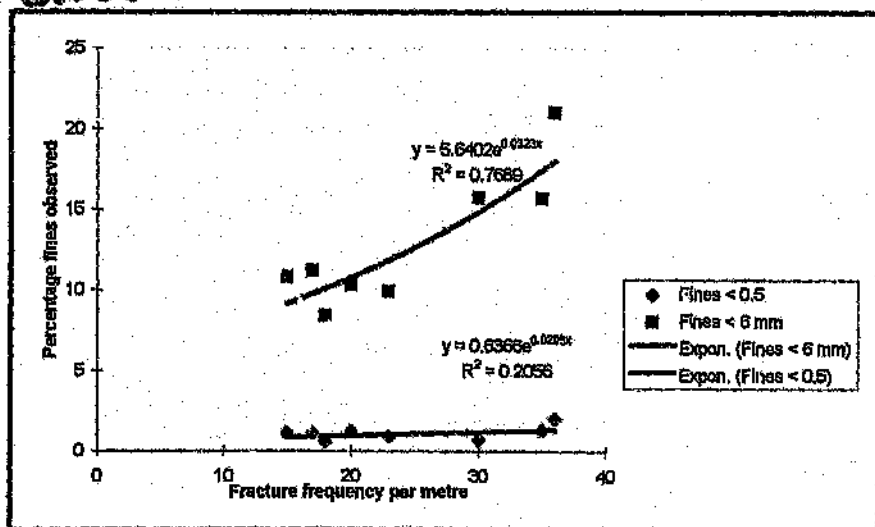


Figure 6.9 Percentage fines in blasted coal vs observed fracture frequency per metre (FF/m)

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It is noted that there is a strong exponential correlation between the generation of coal fines (< 6 mm) and the fracture frequency observed in coal. The coefficient of this correlation is 77%. This corresponds to a confidence level of well above 95%. However, a poor relationship is observed between the < 0.5 mm fraction and the fracture frequency. This result implies that the host of < 0.5 mm fines in coal piles are generated rather by the concussion of the primary crushing mechanism. Given the fracture frequency in coal, the following equation can be used to predict the level of economical generation of coal fines (< 6 mm) at Kleinkopje:

$$Y = 5.64 e^{0.032 x}$$

6.6

where Y is the percentage of < 6 mm fines generated, and
x is the FF/m observed.

6.4 Conclusion

The size of the samples collected from the blasting piles may appear relatively small in comparison to the huge tonnage of broken coal per blast usually handled in surface mining. The economics of the investigation and the reliability of the results required dictated the methodology presented. The technique of coal screening used to determine the fragmentation ensures high accuracy of results despite the limitation of sample size and not taking into account big blocks. The sampling methodology was considered sound and it was implemented carefully to give credibility to the final results of the investigations. These results allow various conclusions to be drawn:

1. The higher the inherent fracture frequency in coal, the higher the amount of fines generated in blasting. There is an exponential relation

between the inherent fractures and the fines generated (see Figure 6.9).

2. Longer blastholes result in less fines being generated in blasted coal. This is due to the favourable distribution of breaking energy resulting from either a longer collar or a reduction in the contact surface between the explosive charge and the coal. However, if the powder factor is increased in a longer hole, more fines can be expected.
3. The use of a decoupled explosive charge in coal blasting is an effective technique for reducing coal fines generation in surface blasting (vertical blastholes).
4. There is no correlation with the fracture frequencies in orthogonal directions (vertical and horizontal). This indicates a difference in their respective setting mechanisms.
5. Coal fines generation is site-specific. A variance of 8.4% to 21% by mass of blasted coal was observed at Kleinkopje. An average of 12% coal fines per blast can be accepted as characterising the in-pit-generated fines. The objective of an optimal blasting approach would then be to reduce this percentage yield.

7 INVESTIGATION OF THE GENERATION OF FINES IN UNDERGROUND BLASTING

7.1 Introduction

The trend in modern coal mining is towards the mechanisation of the primary breaking operations in both underground and opencast mines. However, in South Africa, blasting is still used in all surface coal-mining operations and in a number of underground collieries. The keyword for the survival of the mining industry of this country into the next century is 'productivity'. This means that maximum output has to be achieved at the minimum possible working cost. To satisfy this condition, it is necessary for every individual mining operation, including drilling and blasting, to be run effectively. This chapter presents an overview of the current blasting practice used in the local coal mining industry. It highlights the generation of coal fines resulting from the current blasting technology used. It also presents the results of investigations undertaken to seek ways to optimise coal breaking by reducing the fines. Finally, it discusses the technique suggested to curb excessive fines generation in blasted coal.

7.2 Coal blasting methods for reduced fines

The major problem with the coal fragmentation obtained at the face is essentially the undersize rather than the oversize fraction. The latter can still be reduced to the required size by a mechanical feed-breaker before delivery onto the conveyor belt. For optimisation of its blasting results, a coal operation needs first to consider ways of controlling the generation of fines.

In recent years, one of the new techniques developed to control coal fines in blasting has been the active stemming technique widely used in

opencast coal blasting in Russia. Kutuzov (1994) reports that the fragmentation results obtained with this technique are highly successful. The basic idea of the technique is to use a type of a dense low explosive to stem the main high-explosive charge in the blasthole. In this case, the blasthole is fully active as opposed to the conventional inert stemming. The overall time of interaction between the breaking energy and coal is increased. Better breakage may be expected since, in addition to the extended reaction time, the violence of the impact of the head of the shock waves at the interface of the charge and the inert stemming is reduced. Instead of a violent impact, a rather smooth transition of reaction rates (higher to lower rate) occurs at the interface of the explosive products. These conditions are conducive for reduced fines generation.

Another technique for reducing fines generation in underground mines is Blasting Off The Solid (BOTS). In this method, no mechanical cut is required in the coal face prior to the drilling of blastholes for the blast. The method is similar to that used for the conventional development of underground tunnels. BOTS uses a number of closely drilled blastholes, charged in a particular way with the explosive and fired so as to break a central core in the coal face and provide an additional free face for the blast. Several configurations of cut holes have been tried out overseas and the wedge cut emerged as the most effective. It is used either in the middle of the coal face or on one corner in some mines (slab shoot). As pointed out by Cunningham (1991) a minimum of 30 % more drilling and explosive is required in the BOTS. Despite this disadvantage over the currently used undercutting method, Landman (1993) and recently Lubbe (1995) believe that BOTS may be a viable alternative to undercutting. BOTS produces minimal coal fines since almost 5% of the coal that ends up in the category of fines due to the undercutting required in the blasting method prescribed

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by the South African law, is saved.

The blasting method prescribed by the mining legislation of this country is "Coal Cut Blasting" (CCB). A slot is undercut in the coal face to serve as a free face towards which the coal broken by an individual row of blastholes is thrown. This configuration allows rapid decompression due to the quick release of pressure in the coal, thereby reducing the probability of gas ignitions underground. The cut is made by a mechanical coal-cutter powered by an electric motor supplied through a power line extended to the face. This undercutting produces a considerable amount of small coal particles which, when added to the fines generated by blasting, results in nearly a third of the blast output being only marginally profitable. In this respect, BOTS has a net advantage over the legislated method. However, the additional number of blastholes required to break coal efficiently implies a high powder factor making therefore CCB the preferred method. The alleged hazard caused by inadequate venting of gas at the coal face when BOTS is used has not been proved beyond reasonable doubt. Therefore, it is still possible that BOTS is a viable solution to the high levels of fines generated with the CCB method currently used in the local industry. So far, most trials undertaken locally and overseas have not proved clearly yet that BOTS is in fact an unsafe blasting method (Nagy et al., 1952; Ireland and Stern, 1987; Cunningham, 1991 and Lubbe, 1995). On the other hand considerable further work is required to prove that BOTS can be safely used and to determine the safety criteria to be applied when BOTS is used.

7.3 Blasting technology in underground coal mines

The blasting technology used in underground collieries in South Africa was investigated at the Greenside and Arthur Taylor Collieries which are

situated in the vicinity of the Kleinkopje opencast mine where the previous investigations discussed in Chapter 6 were conducted. This second phase of the research was aimed at finding a technique that will result in fewer fines in blasted coal products mined underground.

The approach used in this research entailed forging an understanding of the problem of fines generation in the local industry by means of observing blasting operations in the selected mines. These mines were considered to be a representative sample of the industry. The approach entailed also searching for the shortcomings in the current blasting practice used underground and proposing technical ways of reducing fines generation in the blasting operation.

7.3.1 Evaluation of in-mine generated coal fines

The Chamber of Mines runs a section at Greenside colliery in which trainee miners from the entire local coal mining industry are trained. Work is done according to the legal recommendations and is of a high standard. This section mines coal in the No. 4 Seam. That is the site which was considered for the present research along with two other sections of Tavistock's Arthur Taylor Colliery (Section 2 of which mines the coal of Seam No. 2 and Section 3 the coal of Seam No. 4). The two sections of Tavistock may be called 'normal' production sections' as opposed to that of Greenside, which does not have a production target, because it uses labour-in-training. Generally speaking, the blasting standards prescribed were fairly well adhered to throughout these investigations.

7.3.1.1 The current technology (see Table 7.1)

Coal cut blasting is used to mine coal by the bord-and-pillar mining method. The bord dimensions are respectively 7.2 m wide x 3.3 m high in Seam 4 at Greenside; 6.6 m x 3.8 m in Seam 2 and 6.2 m x 2.5 m in Seam 4 at Arthur Taylor Colliery (ATC).

Table 7.1 Parameters of the current blasting technology used in coal mines

	Bord size (m x m)	Number of holes	Sequence (ms delay)	Explosive
Greenside	7.2 x 3.3	15	5,3,0	Enercoal
ATC Seam 2	6.6 x 3.8	21	8,5,3,0	MX8
ATC Seam 4	6.2 x 2.5	18	8,5,3,0	MX8

The blast patterns used (drilling and timing) depend on the strength properties of the coal (compressive strength and fracture situation at the face) and the size of the bord. Figures 7.1a, b and c depict the commonly used patterns. At Greenside, blastholes were drilled using a 38 mm bit powered by a rotary drill which was manually operated. At Tavistock, a jumbo drill was used to drill 40 mm diameter holes. The explosive charges used at these mines are respectively ENERCOAL, a permitted watergel product manufactured by DANTEX at Greenside, and MX8, a permitted emulsion product manufactured by SMX at Tavistock. Both products are packaged into cartridges of 29 mm diameter weighing 200 g each. The explosive charge is train-loaded into the blasthole. Four cartridges equivalent to 800 g are used per hole. They are tamped and stemmed

using a mixture of clay and sand. Prior to drilling, a cut is made in the coal face with the coal cutter to a length of approximately 2.5 m at Greenside and 2.7 m at Tavistock. Blastholes are drilled to the respective lengths of 2.3 m at Greenside and 2.5 m at Tavistock. While at Greenside advances of 80 to 100 % of drilled length could be achieved, at Tavistock the average was around 75 %.

Theoretically, engineering characterization of the coal needs to be undertaken regularly so as to determine the strength properties of and fracture situation in the coal in order to update the blasting parameters to accord with the change in the coal's physical and mechanical characteristics. There cannot be a single solution to blasting in an in-situ coal which has changing properties (Kabongo and Du Plessis, 1997). It is advisable for management to monitor the strength properties closely as these constitute an important parameter for the optimization of blasting as the coal face progresses underground. Sing and Pal Roy (1993) defined the optimum blast as that which is associated with the efficient utilization of blast energy in the breaking process. Such blast will reduce the blasting cost through using less explosive and wasting less explosive energy, with less throw of projectiles and materials. It will generate reduced blast vibration and result in a greater degree of safety and stability of the nearby structures.

The definition above mentioned underlines the great importance of the blast initiation system used in blasting. In local coal blasting, the row-by-row timing pattern is preferred. The breaking sequence of coal is obtained through the use of millisecond electric delay detonators, with increasing delays from the cut outwards. In previous research conducted at Kriel Colliery aimed at optimising the inter-row timing in blasting, Lubbe and

Kabongo (1995) observed that there was a considerable increase in the coal face advance per blast and the consequent broken coal obtained per blast when longer inter-row delays were used. It was therefore proved that the optimum inter-row timing is that which delays the initiation of consecutive rows of blastholes long enough to maximize the interaction of the generated breaking stresses between neighbouring holes. It is also important that the timing of the detonation of the next row of holes be delayed long enough to avoid sympathetic detonations which may freeze coal at the face. In addition, longer delays allow a proper recovery of the explosive detonability in the next row. The detonation of the first row of blastholes of the pattern generates shock stresses that are irradiated into the surrounding coal, thereby influencing the properties of the charge in the consecutive row. This may cause what is known as cross-blastholes dynamic desensitisation if a full recovery is not achieved in the subsequent row of blastholes (Wieland, 1987).

Mining legislation restricts the sequence of coal breaking (inter-row timing) in colliery blasting to a maximum delay of 150 ms between rows of blastholes (Mines Acts and Regulations, 1st and 2nd schedules, 1990). Table 7.2 presents a range of detonation delays that may be used in collieries. Numbers 0 to 5, corresponding to delays of 5 ms to 150 ms, are the only legally prescribed for use under normal conditions of coal blasting. The extended delays may be used provided that permission is obtained from the Inspectorate of mines.

Blasting work done for the present investigations used the following timing sequences. At Greenside a 5-3-0 pattern was used with Statsafe Carrick¹

¹ AECI trade mark

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detonators supplied by AECI². 15 blastholes were drilled in to extract coal from the No. Seam 4. At ATC, Tavistock, a 8-5-3-0 pattern was used with copper short period delay (SPD) supplied by SMX³. 21 and 10 blastholes were drilled in respectively the No. 2 Seam and the No. 4 Seam.

Table 7.2 Range of available permitted detonation delays

Statsafe Carrick number	Nominal delay period (ms)	Copper SPD number	Nominal delay period (ms)
0	5	0	0
1	30	1	25
2	55	2	45
3	80	3	65
4	110	4	90
5	150	5	115
6*	200	6	145
7*	250	7	175
8*	300	8	205
		9	235
		10	270
		11	310

* Extended Statsafe Carrick Detonators.

It is noted that the above list of SPDs may be not exhaustive. However, it contains those that are mostly used in the local industry.

2 AECI: African Explosives and Chemical Industries Ltd., explosives manufacturer and distributor
3 SMX: Sasol Mining Explosives Ltd., explosives manufacturer and distributor

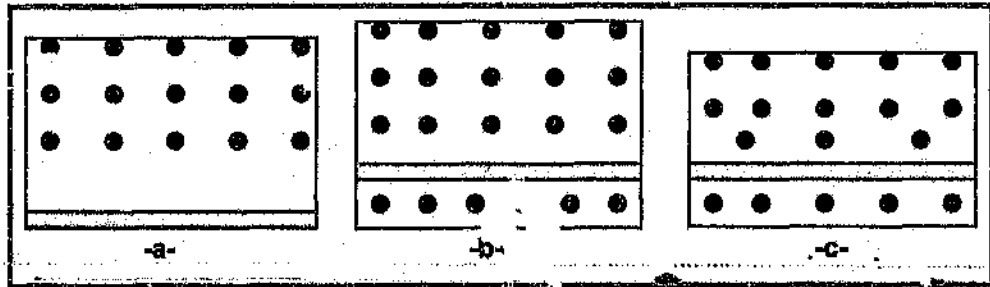


Figure 7.1 Typical blasting patterns used

- a) In Seam 4 at Greenside (soft coal, 15 holes)
- b) In Seam 2 at Tavistock ATC (stronger coal, 21 holes)
- c) In Seam 4 at Tavistock ATC (medium-strength coal, 18 holes)

The trial blast work that was undertaken under the conditions described above was monitored to determine the fragmentation and particularly the levels of coal fines generated in-mine. The methodology used for sizing the coal is described in the paragraph below.

7.3.1.2 The methodology used for coal sampling and sizing after blasts

Singh and Pal Roy (1993) are quoted as asserting that quantitative assessment of fragmentation from blasting on a mine-production scale is a very difficult task. The most reliable and accurate quantitative method for sizing material in the muck pile requires the screening of the entire mass of the blast outputs. However, this is impractical on a productive mine scale. Therefore, the present study used a sampling methodology dictated by the economics of the screening operation and the statistical imperatives for assuring representativeness of the blasted coal.

Approximately 300 kg of blasted coal was collected from eighteen sampling points in nine areas of the pile, as shown in Figure 7.2. Blasted coal in the pile was collected using spades at indicated positions in the pile as the loading of shuttle cars proceeded. A number of full spades was taken in the sampling area as indicated below during the intervals between the loading of consecutive shuttle cars.

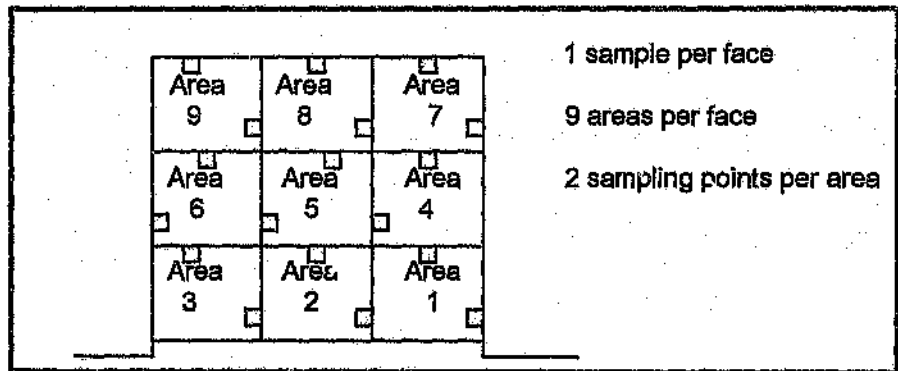


Figure 7.2 Sampling method used

Coal sizing was done by sieving through a four-stage column of sieves of the following apertures: 25 mm, 6.3 mm, 3 mm and 0.5 mm. The sieves were shaken by electrical vibrating equipment ensuring that additional coal breakage by material collision was minimal. Five coal fractions were classified, providing data for the plots of fragmentation characteristics presented in Tables 7.3, 7.4 and 7.5.

7.3.1.3 The observed fragmentation

Approximately ten faces per site were blasted under strict control. The three sites considered were, Seam 4 at Greenside, Seam 2 and Seam 4 at Tavistock, ATC. The following coal yield results were observed:

Table 7.3 Coal yield at Greenside Colliery, Seam 4

YIELD	DESIGNATION	CHARACTERISTIC SIZE
4 (#)	Oversize	> 50 mm
49 %	Nuts	> 25 mm
29 %	Peas	25 mm to 6.3 mm
18 %	Duff	< 6.3 mm
4 %	Dust	< 0.5 mm

Total fines: 22 %

Table 7.4 Coal yield at Tavistock, ATC, Seam 2

YIELD	DESIGNATION	CHARACTERISTIC SIZE
0 (#)	Oversize	> 50 mm
36 %	Nuts	> 25 mm
39 %	Peas	25 mm to 6.3 mm
21 %	Duff	< 6.3 mm
4 %	Dust	< 0.5 mm

Total fines: 25 %

Table 7.5 Coal yield at Tavistock, ATC, Seam 4

YIELD	DESIGNATION	CHARACTERISTIC SIZE
0 (#)	Oversize	> 50 mm
45 %	Nuts	> 25 mm
31 %	Peas	25 mm to 6.3 mm
20 %	4 Duff	< 6.3 mm
4 %	Dust	< 0.5 mm

Total fines: 24 %

It was noted that Seam 4 at Greenside yielded fewer coal fines (22 %) compared with the same Seam 4 at Tavistock (24 %). The difference in the coal strength (see Tables 1 and 2 of Appendix B) and the number of holes drilled per face (see Figure 7.1) could be the cause of the increase in fines. However, the respective values of the dynamic elasticity modulus of coal at these sites are so close that the big difference in crushability observed does not appear to be justified. Consideration of the explosive charges used gave some insight, as the VOD of the watergel is much lower than that of the emulsion used at Tavistock (3000 against 5000 m/s). It was concluded, therefore, that the in-hole performance of an explosive charge as assessed from the VOD achieved may be the factor that determines the reducible coal fines.

In conclusion, it was found that under controlled conditions, the blasting technology used in local mines results in nearly 30 % of the coal output

being in the form of fines (under 6 mm). The blasting technique alone generates approximately 25 % of fines. To this percentage must be added almost 5 % more, calculated from the assumption that each blast produces 60 tons of coal and generates 3 tons of duff (5 %) from the undercutting at the face to create the breaking face.

7.3.2 Criticism of the actual blasting technology

7.3.2.1 Understanding of the coal fines problem

The provision for two free faces in coal blasting contained in the country's mining legislation is based on the apparent belief, widely held in the industry, that the undercutting of coal at the face provides safer conditions than solid blasting. Landman (1993) questioned Murphy of the USBM on the validity of this opinion. The latter was critical of the research done so far that has been unable to substantiate the opinion beyond reasonable doubt. It is contended that the use of recent technical developments in the field of blast-timing systems (extended delays, accuracy in timing, etc.) would result in gas venting at the face of a solid blast equally well as that in a CCB section. Surely, a great deal of research work needs to be invested into this question. Such research effort, in the light of the experience with long delay timing that is reported further in this thesis, may provide a ground for seeking for a global review of the present legislation for coal mining in collieries.

The first step towards reducing coal fines in the industry is surely to do away with the undercutting of coal that generates an additional 5 % of fines even prior to the shooting.

The high fines generated with the current blasting technology has been associated with the recent striving for increased production, which has resulted in an increase in the drilled hole lengths underground. Currently, drilled blastholes are always more than 2 m in length. Despite the increase in drilled length, the charge mass per hole has been kept by the law to 800 g. The consequence has been that the explosive charge is always confined to the bottom of the blasthole, leaving a considerable collar of uncharged length at the coal face. Tamping of the explosive cartridges in the blasthole further compresses the charge to fill up the entire section area of the hole to the extent that intimate in-hole coupling is caused. Tight blast conditions are therefore created. The energy liberated by the explosive is concentrated at the bottom of the hole and therefore over-crushes the coal immediately around the charge, thus resulting in high levels of shock-wave-generated fines. Despite the undercutting, the inaccuracy and limited span (short time delays) of the timing system used (scatter time) contribute to fines generation by way of increasing burdens in front of an individual blastholes, which is detrimental to good breaking results.

7.3.2.2 The solutions suggested and their practicability

Konya, Britton and Lukovic (1987) recommended that the explosive charge in the blasthole be decoupled to reduce crushing and hence the generation of fines in blasting. Examining the fragmentation generated by decoupled charges, they found that, in any case, better fragmentation in blasting was produced by a decoupled charge rather than a fully coupled charge in the blasthole. They concluded that the first step in any blast optimization would be to determine the best coupling ratio for the particular medium, i.e. the ratio that maximizes both the energy release and the

fragmentation leading to reduced over- and undersize. Van Zyl (1993) applied this recommendation at Kleinkopje and reported having improved his blasting fragmentation results quite remarkably. Delaporte and Heizmann (1995) reported a reduction in coal fines of 2.6 percentage points brought about by the use of decoupled charges at the same mine. The successful results of this technique when used in the vertically drilled blastholes of surface-blasting operations prompted the present attempt to use the technique, for the purposes of this thesis, in the horizontal blastholes of underground coal-mining operations. Two alternative decoupling techniques were devised in this attempt to solve the problem of excessive fines generation during blasting. The techniques involved a short- and long-term plan of action, with different purposes, as follows:

1. to increase the length of the in-hole explosive charge, thereby increasing the decoupling of the charge in the blastholes as well, and
2. to develop a permitted detonating cord to allow taping of a number of spaced cartridges onto the cord in order to enhance energy distribution in the coal along the blasthole.

It was decided to investigate increasing the charge decoupling by spreading the same explosive mass (800 g) to cover a greater length of the blasthole.

The study considered the following different possibilities:

1. increasing the charge length in blastholes by reducing the charge diameter (use of smaller-diameter cartridges) while keeping the same charge mass (800 g),
2. using low density charges to allow packing of longer explosive cartridges with the same mass, and

3. Using shorter blastholes. (As the productivity would be affected negatively (reduction), this alternative would not be acceptable to the industry).

7.3.3 Summary of research results

7.3.3.1 Introduction

The blast trials were designed to test the ideas presented above with minimum disturbance of the normal mining operations underground. Special explosive products were ordered from the local manufacturers, DANTEX⁴ and SMX. The products supplied for the trials were essentially of the same composition, but different packaging was used. Small-diameter cartridges of 25 mm were specially produced for the project. They were used in standard drilled holes with diameters of 38 mm at Greenside and 40 mm at Tavistock. In addition, special arrangements were made to drill small-diameter holes of 36 mm. DANTEX supplied two product ranges, namely DANCOAL, also known as ENERCOAL, and SEAMEX, whereas SMX offered only MX8. A special test was undertaken using a 27 mm explosive cartridge of MX8 to determine the sensitivity trend in the dependence of coal fines generation on the in-hole VOD.

The reasoning for the proposed technique was that reducing the charge diameter would allow a longer charge in standard blastholes and therefore increase the decoupling between the coal and the explosive. This was expected to attenuate the primary crushing responsible for the reducible coal fines. In order to determine the most sensitive parameter, i.e. either the charge length extension or the increase in decoupling, the coupling ratio was further increased through the use of small-diameter blastholes. The results of these investigations are summarized in the following pages.

⁴ DANTEX : another company that manufactures and distributes explosives

7.3.3.2 Coal breaking results**A. Current blasting technology (Current: ©)**

	Greenside:		Tavistock:
	Seam 4	Seam 2	Seam 4
Coal strength (MPa)	12.6	21.9	15.0
Drilled hole diameter (mm)	38	40	40
Charge diameter (mm)	29	29	29
Charged length (m)	4 x 0.27	4 x 0.27	4 x 0.27
Charge density (g/cc)	1.13 - 1.2	1.12 - 1.13	1.12 - 1.13
VOD (unconfined in m/s)	3 000	5 000	5 000
Cut depth (m)	2.7	2.25	2.25
Number of holes per face	15	21	18
Charge per face (kg)	12	16.8	14.4
Energy content (MJ/kg)	2,443	3.19	3.19
Fragmentation: Cobble (#)	4	0	0
Nuts (%)	49.04	36.27	45.37
Peas (%)	28.72	38.62	30.84
Duff (%)	18.58	21.29	19.60
Super fines (%)	3.66	3.82	4.19
Total fines (%)	22.24	25.11	23.79
kg of explosive per % fines	0.54	0.67	0.61
(MJ per % fines)⁵	1.32	2.13	1.93

⁵ This criterion is used instead of the comminution index (ComI) because of the difficulty to assess the total mass of broken coal at the face, which is needed to determine the effective powder factor.

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B. Special techniques tested:

B.1 Use of reduced charge diameter (Small Diameter Charge: SDC)

	Greenside:	Tavistock:	
	Seam 4	Seam 2	Seam 4
Drilled hole diameter (mm)	38	40	40
Charge diameter (mm)	25	25	25
Charged length (m)	4 x 0.36	4 x 0.36	4 x 0.36
Charge density (g/cc)	1.13 - 1.2	1.12 - 1.13	1.12 - 1.13
VOD (unconfined in m/s)	3 000	5 000	5 000
Cut depth (m)	2.7	2.25	2.25
Number of holes per face	15	21	18
Charge per face (kg)	12.8	16.8	14.4
Energy content (MJ/kg)	2.44	3.19	3.19
Fragmentation: Cobble (#)	0	0	0
Nuts (%)	36.29	38.34	45.57
Peas (%)	37.17	39.18	32.09
Duff (%)	22.23	18.92	19.01
Super fines (%)	4.31	3.56	3.33
Total fines (%)	26.54	22.48	22.34
kg of explosive per % fines	0.48	0.74	0.64
MJ per % fines	1.10	2.38	2.06

B2 Use of reduced charge diameter in small-diameter blastholes*(Small Diameter Charge + Small Hole Diameter: SDC+SHD)*

	Greenside:		Tavistock:	
	Seam 4	Seam 2	Seam 4	
Drilled hole diameter (mm)	36	36	36	
Charge diameter (mm)	25	25	25	
Charged length (m)	4 x 0.36	4 x 0.36	4 x 0.36	
Charge density (g/cc)	1.13 - 1.2	1.12 - 1.13	1.12 - 1.13	
VOD (unconfined in m/s)	3 000	5 000	5 000	
Cut depth (m)	2.7	2.25	2.25	
Number of holes per face	16	21	18	
Charge per face (kg)	12.8	16.8	14.4	
Energy content (MJ/kg)	2.443	3.19	3.19	
Fragmentation: Cobble (#)	0	0	0	
Nuts (%)	36.75	40.30	43.62	
Peas (%)	35.19	32.92	32.52	
Duff (%)	22.96	20.88	20.01	
Super fines (%)	5.10	5.90	3.85	
Total fines (%)	28.06	26.78	23.86	
kg of explosive per % fines	0.46	0.63	0.60	
MJ per % fines	1.11	2.00	1.93	

B3. Use of low-density charge in small-diameter blastholes

(Low Density Explosive: LDE)

Greenside:

Seam 4

Drilled hole diameter (mm)	36
Charge diameter (mm)	25
Charged length (m)	4 x 0.36
Charge density (g/cc)	1.13 - 1.2
VOD (unconfined in m/s)	3 000
Cut depth (m)	2.7
Number of holes per face	15
Charge per face (kg)	12.8
Energy content (MJ/kg)	2.26
Fragmentation: Cobble (#)	1
Nuts (%)	38.75
Peas (%)	34.49
Duff (%)	21.68
Super fines (%)	5.08
Total fines (%)	26.76
kg of explosive per % fines	0.48
MJ per % fines	1.08

7.3.4 Discussion of results

7.3.4.1 Introduction

Coal blasting is far from being a pure science because of the variability of the working conditions. Taking into account the changes in strength at the coal faces, the number and timing of the drilled holes, and the human factor influencing the drilling precision, the discussion of the present results should be restricted to a comparison of the blasting work done in each seam.

The total fines generated (see Tables 7.6 to 7.9 and Figures 7.3 to 7.6) in the blasts was not used as the only conclusive criterion for evaluating the outputs. Two other evaluating indices were used as well. The first compared breakage from different blasting techniques in relation to the mass of explosives used in the blasts (see Table 7.9 and Figure 7.6). The second compared comminution in terms of the energy input in each blast (see Table 7.10 and Figure 7.8).

7.3.4.2 Comparison of fragmentation per coal seam

The overall coal fragmentation observed at the three test sites using the various techniques is presented in Tables 7.6 to 7.8 from which it may be noted that:

At Greenside (Table 7.6, Figure 7.3):

1. The total fines generated were lowest with the current blasting technique. However, in the blasted pile, an average of four over-size coal blocks requiring secondary breaking were observed.

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2. The fraction of saleable coal was optimum in the current technique which yielded 5.6 percentage point more exportable coal than in the second-best method (SDC).
3. The result showed that the suggested techniques tried failed to reduce the coal fines generated under the conditions found at this mine. Three possible reasons for the failure to get an improvement were brought forth:
 - The explosive product used (watergel) is characterized by a low level of energy release in the shock phase. The important coal-breaking phase is therefore the energy of gas, which unfortunately cannot be influenced by the suggested techniques.
 - The viscosity of the watergel is such that it easily flows out of the cartridge packing when tamped. Despite the smaller diameter, the tamping stick prevents the desired decoupling effects, which may attenuate the shock crushing and reduce the fines.
 - The soft coal in this seam favors the secondary crushing mechanism whose effects are random and uncontrollable.

Table 7.6 Observed fragmentation in Seam 4 at Greenside

Technique Fragmentation	⊙	SDC	SDC + SHD	LDE
Cobbles	4	0	0	1
Nuts	49.04	36.29	36.75	38.75
Peas	28.72	37.17	35.19	34.49
Duff	18.58	22.23	22.96	21.68
Super fines	3.66	4.31	5.10	5.08
Total fines	22.24	26.54	28.00	26.76

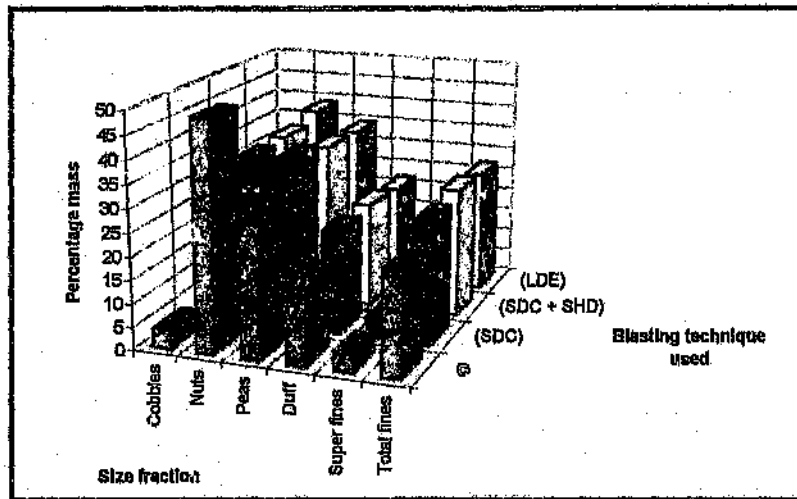


Figure 7.3 Comparative fragmentation in Seam 4 at Greenside

At Tavistock's Arthur Taylor Colliery in Seam 2 (Table 7.7, Figure 7.4)

1. The total fines generated were lowest with the suggested blasting technique involving a higher decoupling ratio obtained by a reduced charge diameter. In general, coal fragmentation in this seam was better than that observed at Greenside. Apart from the current technique, which showed comparatively poor results, both the new techniques improved the overall outputs. No oversize was produced.
2. The SDC technique yielded a perceptible improvement in the fraction of saleable coal (export), i.e. 2.63 percentage point more than with the current blasting technique.
3. This result shows that the technique can be recommended. The additional in-hole charge decoupling obtained with the SDC technique,

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enhanced by the low viscosity of the well-packed emulsion product (internal confinement), successfully reduced fines generation.

4. The increase in the in-hole coupling ratio created with the SDC + SHD technique did not meet the objective of this study. The saleable coal fraction dropped even below that of the current technique (from 74.89% to 73.22 %) with a resultant increase in the total fines (from current: 25.11 % to 26.78 %). This result showed that the in-hole coupling of the explosive charge is the most sensitive parameter determining the fines. The higher the coupling, the higher the fines generated under the conditions of this mine.

Table 7.7 Observed fragmentation in Seam 2 at ATC, Tavistock

Technique Fragmentation	©	SDC	SDC + SHD	LDE
Cobbles	0	0	0	-
Nuts	36.27	38.34	40.30	-
Peas	38.62	39.18	32.92	-
Duff	21.29	18.92	20.88	-
Super fines	3.82	3.56	5.90	-
Total fines	25.11	22.48	26.78	-

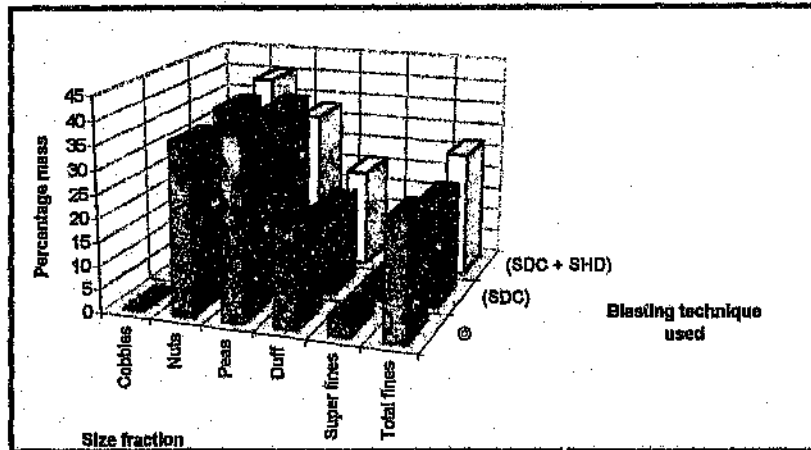


Figure 7.4 Comparative fragmentation in Seam 2 at ATC, Tavistock

At Tavistock's Arthur Taylor Colliery in Seam 4 (Table 7.8, Figure 7.5)

A trend similar to that in Seam 2 was observed.

1. The total fines generated were lowest for the suggested blasting technique with a higher decoupling ratio, i.e. the SDC technique. No oversize was produced. The fines per energy input in a coal face were better than those of Greenside in the same seam.
2. The fraction of saleable coal was increased by 1.45 percentage point, hence there was a fines reduction of the same proportion.
3. The increase in the in-hole coupling ratio (with SDC + SHD) confirmed the previous observation. The higher the coupling, the higher the fines generated in the conditions of this mine.

4. A comparison of the improvements achieved in these two seams using the same emulsion explosive (MX8) shows clearly that more benefit was derived in the hardest coal of Seam 2. The difference in the effects of the technique can be attributed to the different coal strengths. The harder the coal, the more sensitive it is to primary crushing since the higher the velocity of propagation of sonic waves and the higher the acoustic impedance. It is thus apparent that the shock waves is the prominent mechanism for fines generation in hard coals. Therefore the reduction of fines can be achieved through altering the ratio of the in-hole decoupling.

The characteristic impedance of coal from each seam was calculated. Table 7.9 depicts a clear trend of increasing characteristic impedance and the peak pressure transmitted to coal with the density which is proportional to the coal strength. This result confirms the trend observed in laboratory investigations (the harder the coal the higher the strain energy transmitted and the better the fragmentation, that is to say the lower the coal fines).

Table 7.8 Observed fragmentation in Seam 4 at ATC, Tavistock

Technique Fragmentation	©	SDC	SDC + SHD	LDE
Cobbles	0	0	0	-
Nuts	45.37	45.57	43.62	-
Peas	30.84	32.09	32.52	-
Duff	19.60	19.01	20.01	-
Super fines	4.19	3.33	3.85	-
Total fines	23.79	22.34	23.86	-

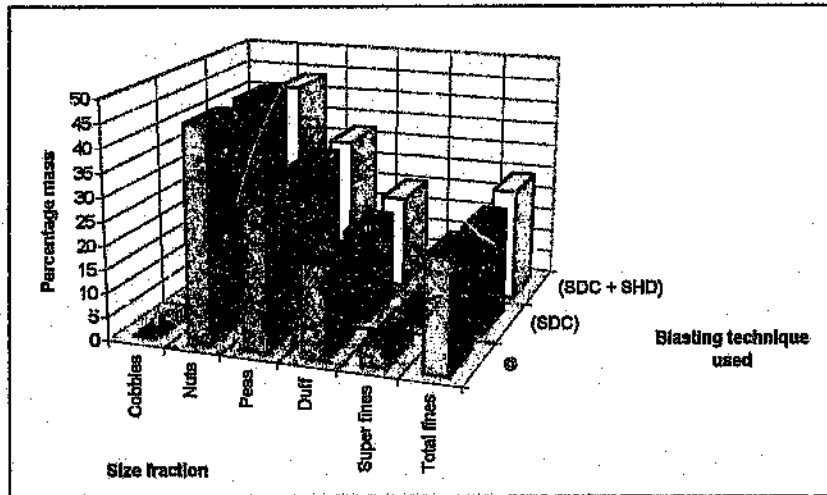


Figure 7.5 Comparative fragmentation in Seam 4 at ATC, Tavistock

Table 7.9 Characteristic impedance and pressure transmitted to coal

	ρ (kg/m ³)	D (m/s)	V_p (m/s)	P_d (GPa)	D (ρD) 10 ⁸	CI (ρV_p) 10 ⁸	P_{coal} (GPa)
Greenside Seam 2	1400	3000	2174	2.7	3.60	3.04	2.47
Greenside Seam 4	1450	3000	2465	2.7	3.60	3.57	2.69
Tavistock Seam 2	1580	5000	3162	7.1875	5.75	4.99	6.68
Tavistock Seam 2	1430	5000	2403	7.1875	5.75	3.44	5.38

ρ is the coal density, D is the velocity of detonation, V_p is the velocity of propagation of body waves, P_d is the detonation pressure ($P_d = 0.25 \rho_e D^2$),

ρ_a emulsion = 1150 kg/m³ and ρ_a Enercoal = 1200 kg/m³, DI is the detonation impedance ($DI = \rho_a D$), CI is the coal characteristic impedance ($CI = \rho V_p$) and P_{coal} is the pressure transmitted to coal that is calculated from the following formula (Sadwin, Okubo and Kennedy, 1963):

$$P_{coal} = \frac{2(CI)P_d}{(CI) + (DI)}$$

7.1

7.3.4.3 Evaluation of blasting techniques for reduced fines

generation

Three new blasting techniques were tried in addition to the current technology. Their respective coal fines generated were compared so as to identify the most appropriate one for each mine's conditions. Three evaluating criteria, as shown in the next pages, were used:

A. Comparison using the first criterion: Total fines generated

The results obtained are summarized in Table 7.10 and Figure 7.6. They show that the current technique works well in the soft coal of Seam 4 at Greenside. Otherwise, the new techniques result in an improvement in the total fines generation. The minimum fines were generated by the SDC technique in both Seam 4 and Seam 2 at Tavistock.

Table 7.10 Comparison of fines generation between the current and suggested techniques (Criterion: Total fines generated)

Technique used	Greenside Seam 4	Tavistock, ATC Seam 2	Tavistock, ATC Seam 4
©	22.24	25.11	23.79
SDC	26.54	22.48	22.34
SDC+SHD	28.06	26.78	23.86
LDE	26.76	-	-

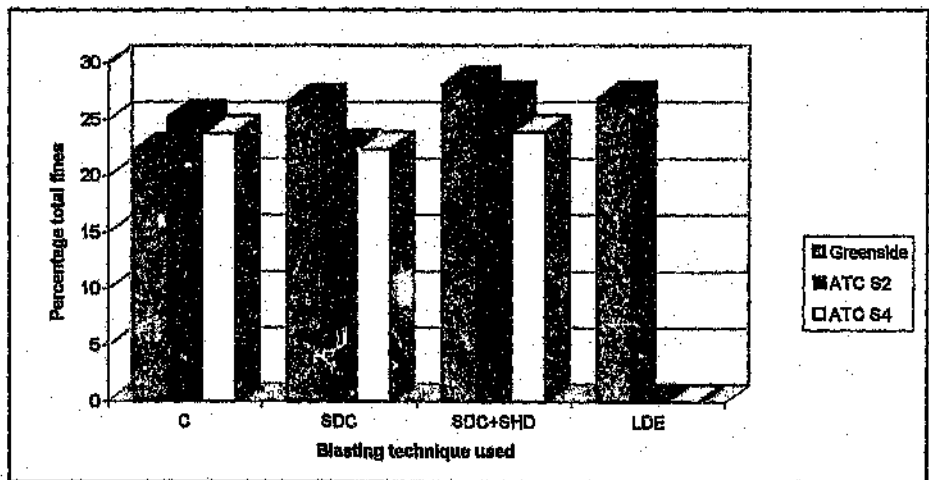


Figure 7.6 Comparison of total fines generated with different techniques used (Total fines considered)

B. Comparison using the second criterion: Mass of explosive used in kg per % fines generated

It is worth noting that higher explosive consumption per % fines means that more energy during the blast has been optimally used in coal breaking. The best blasting technique is that which uses more energy in fragmentation rather than in coal crushing. In this context, the higher the explosive consumption (kg) per % fines the better the blasting technique.

The results of this aspect of the investigation are summarized in Table 7.11 and Figure 7.7. They show that, despite the biased result observed at Greenside^a in Seam 4, it was preferable to reduce the charge per meter of drilled hole, thus extending the in-hole explosive charge length. The implication of doing so is that, for the same charge mass, a reduction in the confinement of the explosive charge in the hole is achieved and less concussion occurs. The extent of improved fragmentation obtained with this technique depends on both the coal strength and the shock energy available in the explosive product used for blasting. The interaction between these two parameters can be assessed from the in-hole VOD.

^a This result is attributed to the change in coal strength at the blasting faces. It is noted that the trial blasts using the current and suggested techniques compared here were conducted six months apart.

Table 7.11 Comparison of fines generation between the current and suggested techniques. (Criterion: Mass of explosive in kg per (%) fines)

Technique used	Greenside Seam 4	Tavistock, ATC Seam 2	Tavistock, ATC Seam 4
©	0.54	0.67	0.61
SDC	0.48	0.75	0.64
SDC+SHD	0.46	0.63	0.60
LDE	0.48	-	-

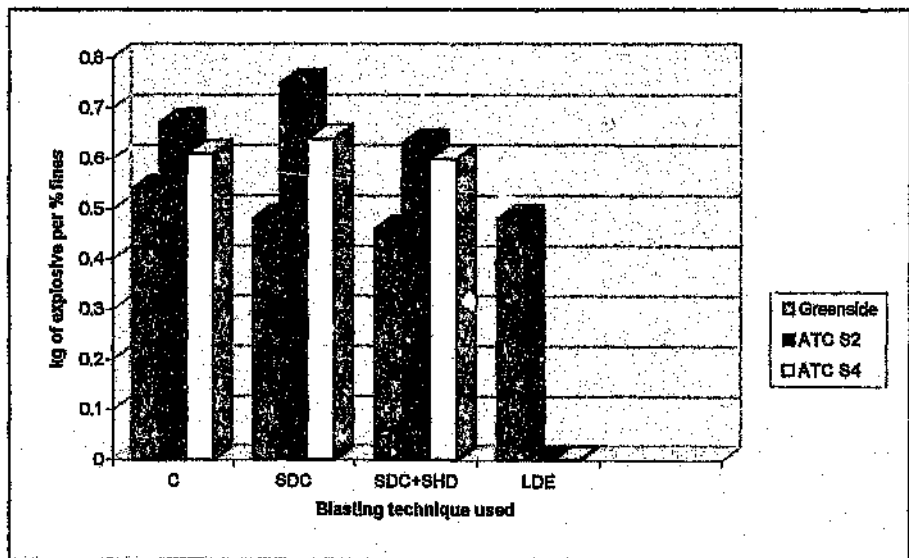


Figure 7.7 Comparison of fines generated with different techniques used
(Explosive mass input considered)

C. Comparison using the third criterion: Energy Input⁷ in MJ per % fines generated

It is important to remember that the higher the explosive energy input in the blast (MJ) per % fines, the better the blasting technique.

The result of this aspect of the investigation is depicted in the Table 7.12 and Figure 7.8. It was noted that both criteria that were used to assess the results of the blasts, namely the fines per mass of explosive used and the fines per input energy in the coal face pointed towards the benefits of the technique that involved the in-hole decoupling of the explosive charge.

Table 7.12 Comparison of fines generation between the current and suggested techniques, (Criterion: Energy input in MJ per % fines)

Technique used	Greenside Seam 4	Tavistock, ATC Seam 2	Tavistock, ATC Seam 4
©	1.32	2.13	1.93
SDC	1.10	2.38	2.06
SDC+SHD	1.11	2.00	1.93
LDE	1.08	-	-

⁷ Parameter associated with the comminution index (ComI), used because of the difficulty to assess the total mass of broken coal at the face, which is needed to determine the effective powder factor.

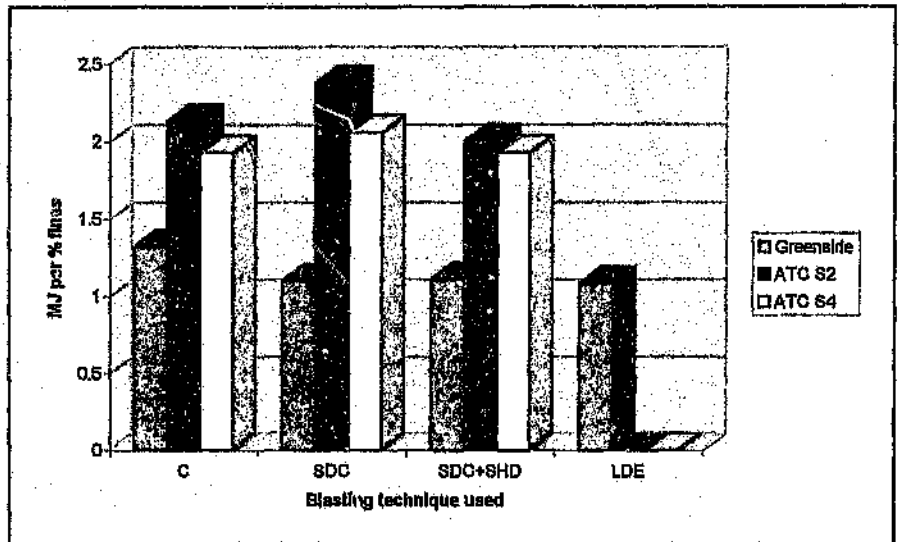


Figure 7.8 Comparison of fines generated with different techniques used (energy considered)

7.3.4.4 Influence of the velocity of detonation (VOD)

The VOD was monitored in some trial blasts with the aim of assessing its influence on coal fines generation. Five faces were instrumented. The VOD was monitored in blastholes using a VODR 1 apparatus provided by SMX. The measurements obtained from the blastholes are shown in Appendix F, however a summary of these results is given in Table 7.13.

In-hole explosive behaviour

The trend observed from the In-hole VOD recordings (see Figure 7.9) shows that the detonation of permitted emulsion explosive in blastholes is initiated by the boosted detonator with an instantaneous velocity well above the steady final VOD (between 4 650 to 6 500 m/s). In four out of

five observations made, the VOD decreased as the shock wave front moved away from the priming end towards the stemming. The only time the instantaneous VOD was seen to increase with time was when the highest VOD observed in the trials (5 450 m/s) was recorded, corresponding to the second lowest fines generation result. The trend of increasing VOD with time suggests a high rate of combustion of the explosive. This can be attributed either to the explosive's inner confinement (viscosity and/or packaging) or to its outer confinement (tight fit of the explosive in the blastholes). Among these two causes, the inner confinement is the most probable because a tightly fitted charge would increase the in-hole coupling and hence the transfer of shock energy to the coal resulting in more primary crushing as opposed to the inner confinement which evolves an action harmless to the coal. Its subsequent high combustion rate would be therefore conducive to less crushing.

It is thus noted that a high degree of explosive packing in the individual cartridges causes an increase in VOD which when associated to the in-hole decoupling technique leads to a reduction in coal fines generation. The high breaking energy developed in a detonation of such charge is well distributed through the air gap of decoupling into the coal burden, especially in the coal of the uncharged blasthole collar, hence favoring fragmentation.

It was also noted that less steep graphs of VOD were obtained from the blasts in the hard coal of Seam 2 (see Appendix F). The VOD graphs with a steeper slope angle were related to soft coal as shown in the typical representation of Figure 7.10. This observation is understandable since the rate of loss of the outer confinement as the detonation front progresses in the charge depends on the coal strength. This is to say that in general

the stronger coals, which are more brittle than the softer ones, do lose their ability to provide confinement to the shock waves front more quickly than the less brittle (soft) coals. The rapid loss of confinement in stronger and brittle coals is associated with plastic deformation and their easiness to propagate fractures and hence to sustain deeper damage. In soft coal, more of elasto-plastic deformation takes place causing only superficial damage. Therefore, the slower the loss of confinement of the in-hole shock waves' front, the higher the energy in the front and the lower the amount of fines generated.

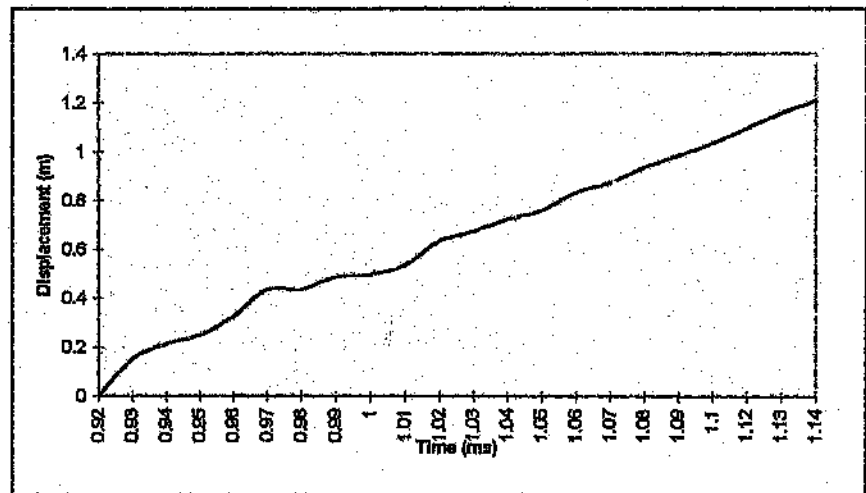


Figure 7.9 Typical trace record of in-hole VOD (emulsion MX8) in a 40 mm diameter blasthole

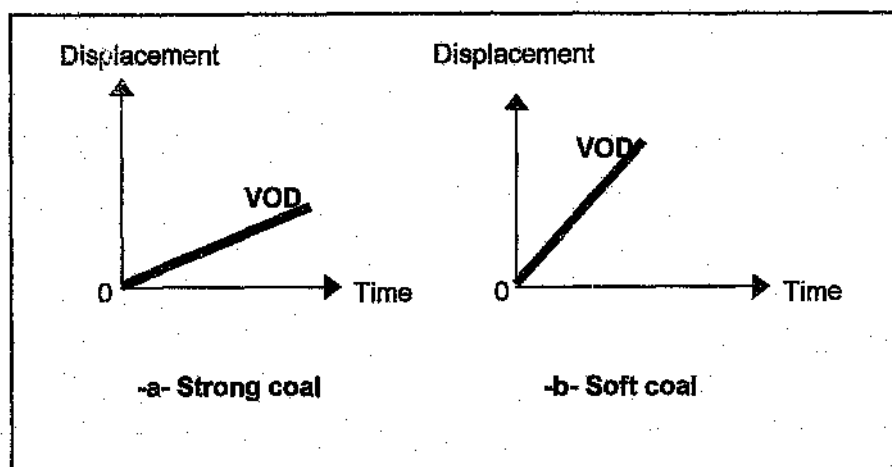


Figure 7.10 Observed trend of VOD in strong and soft coals

Table 7.13 Records of In-hole VOD and coal fines generation measured underground

Coal face #	Charge length (m)	VOD (m/s)	Charge diameter (mm)	In-hole charge diameter (mm)	Decoupling ratio	Crushing Index (%)
4#251	1.07	5 100	25	29.0	1.9	17.4
4#252	0.87	5 450	25	31.2	1.6	19.3
4#271	1.30	5 396	27	27.4	2.1	24.7
2#251	0.79	5 071	25	32.2	1.5	25.9
2#291	0.82	4 709	29	31.9	1.6	24.4

Note: Crushing Index (%) = % Coal fines observed

Dependence of coal fines on the in-hole VOD

The information collected from successful measurements of the in-hole VOD at five coal faces is plotted in the graph of Figure 7.11. The general trend in this figure shows that high VOD would generate fewer coal fines and better fragmentation. The interpretation of this result suggests that increasing the detonability of a permitted explosive of nearly 250 m/s will bring about 1 percentage point reduction in coal fines generated. This implies that increasing VOD may be a way of reducing both the number of large coal fragments and the total fines in blasted piles, as long as a proper degree of decoupling is applied.

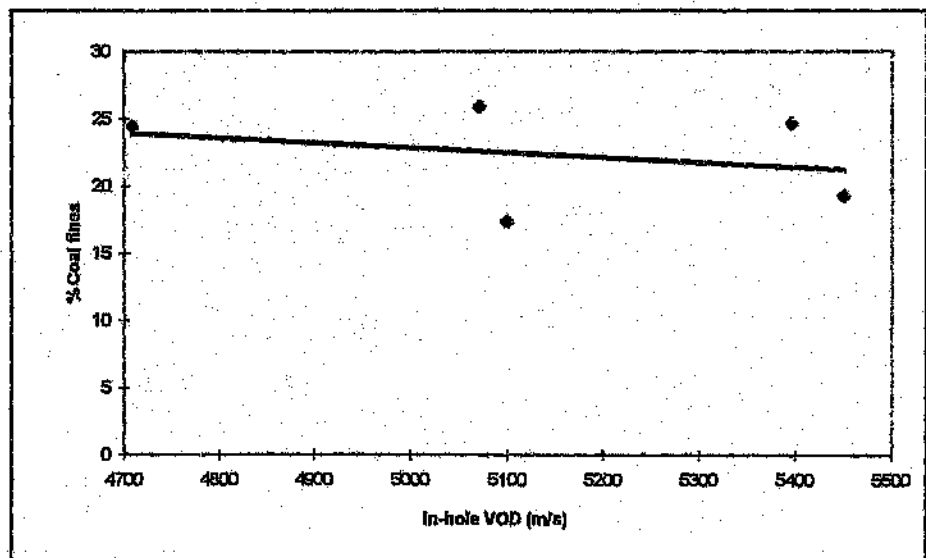


Figure 7.11 Observed dependence of the coal fines on the in-hole VOD

7.3.4.5 Impact of the charge confinement on coal fines generated

The distribution of the breaking energy in the coal burden is a very important parameter that determines the resulting breakage observed after a blast. Fragmentation results showed that increased charge length in the blasthole for the same mass of explosive (using a reduced charge diameter) resulted in better overall breakage of the coal. The number of large coal fragments was reduced considerably. However, this finding must be treated with caution as a 4.3 % increase in fines was observed at Greenside although a net reduction was obtained at Tavistock.

It was proved therefore that in-hole coupling of explosives favors the transfer of the dynamic stresses from a high VOD directly to the coal, hence producing more fines. A decoupled charge develops attenuated stresses that save coal from over-crushing and drives breaking energy into the coal at the blasthole's collar. However, the air gap in the charge needs to be optimal, as if too big, it may result in the dynamic desensitization of the explosive charge through the channel effect. In the present trials, no signs of such a phenomenon was observed, either in the blastholes monitored for VOD measurement or in the coal piles from the other blasts performed. The explosive charge was equally tamped in the blastholes.

It is worth mentioning that it was observed that the reduced-diameter charge resulted in an increase in coal fines in Seam 4 at Greenside and in decreased fines at Tavistock, compared with the current technique. When considering the measured coal strength, it appeared that the mechanisms determining coal fines vary profoundly from the soft coal of Greenside (compressive strength of 12.6 MPa) to the slightly harder coal of Tavistock in the same Seam 4 (compressive strength of 15 MPa). Badal (1995)

asserted that inherent discontinuities (micro and macro fractures) are responsible for the deformability of rock material submitted to dynamic stress loading. They determine the strength and the ability of a given rock to resist breaking and, in particular, crushing during blasting.

Figures 7.12 and 7.13 show that for a charge length, after tamping, of approximately 1m, the optimum in-hole confinement, in terms of fines generation, corresponds to a decoupling ratio of 1.8. This means that when using normal explosive cartridges of 29 mm in blastholes of 40 mm diameter, there is no need for tightly confining the charge in the hole bottom as slight tamping to increase the charge diameter to only 29.8 mm is require for an optimum blast.

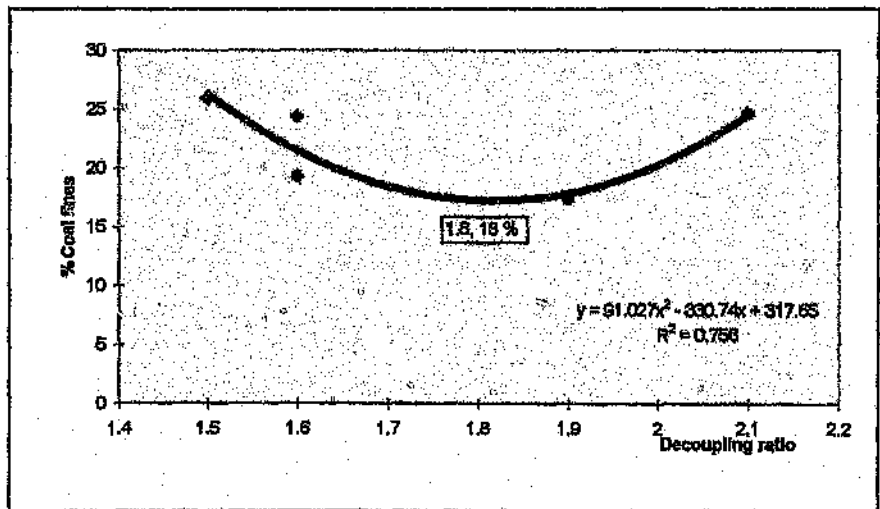


Figure 7.12 Dependence of coal fines generation on charge confinement

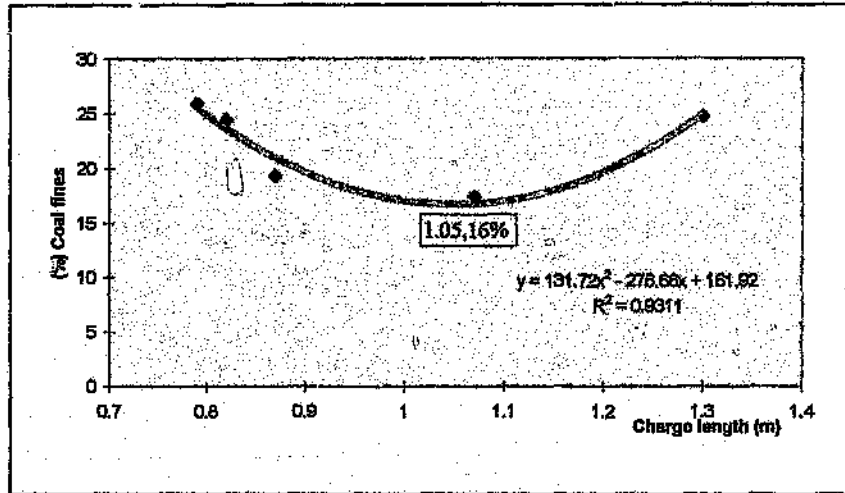


Figure 7.13 Dependence of coal fines generation on the in-hole charged length

Considering the optimum decoupling ratio of 1.8, it may be stated that the minimum drilling diameter to minimize coal fines generation when using the standard explosive cartridge diameter of 29 mm is 38.9 mm. It is therefore concluded that the problem of fines generation at Graesside where the standard drilling diameter of blastholes is 38 mm pertains partly to the incompatibility of the drilling and the explosive sizes.

7.4 Conclusion

The results obtained in these field trials indicated that the use of high-VOD explosives in decoupled blastholes is an effective way of reducing fines generation in coal blasting. This technique can be used for any coal strength (strong or brittle and soft). The degree of such decoupling is a site-specific parameter that needs to be determined from blast

investigations for optimization on site. Despite the indication of good results of fines reduction in hard rock beyond the decoupling ratio of 2 mentioned by Konya, Briton and Lukovic (1987), the present investigations undertaken in coal, a soft rock type, have indicated that the optimum in-hole decoupling ratio is 1.8. The same trend of decreasing fines with the increasing decoupling reported in the previous work referred to above has also been observed. The decoupling is found to improve fragmentation in coal output up to a ratio of 1.8 that corresponds to a minimal of 18 % of generated fines. Beyond this threshold further decoupling resulted in a drop in fragmentation (increased fines generation).

The results of this project can be summarized as follows:

1. The blasting techniques currently used in the local coal mining industry do have merit. However, they can be altered to optimize the outputs. Careful supervision of the charging operations underground is critical to the success of the blasting operation (i.e. to avoid current over-loading and over-filling of blastholes).
2. Depending on the coal strength and the nominal VOD of the explosive used, decoupling the explosive charge in the blastholes minimizes coal fines generation therefore improving the overall fragmentation. It was noted that, with the current practice, at Greenside a theoretical decoupling ratio of 1.7 yielded the lowest fines generation with some oversize. When this was increased further to 2.3, the fraction of fines was increased by 3.4 percentage point with no oversize produced. When the same technique was applied in smaller-diameter holes and the decoupling ratio was decreased to 2, the resulting fines showed the same increasing trend, rising to 5.8 percentage point. This funny

behaviour showed that in soft coal, the use of low-VOD explosive makes fines production insensitive to variations in shock energy. The mechanism of fines' creation in the coal of Seam 4 at Greenside is predominantly influenced by the interaction between the elastic waves, the gas energy effects and the inherent fractures in the coal. This is exacerbated by the apparent incompatibility of the drilling diameter used with the size of the explosive cartridge. That is the fundamental shortcoming found out by these investigations.

3. For coal blasting, permitted high-VOD explosives have more advantages than low-VOD ones. They can be used successfully in coal of any strength (hard and soft), provided a proper decoupling ratio is designed and implemented on site. The opinion that a big air gap between the explosive and the blasthole walls may increase the hazard of methane ignition still needs in-depth consideration in further work. However, no problem of in-hole dynamic desensitization or methane ignition was encountered during the present trials.
4. The viscosity of the explosive charge plays an important role in maintaining the in-hole decoupling. It also allows higher steady-state VOD through increased inner confinement. Such conditions render coal fines controllable and amenable to reduction by in-hole decoupling. For example, it was noted that the watergel explosive used at Greenside was flowing into the blastholes when squeezed by the tamping stick, thus further shortening the charged length and reducing the decoupling.

Chapter 7

5. This project has established that, any blasting work undertaken in the Witbank coalfield may be expected to generate a minimum 18 % fines.

8. OPTIMIZATION AND CONTROL OF FRAGMENTATION IN COAL

BLASTING

8.1 Introduction

This chapter intends to place into context the discussion of results obtained from the investigations into fines generation described in Chapters 6 and 7 with regard to the recent worldwide research effort which focuses on the control and optimization of fractures processes in industrial dynamic breaking operations. It also aims to outline the contribution of this on-site research and pinpoints the subsequent recommendable practical measures for optimizing coal blasting in the context of the local coal mining industry. The scientific background to the empirical results of this work is reviewed.

8.2 Decoupling and the function of coal fines generation

The investigations of this thesis have shown that the reduction of fines in blasting of the traditionally called soft rock materials, i.e. coal, may be achieved successfully by using a higher VOD-explosive product rather than a low-VOD product.

Previous work done by Johansson and Persson (1971), Atlas Powder Company (1987), Persson, Holmberg and Lee (1994) to name only a few, was used to secure a background to the results of the present thesis. These scientists ascertained that upon detonation of an exploding charge with a length to diameter ratio greater than 6:1 in a bottom-primed blasthole, there is a generation of shock waves whose created strain waves propagate in the material in a conical front coaxially with the blasthole. The consequence of the quasi-simultaneous axial and radial

expansions of the blasthole that are due to the products of the reaction created (shock waves) is that the pressure at the blasthole walls rises up quickly to its peak and then rapidly drops, decaying exponentially. The rapid decay is due to the great loss of energy through pulverization, crushing and/or displacement of material as well as the gas cooling. The axial expansion of the shock front is influenced by the magnitude of the shock that mainly depends on the composition of the explosive and on the VOD. However, the radial expansion is dependent not only on the composition of the explosive charge, but also on its in-hole geometry. In a blasthole that is fully packed with explosive (decoupling ratio equal to 1), the radial shock wave is directly transmitted to the coal. Consequently, significant damage to coal is possible. However, in a blasthole with a decoupling ratio greater than 1, the radial component of the shock wave will have decreased in magnitude due to the loss of energy at the interface between the explosive and air (i.e. flow into a different medium), and have expanded to fill the annular gap before impacting against the coal. Upon impact, the strength properties of the coal come into play. If the coal is stronger and generally brittle, the radial component of energy in the shock waves' front that is directed against the coal of the blasthole walls meets a strong opposition to penetration. However, because of its high magnitude, fractures are likely to be induced in the coal. Such fractures propagate much easier resulting in deeper damage into the coal (more coal fines generation). If a considerable portion of energy of the shock waves' front is associated with the damaging action described above, the axial energy component moving along the explosive charge may be slowed down resulting in a less steeper VOD trace being recorded in the blasthole as seen in Figure 7.10-a.

8.3 Stemming in coal blasting

In view of the considerable magnitude of the expanding ascendant component of the shock wave propagating in the axial direction, and considering the bottom-initiation system which is mainly used in local coal mining, both underground and in opencast, the stemming of blastholes is crucial in coal blasting. It plays a dual role: firstly, it is an excellent safety feature because it prevents the generated high-pressure and high-temperature detonation products from being propelled out of the blasthole, thereby limiting the hazard of the ignition of coal dust. Technically, it controls the retention time, which is the time period for which the stemming remains in place, closing the hole after the explosives column has completely detonated (Ladds, et al. 1992). The longer the retention time, the better the interaction between the explosive energy and the coal. Kabongo (1993) observed that an unstemmed blast resulted in 30 to 35 % loss of fragmentation. Earlier, Brinkman (1990) reported to have measured, in 40 mm blastholes, that one half of the total explosive energy is lost through gases escaping out of the blasthole collar when stemming is not used.

8.4 Fragmentation in industrial coal blasting

Measurement of fragmentation after a full-scale blast has always been a difficult task because of the size of industrial blasts and the rudimentary method of evaluation available. Even in small-scale models, the processes involved in the fragmentation observed around a blasthole in detonation are still a subject of contention between scientists. A definite lack of a clear fundamental understanding of the mechanism underlying the process has led to two opposing schools of thought in the past forty years. Johansson and Persson (1970), Comeau (1993), Persson, Holmberg and Lea (1994)

and others hold to the belief that fracture around a borehole in detonation is not initiated from the walls but, instead is seen to start from a point one or two radii inside the material. It is then from this locus that the initiated fractures propagate radially outwards from and inwards into the material. Allsman (1960), Kutter and Fairhurst (1971), Cunningham (1972), Shultz (1972), Rosmanith (1997) and others are of the opinion that is favored by the author of this thesis that consider fractures to initiate from the walls of the expanded blasthole. They initiate thus from the interface between the zone of fines, also termed crushed zone (Kutter and Fairhurst, 1971), and the solid material. The fractures therefore propagate radially outwards until the driving energy at their tips falls below the fracture stress intensity factor (K_{Ic}) characteristic of the material toughness. However, it worth noting that whichever theory is favored, there is a consensus on the mechanism underlying fragmentation. The latter is a result of the combined effect of the network of inherent fractures, those freshly induced during the dynamic phase of the detonation and the additional ones that may be created during the quasi-static loading phase of expanding gas products.

Fragmentation during mining is defined globally as covering three fractions or size groups: the over-, mid and undersize. The ultimate purpose of any blast is to produce mostly the desired mid-size. However, because of the lack of control over the breaking process, the three size groups are always present. Usually, industrial optimization of blasting is most concerned with the control of oversize. The parameters used at the blasting face are aimed mainly at the primary breaking which produces fragments to suit the aperture of the grizzly on the ore pass (production), waste rock pass (development) or the crusher. Even the commonly used blast-evaluation techniques, i.e. counting of boulders, photographing of the muck pile, visual analysis, assessment of the explosive consumption in secondary

blasting, and measuring of the shovel loading rate and bridging delays at the crusher, are all biased in favor of the oversize fragments. The current mathematical models used for the prediction of fragmentation in blasting also model the big fragments far better than they account for the undersize, particularly the fines. The equations of Gaudin-Meloy, Rosin Ramler, Kuz-Ram and the Sve De Fo (Holmberg, 1974) all have the same shortcoming of concentrating on the big size fragments, with the result that the predictions always differ from the post-blast realities. However, among the prediction models listed above, the Kuz-Ram model proposed by Cunningham (1983 and 1987) is reputed to provide predictions very close to reality (Singh and Pal Roy, 1993). This model takes into account a number of blasting parameters pertaining to the explosives properties, the geometry of the blast and the rock conditions, including the discontinuities. It is evident that the reason for its successful modelling of fragmentation lies in its ability to account for discontinuities. However, only the macro-fractures (joints) and, to a very limited extent, the meso-fractures can be comprehensively considered in the model. But, since the meso- and micro-fractures are the most sensitive to coal fines generation, it is therefore of little use in the prediction of comminution (fines generation).

8.4.1 Comminution model and reduction of coal fines

Udy (1994) pinpointed the challenge to the blasting engineer at any particular site as understanding the rock material he is dealing with. He commended Lilly (1986) for having provided an excellent technical and very practical method of classification of rock in terms of its ability to resist blasting and produce the required outputs, termed "blastability". Among the five rock mass parameters recommended for rock classification, the structural nature of the rock is the most important one determining the fragmentation and its economic implications in the blast. In a massive rock

with relatively few joints or in blocky ground, the fragmentation is entirely dependent on the interaction between the explosive energy input into the blast and the rock strength characteristics. Whereas in a finely jointed rock or one of a powdery nature, the fragmentation is of little concern as the explosive energy is only needed to disrupt the poor cohesion between the blocks and move the material from the blasting face.

It is evident that fragmentation in coal blasting would combine the effects of the primary breaking that result from the strength interaction between the explosive energy and the coal's resistance, and the wedging action that controls the liberation of the individual pre-existing blocks. These pre-existing blocks are delineated by the in-born fractures in the coal, as shown in Figure 8.1.

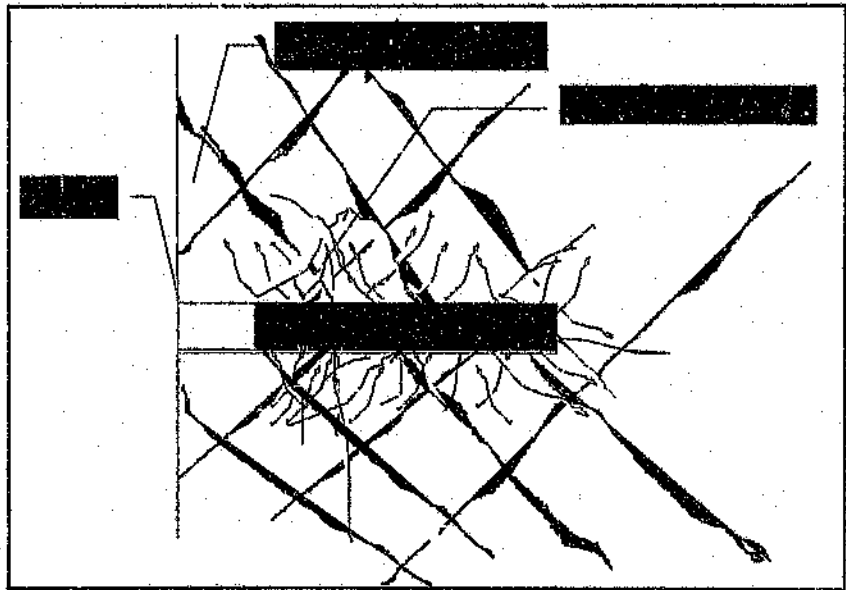


Figure 8.1 Two identifiable stages of fragmentation formation
(After Scott, Chitombo and Kleins (1993))

Stagg, Rhoil and Otterness (1989) confirmed, once again in another investigation of fragmentation in multiple-row blasting, that fragmentation involves two separate fractures mechanisms. These mechanisms establish respectively the coarse sizes, which appear to fit a Gaussian distribution, and the fine sizes, which include both the pre-blast fracture distribution and the radial fracturing around the borehole. They also observed that decoupling of the explosive charge in the blasthole considerably reduced the amount of finer sizes.

The investigations of the present thesis appear to confirm this theory of two distinct mechanisms (dynamic and quasi-static) occurring during the blast, resulting in the generation of the two size fractions, namely the under- and oversize. However, it is clear that there is a degree of overlap between these mechanisms to the extent that the total amount of undersize observed in the piles originates from both mechanisms. Singh (1995) explains the crushing mechanism that occurs in cut blasting as being caused partly by the gases (the quasi-static energy phase) which are generated after the explosion. He acknowledges the primary role that is played by the compressive strain waves which radiate outwardly from the explosive in detonation with a magnitude higher than that of the compressive breaking strength of the material. However, he underlines that the explosion gases compress the thin wedges of rock between adjacent fractures thereby creating additional fines. The overlap of the mechanistic effects creating the size fractions in blasting is not only obvious in the shock phase, which is traditionally known to be responsible for both the fines and the network of radial cracks around the blasthole. At the free face, the shock waves are reflected back into the coal burden, causing tensile fracture by slabbing or spalling which may result in large fragments of material breaking loose.

To sum up this discussion on the mechanistic insight gained from previous experience and the present investigations on the generation of coal fines in coal blasting, it is worth quoting Chiappetta (1994). He stated that: "Contrary to the popular belief, there is not a single parameter that contributes solely to coal damage during the blast, but rather a number of interrelated blast design and geological factors which make it unwise to expect coal fines generation to absolutely follow a unique trend".

It may be wise here to agree with Borovikov and Vanyagin (1995). They concluded their lengthy discussion on the modelling of rock fragmentation in blasting by stating that "modelling of the rock fragmentation in its entirety is a very difficult problem which cannot be solved practically, only an approximate and/or a partial modelling of the phenomenon is possible". This implies that the idea of a unique mathematical or physical model for coal fines generation in blasting is simply unobtainable. The empirical route is the only way to work out a prediction model for damage in coal during blasting. The investigation of the present thesis has therefore been a useful experience, which needs to be repeated in order to establish a database from which strong relationships between coal blasting and structural parameters can be derived and empirical formulae deduced (Kabongo and Darkwah, 1996).

8.4.2 Blast timing: the main factor in in-mine coal fines optimization

The technique of decoupling the explosive charge in the blasthole was successfully applied to attenuate the primary crushing and hence reduce the generation of coal fines. However, the keyword that represents the actual challenge to the local coal mining industry is 'productivity'. This means breaking the maximum amount of coal at the minimum possible working cost. Each individual mining operation, including the drilling and

blasting, must be conducted effectively so as to reinforce the trend towards optimisation. Among the factors that would influence productivity, the timing of the breaking of the individual coal burdens in front of each blasthole at the coal face is probably the most important.

The blast timing has a great influence on the end-results of the blast. In the mining of strong coal locally, blasting faces are prepared so as to break coal in sequence, involving a row-by-row breaking pattern. The standard blasting pattern used is depicted in Figure 8.2. In this figure the in-hole delay time used to initiate the detonation in the blastholes are designated by a delay number which corresponds to a range of time delay whose time value is given in Table 8.1.

$$\text{Time delay} = \text{Time value} \pm \text{Scatter time}$$

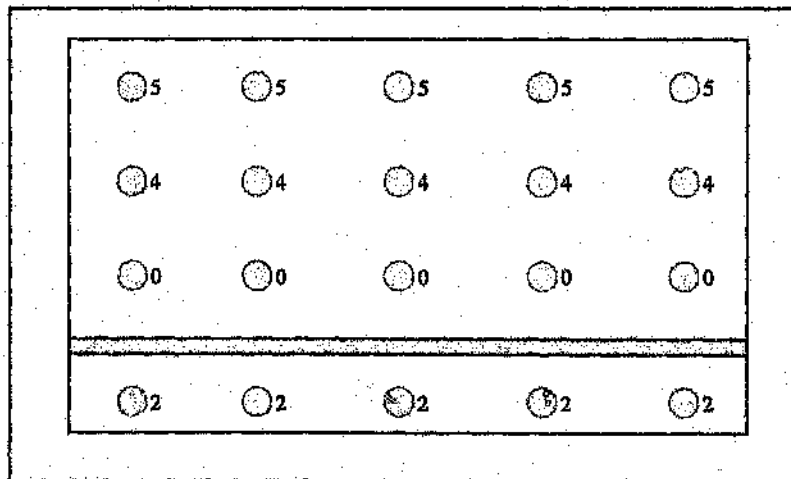


Figure 8.2 Conventional CCB pattern

Table 8.1 Delay numbers and corresponding delay time used in coal mines

Delay number	Delay time (ms)	Delay number	Delay time (ms)
0	0	6	200
1	30	7	250
2	55	8	300
3	80		
4	110		
5	150		
Permitted by law		Require special exemption	

Lubbe (1995), a former postgraduate student who worked under the author's supervision reported a considerable increase in broken coal output (28 % to 47 %) obtained in trial blasts which used time delays between rows of blastholes longer than those conventionally used. He compared the face advances obtained in trial blasts undertaken at Kriel Colliery. The trials used on one hand, the time delay system prescribed in the South African mining legislation (No. 0 to 5 in Table 8.1) and, on the other hand and under an exemption granted by the Inspectorate, a special range of extended delay detonators given in the Table 8.1. The results of this investigation are summarised in Table 8.2 and Figure 8.3.

Table 8.2 Summary of results obtained from investigation on the optimum inter-row timing in coal blasting at Kriel Colliery

Detonators	Inter-row delay (ms)	Advance per hole (%)	Broken tons per blast
CARRICKS	50,55,40 ¹	81	60.20
EDD	25	84	59.16
EDD	75	76	69.73
EDD	100	88	76.16
EDD	125	79	66.54
EDD	150	73	60.14
EXT. CARRICK	100	91	87.91
EDD	100 REV ²	86	70.24
EXT. CARRICK	100 REV	73	72.00
CARRICK	50,55,40 CHEV ³	77	55.36

It was observed that, on average, the inter-row delay of 100 ms produced the best advance with the optimum result being that obtained with the EXTENDED CARRICK of 100 ms delay which produced the highest call of broken coal per blast (see Figure 8.3). The change of the patterns of inter-row delay timing from 50–55–40 ms, traditionally used, to nearly 100 ms (see Figure 8.2) resulted in improved blast results (higher advance and hence higher tonnage). To quote the figures of results reported, the trial achieved an increase of between a third to a half of the usual output, without any change to the mining method. The optimised pattern that was recommended thereafter is shown in Figure 8.4.

1 The inter-row sequence that is currently used in local coal mining (in accordance to law).

2 REV stands for REVERSED firing as opposed to the normal or forward firing (see Appendix G).

3 CHEV stands for CHEVRON blasting tie-in pattern (see Appendix G).

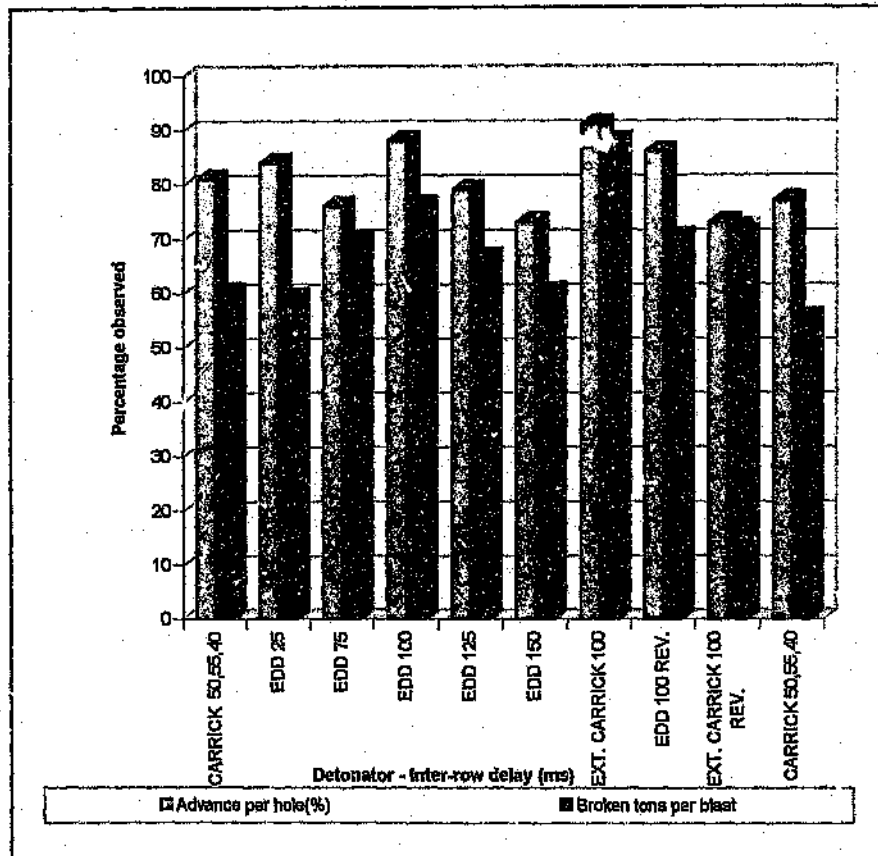


Figure 8.3 Advance and subsequent tons of broken coal per blast obtained using different inter-row timing. (After Lubbe and Kabongo, 1995)

In a recent project report on the follow-up work undertaken so as to provide additional support for the previous Investigation, Du Plessis (1997) working under the author supervision, confirmed the merit of the blasting

pattern using long inter-row delays. The new challenge assigned to this project was to investigate a number of delay pattern's configurations so as to reconcile the technical merit of longer delay timing (relief of coal burden and fragmentation) to the cost. A paper has been extracted from the project report and, as it has not yet been published, it is included in the Appendix G. It worth noting that the cost aspect of the work could not be reported in this paper as it is considered sensitive company information therefore confidential. However, this project showed that the combination of delays in a pattern such as that shown in Figure 8.5 produced greater advance, better fragmentation and consequently a high return as compared to the in-line pattern of Figure 8.4. The fragmentation results of this latter project supported those reported by Lubbe (1995) from the earlier investigation which compared the long 'delays' pattern to the legislated one. However, it is worth acknowledging that in both cases visual assessments were used and so, some quarters may consider this allegation to be less convincing.

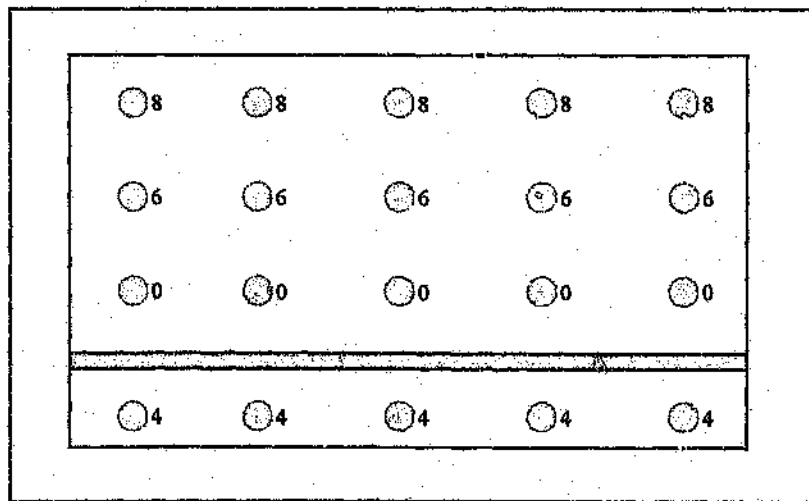


Figure 8.4 Optimised blasting pattern: (After Lubbe and Kabongo, 1995)

In fact, this result is corroborated by a consideration based on the respective comminution indices (ComI) in these blasts. The ComI of a full-scale blast is defined as the ratio of the powder factor to the percentage of coal fines generated. Considering that the coal output has been increased for the same explosive energy used (powder factor) in these blasts, it may be inferred that much more energy was used into breaking new surfaces in coal therefore improving the fragmentation. However, even if it is assumed that the same amount (by mass) of fines was generated in both blast situations, a high-output blast will result in a higher ComI. This means that a better energy distribution is achieved in such a blast.

It can therefore be concluded that the comminution index is a measure of the blasting efficiency. The greater the advance achieved in blasting, the higher the tonnage of coal broken and the higher the comminution index.

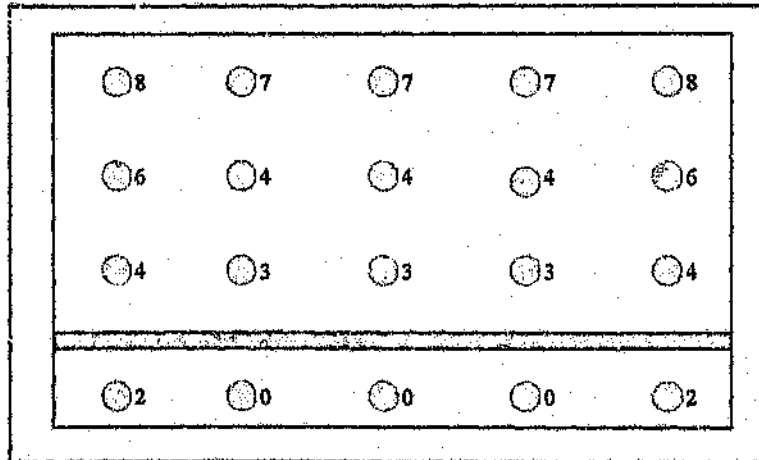


Figure 8.5 Recommended optimum blasting pattern for mining strong coal

8.5 Problems associated with coal blast optimization through

decoupling

8.5.1 Sympathetic detonation

Hagan (1979) pointed out that shorter inter-row delays cause high levels of fines generation, incomplete detachment of burdens, hindering of the fragments' displacement and tight muck piles. The optimum timing sequence in blasting is one which delays the initiation of the consecutive blastholes or rows of blastholes long enough to allow maximum interaction between the breaking stresses created in neighbouring holes. In coal cut blasting, it is important that the breaking of the next row of holes be delayed long enough to avoid 'sympathetic detonations' of explosive charges. Singh (1995) described this phenomenon as the premature initiation of a nearby charge by shock waves created by an earlier-firing charge. The explosive charges in the nearby rows are therefore set off nearly simultaneously, resulting in coal being frozen at the face because of partial flushing of the broken coal from each row of blastholes.

The probability of sympathetic detonation increases with decreasing inter-row burden, increasing sensitivity of the explosive charge and/or the initiator, increasing strain conductivity of the medium, increasing number of open fissures, and decreasing precision of the timing system (scatter time) when shorter delays are used. The use of a long inter-row delays sequence in coal cut blasting is therefore an effective way to avoid this problem which threatens productivity in collieries. However, the most definitive solution would be to use a high precision initiation system such as the electronic detonator. At this stage, the cost of the electronic technology does not justify its use in routine blasting work despite the fact that several trials already done have yielded exceptionally good results.

8.5.2 Dynamic desensitization

The shock waves generated in an earlier-firing blasthole may not only trigger a reaction in the neighbouring hole but also, and usually, may inhibit the detonability of its charge. Aler and Wieland (1991) maintain that when a blasthole charge is detonated during a delayed blast sequence in a coal mine, the shock waves it generates radiate in all directions through the coal and interact with adjacent blasthole charges that are timed for later detonation. The effects of such interaction are debilitating in nature, and can cause damage ranging from misfires to malfunctions, such as weak detonations or out-of-sequence detonations. These effects are termed 'cross-blastholes dynamic desensitization'. The charge desensitization is usually temporary, although it may be permanent in which case the detonation is completely halted (misfire). Otherwise, in the case of temporary desensitization, longer-delay timing always leads to a full recovery of the charge properties, even in explosives of the non-resilient type (Wieland, 1987).

Another type of explosive charge dynamic desensitization may take place within the blasthole due to the decoupling of the charge in the blasthole. This is known as the 'channel effect' (Johansson and Persson, 1970). When a decoupled charge is detonated in a dry hole, the shock wave generated in the air annulus always outruns the detonation wave in the explosive column thereby creating an air shock ahead of the detonation front. This air shock exerts a lateral pressure on the fresh part of the unexploded explosive that densifies and desensitizes the explosive column shortly before the arrival of the detonation wave. The probability of occurrence of the channel effect in blasting increases with the decoupling ratio. This has been the major challenge in the blast trials of this thesis that aimed at investigating the reduction of coal fines by using the charge

decoupling technique. The problem was overcome by adopting a charging configuration which allowed air gaps in the charge but obstructed the channel annular air gap, as shown in Figure 8.6. The volumetric decoupling ratio was therefore varied to achieve optimisation and the channel effect avoided. After each blast, the entire coal pile was inspected for signs of any explosive desensitization effect, but no such effect was detected.

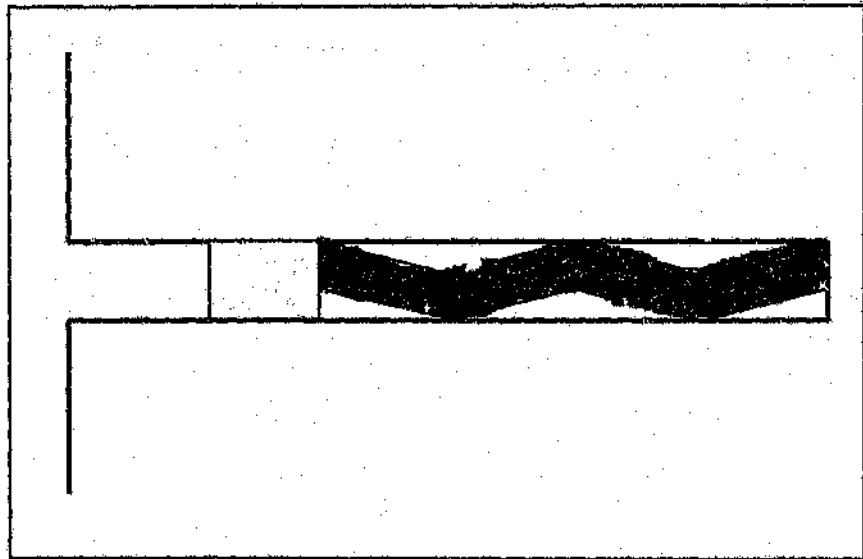


Figure 8.6 Ideal charging configuration to avoid the channel effect

8.6 Blasting technique for minimum coal fines

Blasting Off The Solid (BOTS) is an alternative method of coal blasting which does not require undercutting the coal face to provide for a second free face. The use of machine at the blasting face involves maintenance

costs and waste of time in the blasting cycle which may be saved if the quasi- conventional development type of blasting technique (BOTS) is considered.

In BOTS the second free face is created simultaneously by a number of blastholes particularly displayed and charged so as to initiate the breaking of a central core of coal to provide a free breaking space for the rest of the blast. Landman (1993) and Lubbe (1995) project this technique as a potential viable alternative to the undercutting. However, in a response to Cunningham (1991), Mainiero sounded rather an opposite opinion bringing forward the improved powder factor efficiency and the reduced confinement effects which ensures more safety underground when the undercutting is performed. The great merit of the CCB over BOTS lies on these key points of its merit. Mainiero stated as well that these advantages constitute the major reason why this method is the most preferred in the coal mining industry (flery mines) world-wide. Nevertheless, the debate to determine the best blasting technique for coal mining will still continue so long that research into BOTS is not intensified so as to prove beyond reasonable doubt the so-called safety deficiencies attributed to the method.

8.7 Conclusion

Rock breaking during blasting is the subject of much controversy by virtue of the complex interactions of phenomena characterised by a multitude of variable parameters. The results of the investigations reported in this chapter lead to conclude that the comminution index $ComI$, brought to the fore in this thesis, may be used to measure the blasting efficiencies. The greater the advance achieved in blasting, the higher the output of coal broken and the higher the comminution Index.

9 CONCLUSIONS

9.1 Review of this thesis

This thesis has established that coal fines generation during blasting is created by two categories of effect from the blast. They are: the impact of the dynamic shock waves on the coal of the blasthole's walls (primary crushing mechanism), and the interaction between the elastic waves and the expanding quasi-static gas-generated stresses with the inherent fracture network in the coal (secondary crushing mechanism).

The first category of fines generated may be reduced by a controlled blasting technology that uses the technique of decoupling the explosive charge in the blastholes. The investigations conducted on laboratory models showed that the extent of the zone of crushing which is established at the interface between the medium of the blasthole walls and the explosive may be altered by using this technique. The dynamic impact test for coal crushability proposed in this thesis provided an empirical formula for predicting the ability of coal to resist crushing in blasting. The equations derived from this study can be used, as first approximation into the prediction of the energy required for generating a required degree of comminution in blasting at the two pilot mines. However, it is expected that in the near future, this test will be performed on several other coals from different mines to expand the present database. Accumulated data will be used to derive more general comminution relationships.

In the field investigations, the coal fines generated in blasting were also observed to be sensitive to the decoupling technique. In both the qualitative report by Van Zyl (1993) and the quantitative assessment by

Delaporte and Heizmann (1995), the charge decoupling technique was confirmed to be effective in reducing coal fines in opencast blasting operations. The site investigations undertaken for the present thesis have extended the boundaries of knowledge further by evaluating the application of the technique in underground coal operations. It has been proved here that the technique can be used successfully in strong coals. The stronger the coal the higher the characteristic impedance and the more suited to fines reduction by decoupling. However the technique of decoupling may also be used in medium and soft coals, but it should be borne in mind that the degree of reduction achievable might be less perceptible.

The second category of fines generated during blasting is associated with the inherent fractures in the coal. These are not controllable due to the difficulty of determining the inherent fractures embedded in rock formations. At this stage of research in this field, the nature, number or the extent and location of inherent meso- and micro-fractures in a coal mass cannot be defined accurately. This thesis has demonstrated their influence on fines generation as evidenced by the strong correlation which was found to link the fracture frequency in diamond-drilled coal cores to the fines observed in coal piles after blasting. However, it is acknowledged that the relationship between the pre- and post-blast fracture situation established in this thesis may be of limited validity since it is based on the assumption that the discontinuities observed in the diamond-drilled cores are representative of the fracture network in the coal seam. Since the rock medium is rarely homogeneous and isotropic, a high degree of uncertainty is associated with blasting which hampers efforts to control the blast outputs.

The main concluding features of this thesis may be summarised as follows:

1. Strong correlation have been found to exist, on one hand between the strength properties of coal (the dynamic elastic modulus) and its comminution index (laboratory results) and, on the other hand, between the inherent fracture frequency in coal and the coal fines generated after blasting (site work results).
2. The higher the inherent fracture frequency in coal, the higher the quantity of fines generated. The relationship between these parameters has been identified as an exponential function for the coal tested in the three mined seams of the Witbank coalfield (Seam 1, 2 and 4).
3. No correlation was found to exist between the fracture frequencies (total fractures, i.e. inherent and induced) in the two orthogonal directions of a blasting face of coal, i.e. vertical and horizontal. This indicates a difference in their respective setting mechanisms.
4. The powder factor is the prime controlling factor of coal fines generation. However, the distribution of the explosive breaking energy in coal may be improved by changing the geometry of the charge in the blasthole. The technique of decoupling the explosive charge in the blasthole provides an effective way of reducing fines generation in both coal blasting in surface mining where, generally, large-diameter vertical blastholes are used, and in underground collieries where small-diameter horizontal blastholes are used.

5. It has come out clearly that despite the illusive trend of coal fines decreasing with the increasing coal strength characteristics observed in small size coal specimen in laboratory (size effect), the site work revealed the inverse trend. This result confirms once more that coal fines generation is site-specific. The variable amounts of fines observed in the field investigations described in this thesis appeared to indicate a trend of decreasing fines generation with increasing coal strength. On average, 22 % of blasted coal in the relatively soft coal of Seam 4 at Greenside was found to end up in the category of fines. In the medium-strength coal of Seam 4 at ATC, Tavistock, 24 % of blasted coal was observed to fall into this category. Finally, in the strong coal of Seam 2 at ATC, the fines proportion increased to 25 %. However, a score of between 8.4 and 21 percent (with an average of 12%) of blasted coal was observed to end up into fines in the surface mining operations at Kleinkopje.
6. It is inferred from the results above that opencast operations produce fewer fines than underground ones. This is attributed to the advantage of having two or three free faces widely available in the benching method used in opencast mining to break coal (see Figure 6.1a,b and c) which implies the use of reduced powder factors.
7. It is observed that the decoupling technique enables better reduction of coal fines in hard coals than it does in soft coals. The harder the coal the higher the velocity of propagation of sonic waves, the higher the characteristic impedance, the higher the strain energy transmitted to coal and the more it is sensitive to primary crushing.

8. In coal blasting, permitted high-VOD explosives have been shown to have more advantages than low-VOD ones. They have been used successfully in coal of different strengths (strong and medium) and they may also be used in soft coal, provided a proper decoupling ratio is designed and implemented on site. It remains true that a big air gap in the blasthole may increase the hazard of methane ignitions. However, this may be by-passed by using a longer charge column which, after tamping, will be deployed across the hole so as to avoid the formation of a continuous annular air gap around the charge. This will minimize the risk of in-hole dynamic desensitization or methane ignition as has been the case in the investigations of this thesis.
9. The viscosity of the explosive charge plays an important role in maintaining the in-hole decoupling. It also allows higher steady-state VOD through increased inner confinement. Such conditions render coal fines controllable and amenable to reduction by in-hole decoupling. For example, it was noted that the watergel explosive used at Greenside was flowing into the blastholes when squeezed by the tamping stick, thus further shortening the charged length and reducing the decoupling.
10. The comminution index (Comi) is a useful tool for the evaluation of blasting results. It measures the degree of breaking achieved in a blast. It combines together the information of blast fragmentation and the blasting efficiencies achieved in the blast.

9.2 Recommended future work

An engineering approach to the blasting problem is the key to improved productivity. There is a need for the characterization of coal at every mine. The data on the coal's strength properties and the fracture situation in the coal mass needs to be updated regularly as there cannot be a single solution to blasting (routine) in an insitu coal which changes its properties.

This thesis has shed some light on the relationships between coal fines and coal strength on one hand, and the fracture frequency on the other hand. In addition it has put forwards a technique for optimizing the generation of fines in blasting. However, additional, different problems which require further investigation for a better understanding of the complexity of the problem of fines generation have been brought to the fore. For instance, it has become imperative to acquire additional insight into the effectiveness of the VOD into the reduction of fines. This would entail testing a wide range of explosive products of different VODs under different coal conditions.

APPENDIX A

Results of laboratory investigation: Strength properties of coal

APPENDIX A

Results of laboratory investigation: Strength properties of coal

Table 1: Static strength properties measured on coal samples from
Greenside Colliery

GREENSIDE COLLIERY						
Compressive strength (MPa)						
	SEAM 2			SEAM 4		
	Top	Medium	Bottom	Top	Medium	Bottom
Parallel	-	36.02	35.95	-	11.48	13.57
Perpendicular	-	44.1	42.8	-	-	12.63
Elasticity Modulus at 0.5 x UCS (GPa)						
	Top	Medium	Bottom	Top	Medium	Bottom
Parallel	-	3.2	2.88	-	5.09	4.83
Perpendicular	-	5.08	4.84	-	-	3.5
Poisson's ratio at 0.5 x UCS						
	Top	Medium	Bottom	Top	Medium	Bottom
Parallel	-	0.23	0.27	-	1.85	0.21
Perpendicular	-	0.23	0.35	-	-	0.21

APPENDIX A

Results of laboratory investigation: Strength properties of coal

Table 2: Static strength properties measured on coal samples from
Tavistock, Arthur Taylor Colliery

TAVISTOCK, ARTHUR TAYLOR COLLIERY									
Compressive strength (MPa)									
	SEAM 1			SEAM 2			SEAM 4		
	Top	Med.	Bot.	Top	Med.	Bot.	Top	Med.	Bot.
Parallel	18.77	34.62	-	18.33	37.39	-	10.12	16.89	-
Perpendicular	23.1	38.2	-	21.89	-	-	13.14	20.07	-
Elasticity modulus at 0.5 x UCS (Gpa)									
	Top	Med.	Bot.	Top	Med.	Bot.	Top	Med.	Bot.
Parallel	4.03	11.58	-	4.64	13.88	-	2.2	3.75	-
Perpendicular	5.61	6.32	-	6.18	-	-	11.89	5.38	-
Poisson's ratio at 0.5 x UCS									
	Top	Med.	Bot.	Top	Med.	Bot.	Top	Med.	Bot.
Parallel	0.15	0.21	-	0.35	0.27	-	0.35	0.27	-
Perpendicular	0.18	0.27	-	0.31	-	-	0.31	0.23	-

Note: Med. = Median and Bot. = Bottom

APPENDIX B

Results of laboratory investigation: Comminution properties of coal

Dynamic strength properties and size distribution of coal tested

The data contained in this Appendix is also included in a computer disk attached to the thesis. The tables are presented in Lotus spreadsheets. The sample ID number contained in the first column of the tables identifies data of each table.

PROJECT: CHANGE IN CORE DYNAMIC LOADS TEST FOR COAL CIRCUMFERENCE										DATE: 04/23/87	
SAMPLE ID (EARTH INSTRUMENT #)		LOAD (K)	TIME (S)	DISP (IN)	CP (K/IN)	CR (K/IN)	DENSITY (G/CC)	MS	DISP (IN)	TIME (S)	
1.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
2.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
3.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
4.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
5.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
6.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
7.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
8.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
9.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
10.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
11.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
12.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
13.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
14.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
15.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
16.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
17.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
18.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
19.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
20.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
21.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
22.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
23.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
24.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
25.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
26.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.96	0.26
27.00	100.00	214.80	10.00	42.20	49.00	17.00	2594.92	1755.10	1417.21	41.9	

PROJECT CHARGES ON COMB CRUSHERABILITY													
OPTIMUM LOADING TEST FOR COMB CRUSHERABILITY													
NUMBER OF TESTS PER HOUR (%)	LOAD (%)	TON	1st (t)	2nd (t)	Op (t)	Cr (t)	DENSITY (g/cc)	Mo	2nd (g/cc)	3rd (g/cc)	4th (g/cc)	5th (g/cc)	6th (g/cc)
1.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
2.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
3.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
4.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
5.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
6.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
7.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
8.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
9.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
10.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
11.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
12.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
13.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
14.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
15.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
16.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
17.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
18.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
19.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26
20.00	100.00	214.80	10.00	42.00	77.00	2294.72	1715.10	1417.21	41.96	0.26	0.26	0.26	0.26

GRAVIMETRIC ANALYSIS										TOTAL										WATER (LOAD TIME CODE 4)		ROBUSTNESS		CHG (%)		COST (\$/KG)	
SAMPLES TO BE TESTED (kg/m3)										51=12.2	52=2.5	53=6.7	54=4.75	55=2.35	56=1.75	57=7.5	58=0.75	59=0.15	TOTAL	WATER	LOAD TIME CODE 4	ROBUSTNESS	CHG (%)	COST (\$/KG)			
CRIP	1	102.00	1500.00	WATER %	69.04	33.63	25.25	21.65	25.05	15.14	11.00	5.87	2.95	214.05	9.42	0.173151	1.932102	254.3	15.17034	536.3318							
				% WET WEIGHT	27.12	15.93	13.13	11.81	14.31	7.94	5.15	2.76	1.00	100.00													
				% DRY WEIGHT	27.12	42.24	55.97	67.58	67.58	86.43	91.51	93.8	100.00														
				% LOSS	72.88	57.76	44.05	32.42	18.12	10.53	5.85	2.88															
3.50	50.00	1200.00	WATER %	10.16	25.91	32.53	37.87	38.57	21.93	14.53	7.00	5.17	310.20	4.05	0.102948	52.73588	235.44	22.20558	249.507								
			% WET WEIGHT	4.85	18.48	20.21	15.65	18.25	10.53	5.73	3.33	2.68	100.00														
			% DRY WEIGHT	4.85	23.32	41.48	59.11	77.28	87.39	94.31	97.54	100.00															
			% LOSS	95.15	76.78	68.52	40.23	32.64	13.00	5.78	2.68																
10.00	105.00	1200.00	WATER %	18.28	43.88	31.03	35.51	41.05	22.17	13.37	7.44	5.93	254.50	4.25	0.237110	142.0533	335.46	38.30333	856.7135								
			% WET WEIGHT	8.81	19.57	16.78	14.12	16.78	8.43	5.23	2.97	2.79	100.00														
			% DRY WEIGHT	8.81	28.65	43.26	59.28	78.14	87.54	93.77	97.24	100.00															
			% LOSS	91.19	71.52	54.74	46.82	21.86	12.45	6.23	2.76																
13.00	107.00	900.00	WATER %	37.37	57.16	38.18	21.78	22.08	12.58	9.34	4.28	3.73	218.35	3.48	0.091482	53.0337	177.72	21.18124	127.8573								
			% WET WEIGHT	28.80	26.42	19.52	10.58	15.21	5.81	3.95	1.97	1.35	100.00														
			% DRY WEIGHT	28.80	52.89	65.92	70.50	87.11	92.40	96.77	98.74	100.00															
			% LOSS	71.20	47.10	34.08	23.10	12.85	7.58	3.23	1.25																
15.00	103.00	1500.00	WATER %	23.25	48.94	35.75	26.85	33.94	20.47	11.08	8.88	5.05	218.35	5.42	0.0514612	2.74345	594.3	0.829725	1318.1872								
			% WET WEIGHT	12.83	22.57	14.08	13.55	18.11	8.37	5.34	3.12	2.69	100.00														
			% DRY WEIGHT	10.83	35.00	47.54	53.74	70.55	86.22	94.17	97.32	100.00															
			% LOSS	89.17	64.92	52.46	33.96	21.15	11.76	5.59	2.87																
16.00	106.00	900.00	WATER %	100.10	23.40	19.78	17.33	21.55	12.50	12.88	4.78	2.97															

[illegible]

Appendices

Results of laboratory investigation: Strength properties of coal

APPENDIX B

24-07-03

UNIVERSITY OF THE VIRGINIA
DEPARTMENT OF CHEMISTRY
HARRISON HALL

NO. 5127 DEANARDY & CO. INC.

[illegible]

1. *Journal of the American Medical Association*, 1997; 278: 1039-1044.

DATE	TIME	FROM	TO	FLIGHT	CLASS	FARE	TAXES	TOTAL	STATUS
01/12/2018	07:00	NEW YORK	LOS ANGELES	DL 123	ECONOMY	120.00	15.00	135.00	OK
01/12/2018	08:00	LOS ANGELES	NEW YORK	DL 124	ECONOMY	120.00	15.00	135.00	OK
01/12/2018	09:00	NEW YORK	CHICAGO	DL 125	ECONOMY	110.00	14.00	124.00	OK
01/12/2018	10:00	CHICAGO	NEW YORK	DL 126	ECONOMY	110.00	14.00	124.00	OK
01/12/2018	11:00	NEW YORK	ATLANTA	DL 127	ECONOMY	100.00	13.00	113.00	OK
01/12/2018	12:00	ATLANTA	NEW YORK	DL 128	ECONOMY	100.00	13.00	113.00	OK
01/12/2018	13:00	NEW YORK	PHOENIX	DL 129	ECONOMY	90.00	12.00	102.00	OK
01/12/2018	14:00	PHOENIX	NEW YORK	DL 130	ECONOMY	90.00	12.00	102.00	OK
01/12/2018	15:00	NEW YORK	SEATTLE	DL 131	ECONOMY	80.00	11.00	91.00	OK
01/12/2018	16:00	SEATTLE	NEW YORK	DL 132	ECONOMY	80.00	11.00	91.00	OK
01/12/2018	17:00	NEW YORK	PORTLAND	DL 133	ECONOMY	70.00	10.00	80.00	OK
01/12/2018	18:00	PORTLAND	NEW YORK	DL 134	ECONOMY	70.00	10.00	80.00	OK
01/12/2018	19:00	NEW YORK	MINNEAPOLIS	DL 135	ECONOMY	60.00	9.00	69.00	OK
01/12/2018	20:00	MINNEAPOLIS	NEW YORK	DL 136	ECONOMY	60.00	9.00	69.00	OK
01/12/2018	21:00	NEW YORK	DETROIT	DL 137	ECONOMY	50.00	8.00	58.00	OK
01/12/2018	22:00	DETROIT	NEW YORK	DL 138	ECONOMY	50.00	8.00	58.00	OK
01/12/2018	23:00	NEW YORK	BUFFALO	DL 139	ECONOMY	40.00	7.00	47.00	OK
01/12/2018	00:00	BUFFALO	NEW YORK	DL 140	ECONOMY	40.00	7.00	47.00	OK
01/12/2018	01:00	NEW YORK	ALBANY	DL 141	ECONOMY	30.00	6.00	36.00	OK
01/12/2018	02:00	ALBANY	NEW YORK	DL 142	ECONOMY	30.00	6.00	36.00	OK
01/12/2018	03:00	NEW YORK	SARASOTA	DL 143	ECONOMY	20.00	5.00	25.00	OK
01/12/2018	04:00	SARASOTA	NEW YORK	DL 144	ECONOMY	20.00	5.00	25.00	OK
01/12/2018	05:00	NEW YORK	MIAMI	DL 145	ECONOMY	10.00	4.00	14.00	OK
01/12/2018	06:00	MIAMI	NEW YORK	DL 146	ECONOMY	10.00	4.00	14.00	OK

APPENDIX B

Results of laboratory investigation: Strength properties of coal

Appendix 11

UNIVERSITY OF MINNESOTA
DEPARTMENT OF MINERAL ENGINEERING
MASTERS TECHNOLOGY

CASUALTY IN TOTAL STRANDED
BRITISH & FOREIGN AIRCRAFT

項目	1970年	1971年	1972年	1973年	1974年	1975年	1976年	1977年	1978年	1979年	1980年	1981年	1982年	1983年	1984年	1985年	1986年	1987年	1988年	1989年	1990年	1991年	1992年	1993年	1994年	1995年	1996年	1997年	1998年	1999年	2000年	2001年	2002年	2003年	2004年	2005年	2006年	2007年	2008年	2009年	2010年	2011年	2012年	2013年	2014年	2015年	2016年	2017年	2018年	2019年	2020年	2021年	2022年	2023年	2024年	2025年	2026年	2027年	2028年	2029年	2030年	2031年	2032年	2033年	2034年	2035年	2036年	2037年	2038年	2039年	2040年	2041年	2042年	2043年	2044年	2045年	2046年	2047年	2048年	2049年	2050年	2051年	2052年	2053年	2054年	2055年	2056年	2057年	2058年	2059年	2060年	2061年	2062年	2063年	2064年	2065年	2066年	2067年	2068年	2069年	2070年	2071年	2072年	2073年	2074年	2075年	2076年	2077年	2078年	2079年	2080年	2081年	2082年	2083年	2084年	2085年	2086年	2087年	2088年	2089年	2090年	2091年	2092年	2093年	2094年	2095年	2096年	2097年	2098年	2099年	2100年	2101年	2102年	2103年	2104年	2105年	2106年	2107年	2108年	2109年	2110年	2111年	2112年	2113年	2114年	2115年	2116年	2117年	2118年	2119年	2120年	2121年	2122年	2123年	2124年	2125年	2126年	2127年	2128年	2129年	2130年	2131年	2132年	2133年	2134年	2135年	2136年	2137年	2138年	2139年	2140年	2141年	2142年	2143年	2144年	2145年	2146年	2147年	2148年	2149年	2150年	2151年	2152年	2153年	2154年	2155年	2156年	2157年	2158年	2159年	2160年	2161年	2162年	2163年	2164年	2165年	2166年	2167年	2168年	2169年	2170年	2171年	2172年	2173年	2174年	2175年	2176年	2177年	2178年	2179年	2180年	2181年	2182年	2183年	2184年	2185年	2186年	2187年	2188年	2189年	2190年	2191年	2192年	2193年	2194年	2195年	2196年	2197年	2198年	2199年	2200年	2201年	2202年	2203年	2204年	2205年	2206年	2207年	2208年	2209年	2210年	2211年	2212年	2213年	2214年	2215年	2216年	2217年	2218年	2219年	2220年	2221年	2222年	2223年	2224年	2225年	2226年	2227年	2228年	2229年	2230年	2231年	2232年	2233年	2234年	2235年	2236年	2237年	2238年	2239年	2240年	2241年	2242年	2243年	2244年	2245年	2246年	2247年	2248年	2249年	2250年	2251年	2252年	2253年	2254年	2255年	2256年	2257年	2258年	2259年	2260年	2261年	2262年	2263年	2264年	2265年	2266年	2267年	2268年	2269年	2270年	2271年	2272年	2273年	2274年	2275年	2276年	2277年	2278年	2279年	2280年	2281年	2282年	2283年	2284年	2285年	2286年	2287年	2288年	2289年	2290年	2291年	2292年	2293年	2294年	2295年	2296年	2297年	2298年	2299年	2300年	2301年	2302年	2303年	2304年	2305年	2306年	2307年	2308年	2309年	2310年	2311年	2312年	2313年	2314年	2315年	2316年	2317年	2318年	2319年	2320年	2321年	2322年	2323年	2324年	2325年	2326年	2327年	2328年	2329年	2330年	2331年	2332年	2333年	2334年	2335年	2336年	2337年	2338年	2339年	2340年	2341年	2342年	2343年	2344年	2345年	2346年	2347年	2348年	2349年	2350年	2351年	2352年	2353年	2354年	2355年	2356年	2357年	2358年	2359年	2360年	2361年	2362年	2363年	2364年	2365年	2366年	2367年	2368年	2369年	2370年	2371年	2372年	2373年	2374年	2375年	2376年	2377年	2378年	2379年	2380年	2381年	2382年	2383年	2384年	2385年	2386年	2387年	2388年	2389年	2390年	2391年	2392年	2393年	2394年	2395年	2396年	2397年	2398年	2399年	2400年	2401年	2402年	2403年	2404年	2405年	2406年	2407年	2408年	2409年	2410年	2411年	2412年	2413年	2414年	2415年	2416年	2417年	2418年	2419年	2420年	2421年	2422年	2423年	2424年	2425年	2426年	2427年	2428年	2429年	2430年	2431年	2432年	2433年	2434年	2435年	2436年	2437年	2438年	2439年	2440年	2441年	2442年	2443年	2444年	2445年	2446年	2447年	2448年	2449年	2450年	2451年	2452年	2453年	2454年	2455年	2456年	2457年	2458年	2459年	2460年	2461年	2462年	2463年	2464年	2465年	2466年	2467年	2468年	2469年	2470年	2471年	2472年	2473年	2474年	2475年	2476年	2477年	2478年	2479年	2480年	2481年	2482年	2483年	2484年	2485年	2486年	2487年	2488年	2489年	2490年	2491年	2492年	2493年	2494年	2495年	2496年	2497年	2498年	2499年	2500年	2501年	2502年	2503年	2504年	2505年	2506年	2507年	2508年	2509年	2510年	2511年	2512年	2513年	2514年	2515年	2516年	2517年	2518年	2519年	2520年	2521年	2522年	2523年	2524年	2525年	2526年	2527年	2528年	2529年	2530年	2531年	2532年	2533年	2534年	2535年	2536年	2537年	2538年	2539年	2540年	2541年	2542年	2543年	2544年	2545年	2546年	2547年	2548年	2549年	2550年	2551年	2552年	2553年	2554年	2555年	2556年	2557年	2558年	2559年	2560年	2561年	2562年	2563年	2564年	2565年	2566年	2567年	2568年	2569年	2570年	2571年	2572年	2573年	2574年	2575年	2576年	2577年	2578年	2579年	2580年	2581年	2582年	2583年	2584年	2585年	2586年	2587年	2588年	2589年	2590年	2591年	2592年	2593年	2594年	2595年	2596年	2597年	2598年	2599年	2600年	2601年	2602年	2603年	2604年	2605年	2606年	2607年	2608年	2609年	2610年	2611年	2612年	2613年	2614年	2615年	2616年	2617年	2618年	2619年	2620年	2621年	2622年	2623年	2624年	2625年	2626年	2627年	2628年	2629年	2630年	2631年	2632年	2633年	2634年	2635年	2636年	2637年	2638年	2639年	2640年	2641年	2642年	2643年	2644年	2645年	2646年	2647年	2648年	2649年	2650年	2651年	2652年	2653年	2654年	2655年	2656年	2657年	2658年	2659年	2660年	2661年	2662年	2663年	2664年	2665年	2666年	2667年	2668年	2669年	2670年	2671年	2672年	2673年	2674年	2675年	2676年	2677年	2678年	2679年	2680年	2681年	2682年	2683年	2684年	2685年	2686年	2687年	2688年	2689年	2690年	2691年	2692年	2693年	2694年	2695年	2696年	2697年	2698年	2699年	2700年	2701年	2702年	2703年	2704年	2705年	2706年	2707年	2708年	2709年	2710年	2711年	2712年	2713年	2714年	2715年	2716年	2717年	2718年	2719年	2720年	2721年	2722年	2723年	2724年	2725年	2726年	2727年	2728年	2729年	2730年	2731年	2732年	2733年	2734年	2735年	2736年	2737年	2738年	2739年	2740年	2741年	2742年	2743年	2744年	2745年	2746年	2747年	2748年	2749年	2750年	2751年	2752年	2753年	2754年	2755年	2756年	2757年	2758年	2759年	2760年	2761年	2762年	2763年	2764年	2765年	2766年	2767年	2768年	2769年	2770年	2771年	2772年	2773年	2774年	2775年	2776年	2777年	2778年	2779年	2780年	2781年	2782年	2783年	2784年	2785年	2786年	2787年	2788年	2789年	2790年	2791年	2792年	2793年	2794年	2795年	2796年	2797年	2798年	2799年	2800年	2801年	2802年	2803年	2804年	2805年	2806年	2807年	2808年	2809年	2810年	2811年	2812年	2813年	2814年	2815年	2816年	2817年	2818年	2819年	2820年	2821年	2822年	2823年	2824年	2825年	2826年	2827年	2828年	2829年	2830年	2831年	2832年	2833年	2834年	2835年	2836年	2837年	2838年	2839年	2840年	2841年	2842年	2843年	2844年	2845年	2846年	2847年	2848年	2849年	2850年	2851年	2852年	2853年	2854年	2855年	2856年	2857年	2858年	2859年	2860年	2861年	2862年	2863年	2864年	2865年	2866年	2867年	2868年	2869年	2870年	2871年	2872年	2873年	2874年	2875年	2876年	2877年	2878年	2879年	2880年	2881年	2882年	2883年	2884年	2885年	2886年	2887年	2888年	2889年	2890年	2891年	2892年	2893年	2894年	2895年	2896年	2897年	2898年	2899年	2900年	2901年	2902年	2903年	2904年	2905年	2906年	2907年	2908年	2909年	2910年	2911年	2912年	2913年	2914年	2915年	2916年	2917年	2918年	2919年	2920年	2921年	2922年	2923年	2924年	2925年	2926年	2927年	2928年	2929年	2930年	2931年	2932年	2933年	2934年	2935年	2936年	2937年	2938年	2939年	2940年	2941年	2942年	2943年	2944年	2945年	2946年	2947年	2948年	2949年	2950年	2951年	2952年	2953年	2954年	2955年	2956年	2957年	2958年	2959年	2960年	2961年	2962年	2963年	2964年	2965年	2966年	2967年	2968年	2969年	2970年	2971年	2972年	2973年	2974年	2975年	2976年	2977年	2978年	2979年	2980年	2981年	2982年	2983年	2984年	2985年	2986年	2987年	2988年	2989年	2990年	2991年	2992年	2993年	2994年	2995年	2996年	2997年	2998年	2999年	3000年	3001年	3002年	3003年	3004年	3005年	3006年	3007年	3008年	3009年	3010年	3011年	3012年	3013年	3014年	3015年	3016年	3017年	3018年	3019年	3020年	3021年	3022年	3023年	3024年	3025年	3026年	3027年	3028年	3029年	3030年	3031年	3032年	3033年	3034年	3035年	3036年	3037年	3038年	3039年	3040年	3041年	3042年	3043年	3044年	3045年	3046年	3047年	3048年	3049年	3050年	3051年	3052年	3053年	3054年	3055年	3056年	3057年	3058年	3059年	3060年	3061年	3062年	3063年	3064年	3065年	3066年	3067年	3068年	3069年	3070年	3071年	3072年	3073年	3074年	3075年	3076年	3077年	3078年	3079年	3080年	3081年	3082年	3083年	3084年	3085年	3086年	3087年	3088年	3089年	3090年	3091年	3092年	3093年	3094年	3095年	3096年	3097年	3098年	3099年	3100年	3101年	3102年	3103年	3104年	3105年	3106年	3107年	3108年	3109年	3110年	3111年	3112年	3113年	3114年	3115年	3116年	3117年	3118年	3119年	3120年	3121年	3122年	3123年	3124年	3125年	3126年	3127年	3128年	3129年	3130年	3131年	3132年	3133年	3134年	3135年	3136年	3137年	3138年	3139年	3140年	3141年	3142年	3143年	3144年	3145年	3146年	3147年	3148年	3149年	3150年	3151年	3152年	3153年	3154年	3155年	3156年	3157年	3158年	3159年	3160年	3161年	3162年	3163年	3164年	3165年	3166年	3167年	3168年	3169年	3170年	3171年	3172年	3173年	3174年	3175年	3176年	3177年	3178年	3179年	3180年	3181年	3182年	3183年	3184年	3185年	3186年	3187年	3188年	3189年	3190年	3191年	3192年	3193年	3194年	3195年	3196年	3197年	3198年	3199年	3200年	3201年	3202年	3203年	3204年	3205年	3206年	32
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ENVIRONMENTAL IMPACT

APR 2017	APR 2016	APR 2015	APR 2014	APR 2013	APR 2012	APR 2011	APR 2010	APR 2009	APR 2008	APR 2007	APR 2006	APR 2005	APR 2004	APR 2003	APR 2002	APR 2001	APR 2000	APR 1999	APR 1998	APR 1997	APR 1996	APR 1995	APR 1994	APR 1993	APR 1992	APR 1991	APR 1990	APR 1989	APR 1988	APR 1987	APR 1986	APR 1985	APR 1984	APR 1983	APR 1982	APR 1981	APR 1980	APR 1979	APR 1978	APR 1977	APR 1976	APR 1975	APR 1974	APR 1973	APR 1972	APR 1971	APR 1970	APR 1969	APR 1968	APR 1967	APR 1966	APR 1965	APR 1964	APR 1963	APR 1962	APR 1961	APR 1960	APR 1959	APR 1958	APR 1957	APR 1956	APR 1955	APR 1954	APR 1953	APR 1952	APR 1951	APR 1950	APR 1949	APR 1948	APR 1947	APR 1946	APR 1945	APR 1944	APR 1943	APR 1942	APR 1941	APR 1940	APR 1939	APR 1938	APR 1937	APR 1936	APR 1935	APR 1934	APR 1933	APR 1932	APR 1931	APR 1930	APR 1929	APR 1928	APR 1927	APR 1926	APR 1925	APR 1924	APR 1923	APR 1922	APR 1921	APR 1920	APR 1919	APR 1918	APR 1917	APR 1916	APR 1915	APR 1914	APR 1913	APR 1912	APR 1911	APR 1910	APR 1909	APR 1908	APR 1907	APR 1906	APR 1905	APR 1904	APR 1903	APR 1902	APR 1901	APR 1900	APR 1899	APR 1898	APR 1897	APR 1896	APR 1895	APR 1894	APR 1893	APR 1892	APR 1891	APR 1890	APR 1889	APR 1888	APR 1887	APR 1886	APR 1885	APR 1884	APR 1883	APR 1882	APR 1881	APR 1880	APR 1879	APR 1878	APR 1877	APR 1876	APR 1875	APR 1874	APR 1873	APR 1872	APR 1871	APR 1870	APR 1869	APR 1868	APR 1867	APR 1866	APR 1865	APR 1864	APR 1863	APR 1862	APR 1861	APR 1860	APR 1859	APR 1858	APR 1857	APR 1856	APR 1855	APR 1854	APR 1853	APR 1852	APR 1851	APR 1850	APR 1849	APR 1848	APR 1847	APR 1846	APR 1845	APR 1844	APR 1843	APR 1842	APR 1841	APR 1840	APR 1839	APR 1838	APR 1837	APR 1836	APR 1835	APR 1834	APR 1833	APR 1832	APR 1831	APR 1830	APR 1829	APR 1828	APR 1827	APR 1826	APR 1825	APR 1824	APR 1823	APR 1822	APR 1821	APR 1820	APR 1819	APR 1818	APR 1817	APR 1816	APR 1815	APR 1814	APR 1813	APR 1812	APR 1811	APR 1810	APR 1809	APR 1808	APR 1807	APR 1806	APR 1805	APR 1804	APR 1803	APR 1802	APR 1801	APR 1800	APR 1799	APR 1798	APR 1797	APR 1796	APR 1795	APR 1794	APR 1793	APR 1792	APR 1791	APR 1790	APR 1789	APR 1788	APR 1787	APR 1786	APR 1785	APR 1784	APR 1783	APR 1782	APR 1781	APR 1780	APR 1779	APR 1778	APR 1777	APR 1776	APR 1775	APR 1774	APR 1773	APR 1772	APR 1771	APR 1770	APR 1769	APR 1768	APR 1767	APR 1766	APR 1765	APR 1764	APR 1763	APR 1762	APR 1761	APR 1760	APR 1759	APR 1758	APR 1757	APR 1756	APR 1755	APR 1754	APR 1753	APR 1752	APR 1751	APR 1750	APR 1749	APR 1748	APR 1747	APR 1746	APR 1745	APR 1744	APR 1743	APR 1742	APR 1741	APR 1740	APR 1739	APR 1738	APR 1737	APR 1736	APR 1735	APR 1734	APR 1733	APR 1732	APR 1731	APR 1730	APR 1729	APR 1728	APR 1727	APR 1726	APR 1725	APR 1724	APR 1723	APR 1722	APR 1721	APR 1720	APR 1719	APR 1718	APR 1717	APR 1716	APR 1715	APR 1714	APR 1713	APR 1712	APR 1711	APR 1710	APR 1709	APR 1708	APR 1707	APR 1706	APR 1705	APR 1704	APR 1703	APR 1702	APR 1701	APR 1700	APR 1699	APR 1698	APR 1697	APR 1696	APR 1695	APR 1694	APR 1693	APR 1692	APR 1691	APR 1690	APR 1689	APR 1688	APR 1687	APR 1686	APR 1685	APR 1684	APR 1683	APR 1682	APR 1681	APR 1680	APR 1679	APR 1678	APR 1677	APR 1676	APR 1675	APR 1674	APR 1673	APR 1672	APR 1671	APR 1670	APR 1669	APR 1668	APR 1667	APR 1666	APR 1665	APR 1664	APR 1663	APR 1662	APR 1661	APR 1660	APR 1659	APR 1658	APR 1657	APR 1656	APR 1655	APR 1654	APR 1653	APR 1652	APR 1651	APR 1650	APR 1649	APR 1648	APR 1647	APR 1646	APR 1645	APR 1644	APR 1643	APR 1642	APR 1641	APR 1640	APR 1639	APR 1638	APR 1637	APR 1636	APR 1635	APR 1634	APR 1633	APR 1632	APR 1631	APR 1630	APR 1629	APR 1628	APR 1627	APR 1626	APR 1625	APR 1624	APR 1623	APR 1622	APR 1621	APR 1620	APR 1619	APR 1618	APR 1617	APR 1616	APR 1615	APR 1614	APR 1613	APR 1612	APR 1611	APR 1610	APR 1609	APR 1608	APR 1607	APR 1606	APR 1605	APR 1604	APR 1603	APR 1602	APR 1601	APR 1600	APR 1599	APR 1598	APR 1597	APR 1596	APR 1595	APR 1594	APR 1593	APR 1592	APR 1591	APR 1590	APR 1589	APR 1588	APR 1587	APR 1586	APR 1585	APR 1584	APR 1583	APR 1582	APR 1581	APR 1580	APR 1579	APR 1578	APR 1577	APR 1576	APR 1575	APR 1574	APR 1573	APR 1572	APR 1571	APR 1570	APR 1569	APR 1568	APR 1567	APR 1566	APR 1565	APR 1564	APR 1563	APR 1562	APR 1561	APR 1560	APR 1559	APR 1558	APR 1557	APR 1556	APR 1555	APR 1554	APR 1553	APR 1552	APR 1551	APR 1550	APR 1549	APR 1548	APR 1547	APR 1546	APR 1545	APR 1544	APR 1543	APR 1542	APR 1541	APR 1540	APR 1539	APR 1538	APR 1537	APR 1536	APR 1535	APR 1534	APR 1533	APR 1532	APR 1531	APR 1530	APR 1529	APR 1528	APR 1527	APR 1526	APR 1525	APR 1524	APR 1523	APR 1522	APR 1521	APR 1520	APR 1519	APR 1518	APR 1517	APR 1516	APR 1515	APR 1514	APR 1513	APR 1512	APR 1511	APR 1510	APR 1509	APR 1508	APR 1507	APR 1506	APR 1505	APR 1504	APR 1503	APR 1502	APR 1501	APR 1500	APR 1499	APR 1498	APR 1497	APR 1496	APR 1495	APR 1494	APR 1493	APR 1492	APR 1491	APR 1490	APR 1489	APR 1488	APR 1487	APR 1486	APR 1485	APR 1484	APR 1483	APR 1482	APR 1481	APR 1480	APR 1479	APR 1478	APR 1477	APR 1476	APR 1475	APR 1474	APR 1473	APR 1472	APR 1471	APR 1470	APR 1469	APR 1468	APR 1467	APR 1466	APR 1465	APR 1464	APR 1463	APR 1462	APR 1461	APR 1460	APR 1459	APR 1458	APR 1457	APR 1456	APR 1455	APR 1454	APR 1453	APR 1452	APR 1451	APR 1450	APR 1449	APR 1448	APR 1447	APR 1446	APR 1445	APR 1444	APR 1443	APR 1442	APR 1441	APR 1440	APR 1439	APR 1438	APR 1437	APR 1436	APR 1435	APR 1434	APR 1433	APR 1432	APR 1431	APR 1430	APR 1429	APR 1428	APR 1427	APR 1426	APR 1425	APR 1424	APR 1423	APR 1422	APR 1421	APR 1420	APR 1419	APR 1418	APR 1417	APR 1416	APR 1415	APR 1414	APR 1413	APR 1412	APR 1411	APR 1410	APR 1409	APR 1408	APR 1407	APR 1406	APR 1405	APR 1404	APR 1403	APR 1402	APR 1401	APR 1400	APR 1399	APR 1398	APR 1397	APR 1396	APR 1395	APR 1394	APR 1393	APR 1392	APR 1391	APR 1390	APR 1389	APR 1388	APR 1387	APR 1386	APR 1385	APR 1384	APR 1383	APR 1382	APR 1381	APR 1380	APR 1379	APR 1378	APR 1377	APR 1376	APR 1375	APR 1374	APR 1373	APR 1372	APR 1371	APR 1370	APR 1369	APR 1368	APR 1367	APR 1366	APR 1365	APR 1364	APR 1363	APR 1362	APR 1361	APR 1360	APR 1359	APR 1358	APR 1357	APR 1356	APR 1355	APR 1354	APR 1353	APR 1352	APR 1351	APR 1350	APR 1349	APR 1348	APR 1347	APR 1346	APR 1345	APR 1344	APR 1343	APR 1342	APR 1341	APR 1340	APR 1339	APR 1338	APR 1337	APR 1336	APR 1335	APR 1334	APR 1333	APR 1332	APR 1331	APR 1330	APR 1329	APR 1328	APR 1327	APR 1326	APR 1325	APR 1324	APR 1323	APR 1322	APR 1321	APR 1320	APR 1319	APR 1318	APR 1317	APR 1316	APR 1315	APR 1314	APR 1313	APR 1312	APR 1311	APR 1310	APR 1309	APR 1308	APR 1307	APR 1306	APR 1305	APR 1304	APR 1303	APR 1302	APR 1301	APR 1300	APR 1299	APR 1298	APR 1297	APR 1296	APR 1295	APR 1294	APR 1293	APR 1292	APR 1291	APR 1290	APR 1289	APR 1288	APR 1287	APR 1286	APR 1285	APR 1284	APR 1283	APR 1282	APR 1281	APR 1280	APR 1279	APR 1278	APR 1277	APR 1276	APR 1275	APR 1274	APR 1273	APR 1272	APR 1271	APR 1270	APR 1269	APR 1268	APR 1267	APR 1266	APR 1265	APR 1264	APR 1263	APR 1262	APR 1261	APR 1260	APR 1259	APR 1258	APR 1257	APR 1256	APR 1255	APR 1254	APR 1253	APR 1252	APR 1251	APR 1250	APR 1249	APR 1248	APR 1247	APR 1246	APR 1245	APR 1244	APR 1243	APR 1242	APR 1241	APR 1240	APR 1239	APR 1238	APR 1237	APR 1236	APR 1235	APR 1234	APR 1233	APR 1232	APR 1231	APR 1230	APR 1229	APR 1228	APR 1227	APR 1226	APR 1225	APR 1224	APR 1223	APR 1222	APR 1221	APR 1220	APR 1219	APR 1218	APR 1217	APR 1216	APR 1215	APR 1214	APR 1213	APR 1212	APR 1211	APR 1210	APR 1209	APR 1208	APR 1207	APR 1206	APR 1205	APR 1204	APR 1203	APR 1202	APR 1201	APR 1200	APR 1199	APR 1198	APR 1197	APR 1196	APR 1195	APR 1194	APR 1193	APR 1192	APR 1191	APR 1190	APR 1189	APR 1188	APR 1187	APR 1186	APR 1185	APR 1184	APR 1183	APR 1182	APR 1181	APR 1180	APR 1179	APR 1178	APR 1177	APR 1176	APR 1175	APR 1174	APR 1173	APR 1172	APR 1171	APR 1170	APR 1169	APR 1168	APR 1167	APR 1166	APR 1165	APR 1164	APR 1163	APR 1162	APR 1161	APR 1160	APR 1159	APR 1158	APR 1157	APR 1156	APR 1155	APR 1154	APR 1153	APR 1152	APR 1151	APR 1150	APR 1149	APR 1148	APR 1147	APR 1146	APR 1145	APR 1144	APR 1143	APR 1142	APR 1141	APR 1140	APR 1139	APR 1138	APR 1137	APR 1136	APR 1135	APR 1134	APR 1133	APR 1132	APR 1131	APR 1130	APR 1129	APR 1128	APR 1127	APR 1126	APR 1125	APR 1124	APR 1123	APR 1122	APR 1121	APR 1120	APR 1119	APR 1118	APR 1117	APR 1116	APR 1115	APR 1114	APR 1113	APR 1112	APR 1111	APR 1110	APR 1109	APR 1108	APR 1107	APR 1106	APR 1105	APR 1104	APR 1103	APR 1102	APR 1101	APR 1100	APR 1099	APR 1098	APR 1097	APR 1096	APR 1095	APR 1094	APR 1093	APR 1092	APR 1091	APR 1090	APR 1089	APR 1088	APR 1087	APR 1086	APR 1085	APR 1084	APR 1083	APR 1082	APR 1081	APR 1080	APR 1079	APR 1078	APR 1077	APR 1076	APR 1075	APR 1074	APR 1073	APR 1072	APR 1071	APR 1070	APR 1069	APR 1068	APR 1067	APR 1066	APR 1065	APR 1064	APR 1063	APR 1062	APR 1061	APR 1060	APR 1059	APR 1058	APR 1057	APR 1056	APR 1055	APR 1054	APR 1053	APR 1052	APR 1051	APR 1050	APR 1049	APR 1048	APR 1047	APR 1046	APR 1045	APR 1044	APR 1043	APR 1042	APR 1041	APR 1040	APR 1039	APR 1038	APR 1037	APR 1036	APR 1035	APR 1034	APR 1033	APR 1032	APR 1031	APR 1030	APR 1029	APR 1028	APR 1027	APR 1026	APR 1025	APR 1024	APR 1023	APR 1022	APR 1021	APR 1020	APR 1019	APR 1018	APR 1017	APR 1016	APR 1015	APR 1014	APR 1013	APR 1012	APR 1011	APR 1010	APR 1009	APR 1008	APR 1007	APR 1006	APR 1005	APR 1004	APR 1003	APR 1002	APR 1001	APR 1000	APR 999	APR 998	APR 997	APR 996	APR 995	APR 994	APR 993	APR 992	APR 991	APR 990	APR 989	APR 988	APR 987	APR 986	APR 985
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APPENDIX B

Appendix 2

UNIVERSITY OF THE WITWATERSRAND
DEPARTMENT OF MICRO ELECTRONICS
BUILDING TECHNOLOGY

STUDY: CHANGE IN COAL
DEMAND FOR COAL LIBRARIETY

[illegible]

Year	1980	1981	1982	1983	1984	1985
1980	1.00	1.00	1.00	1.00	1.00	1.00
1981	1.00	1.00	1.00	1.00	1.00	1.00
1982	1.00	1.00	1.00	1.00	1.00	1.00
1983	1.00	1.00	1.00	1.00	1.00	1.00
1984	1.00	1.00	1.00	1.00	1.00	1.00
1985	1.00	1.00	1.00	1.00	1.00	1.00

CRACKING RATE: ANALYSIS

DATE	TIME	FROM	TO	FLIGHT	CLASS	FARE	TAXES	FEES	TOTAL	STATUS	REMARKS
2023	10:00	NEW YORK	LOS ANGELES	AA100	Y	120.00	15.00	20.00	155.00	OK	
2023	11:00	LOS ANGELES	NEW YORK	AA101	Y	120.00	15.00	20.00	155.00	OK	
2023	12:00	NEW YORK	LOS ANGELES	AA102	Y	120.00	15.00	20.00	155.00	OK	
2023	13:00	LOS ANGELES	NEW YORK	AA103	Y	120.00	15.00	20.00	155.00	OK	
2023	14:00	NEW YORK	LOS ANGELES	AA104	Y	120.00	15.00	20.00	155.00	OK	
2023	15:00	LOS ANGELES	NEW YORK	AA105	Y	120.00	15.00	20.00	155.00	OK	
2023	16:00	NEW YORK	LOS ANGELES	AA106	Y	120.00	15.00	20.00	155.00	OK	
2023	17:00	LOS ANGELES	NEW YORK	AA107	Y	120.00	15.00	20.00	155.00	OK	
2023	18:00	NEW YORK	LOS ANGELES	AA108	Y	120.00	15.00	20.00	155.00	OK	
2023	19:00	LOS ANGELES	NEW YORK	AA109	Y	120.00	15.00	20.00	155.00	OK	
2023	20:00	NEW YORK	LOS ANGELES	AA110	Y	120.00	15.00	20.00	155.00	OK	
2023	21:00	LOS ANGELES	NEW YORK	AA111	Y	120.00	15.00	20.00	155.00	OK	
2023	22:00	NEW YORK	LOS ANGELES	AA112	Y	120.00	15.00	20.00	155.00	OK	
2023	23:00	LOS ANGELES	NEW YORK	AA113	Y	120.00	15.00	20.00	155.00	OK	
2023	00:00	NEW YORK	LOS ANGELES	AA114	Y	120.00	15.00	20.00	155.00	OK	
2023	01:00	LOS ANGELES	NEW YORK	AA115	Y	120.00	15.00	20.00	155.00	OK	
2023	02:00	NEW YORK	LOS ANGELES	AA116	Y	120.00	15.00	20.00	155.00	OK	
2023	03:00	LOS ANGELES	NEW YORK	AA117	Y	120.00	15.00	20.00	155.00	OK	
2023	04:00	NEW YORK	LOS ANGELES	AA118	Y	120.00	15.00	20.00	155.00	OK	
2023	05:00	LOS ANGELES	NEW YORK	AA119	Y	120.00	15.00	20.00	155.00	OK	
2023	06:00	NEW YORK	LOS ANGELES	AA120	Y	120.00	15.00	20.00	155.00	OK	
2023	07:00	LOS ANGELES	NEW YORK	AA121	Y	120.00	15.00	20.00	155.00	OK	
2023	08:00	NEW YORK	LOS ANGELES	AA122	Y	120.00	15.00	20.00	155.00	OK	
2023	09:00	LOS ANGELES	NEW YORK	AA123	Y	120.00	15.00	20.00	155.00	OK	
2023	10:00	NEW YORK	LOS ANGELES	AA124	Y	120.00	15.00	20.00	155.00	OK	
2023	11:00	LOS ANGELES	NEW YORK	AA125	Y	120.00	15.00	20.00	155.00	OK	
2023	12:00	NEW YORK	LOS ANGELES	AA126	Y	120.00	15.00	20.00	155.00	OK	
2023	13:00	LOS ANGELES	NEW YORK	AA127	Y	120.00	15.00	20.00	155.00	OK	
2023	14:00	NEW YORK	LOS ANGELES	AA128	Y	120.00	15.00	20.00	155.00	OK	
2023	15:00	LOS ANGELES	NEW YORK	AA129	Y	120.00	15.00	20.00	155.00	OK	
2023	16:00	NEW YORK	LOS ANGELES	AA130	Y	120.00	15.00	20.00	155.00	OK	
2023	17:00	LOS ANGELES	NEW YORK	AA131	Y	120.00	15.00	20.00	155.00	OK	
2023	18:00	NEW YORK	LOS ANGELES	AA132	Y	120.00	15.00	20.00	155.00	OK	
2023	19:00	LOS ANGELES	NEW YORK	AA133	Y	120.00	15.00	20.00	155.00	OK	
2023	20:00	NEW YORK	LOS ANGELES	AA134	Y	120.00	15.00	20.00	155.00	OK	
2023	21:00	LOS ANGELES	NEW YORK	AA135	Y	120.00	15.00	20.00	155.00	OK	
2023	22:00	NEW YORK	LOS ANGELES	AA136	Y	120.00	15.00	20.00	155.00	OK	

UNIVERSITY OF THE WITWATERSRAND
DEPARTMENT OF MINING ENGINEERING
BLASTING TECHNOLOGY

SAMPLE LENGTH (CM) x WEIGHT (g)		LOAD (g)		L (g)		U (g)		Cp (g)		Ca (g)		DENSITY (g/cm ³)	AD (MPa)	STRENGTH (MPa)
321	1 155.90	212.90	10.10	30.60	29.60	43.90	235.14	1867.64	1481.14	40.57	0.04	102.24		
	1 105.70	212.90	10.10	30.70	29.70	47.10	236.12	1871.22	1491.22	41.11	0.11	121.30		
	3.06 115.90	244.90	10.10	32.60	31.60	59.70	328.64	2181.22	1585.22	77.93	0.11	172.60		
	4.02 105.90	212.90	10.10	33.08	32.08	58.60	270.14	1640.26	1456.46	38.23	0.20	62.03		
	6.02 105.90	212.90	10.10	33.19	32.19	58.60	271.12	1650.12	1466.12	38.60	0.10	58.12		
	8.02 105.90	212.90	10.10	33.29	32.29	58.60	272.10	1660.10	1476.10	38.97	0.14	60.85		
	12.02 105.90	212.90	10.10	33.60	32.60	58.60	273.08	1670.08	1486.08	39.34	0.19	59.43		
	14.02 105.90	212.90	10.10	33.65	32.65	58.60	273.42	1680.42	1496.42	39.68	0.18	59.18		
	16.02 105.90	212.90	10.10	33.70	32.70	58.60	273.76	1690.76	1506.76	39.99	0.21	59.12		
	18.02 105.90	212.90	10.10	33.75	32.75	58.60	274.10	1701.10	1517.10	40.31	0.24	59.12		
	20.02 105.90	212.90	10.10	33.80	32.80	58.60	274.44	1711.44	1527.44	40.62	0.28	59.12		
	22.02 105.90	212.90	10.10	33.85	32.85	58.60	274.78	1721.78	1537.78	40.93	0.31	59.12		
	24.02 105.90	212.90	10.10	33.90	32.90	58.60	275.12	1732.12	1548.12	41.24	0.34	59.12		
	26.02 105.90	212.90	10.10	33.95	32.95	58.60	275.46	1742.46	1558.46	41.55	0.37	59.12		
	28.02 105.90	212.90	10.10	34.00	33.00	58.60	275.80	1752.80	1568.80	41.86	0.40	59.12		
	30.02 105.90	212.90	10.10	34.05	33.05	58.60	276.14	1763.14	1579.14	42.17	0.43	59.12		
	32.02 105.90	212.90	10.10	34.10	33.10	58.60	276.48	1773.48	1589.48	42.48	0.46	59.12		
	34.02 105.90	212.90	10.10	34.15	33.15	58.60	276.82	1783.82	1600.82	42.79	0.49	59.12		
	36.02 105.90	212.90	10.10	34.20	33.20	58.60	277.16	1794.16	1611.16	43.10	0.52	59.12		
	38.02 105.90	212.90	10.10	34.25	33.25	58.60	277.50	1804.50	1621.50	43.41	0.55	59.12		
	40.02 105.90	212.90	10.10	34.30	33.30	58.60	277.84	1814.84	1631.84	43.72	0.58	59.12		
	42.02 105.90	212.90	10.10	34.35	33.35	58.60	278.18	1825.18	1642.18	44.03	0.61	59.12		
	44.02 105.90	212.90	10.10	34.40	33.40	58.60	278.52	1835.52	1652.52	44.34	0.64	59.12		
	46.02 105.90	212.90	10.10	34.45	33.45	58.60	278.86	1845.86	1662.86	44.65	0.67	59.12		
	48.02 105.90	212.90	10.10	34.50	33.50	58.60	279.20	1856.20	1673.20	44.96	0.70	59.12		
	50.02 105.90	212.90	10.10	34.55										

[illegible]

Appendices

Results of laboratory investigation: Strength properties of coal

[illegible]

Results of laboratory investigation: Strength properties of coal

VOLUME LOSSING TEST FOR BULK OXYGENATION									
OXYGEN CONCENTRATION ANALYSIS									
TIME	TEMP	PH	DO (mg/L)	DO (%)	DO (mg/L)	DO (%)	DO (mg/L)	DO (%)	DO (mg/L)
1:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
2:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
3:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
4:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
5:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
6:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
7:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
8:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
9:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
10:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
11:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
12:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
13:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
14:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
15:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
16:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
17:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
18:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
19:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
20:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
21:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
22:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
23:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
24:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
25:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
26:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
27:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
28:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
29:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
30:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
31:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
32:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
33:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
34:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
35:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
36:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
37:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
38:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
39:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5
40:00	22.5	7.2	2.5	1.0	2.5	1.0	2.5	1.0	2.5

UNIVERSITY OF THE WATERSHED
DEPARTMENT OF MINING ENGINEERING
BLASTING TECHNOLOGY

PROJECT: DAMAGE IN COAL
DYNAMIC LOADING TEST FOR COAL CRUSHABILITY

SAMPLE ID (LENGTH BY WEIGHT) (g)			LOAD (N)	δ (m)	ω (s)	ΔE (J)	Co (m/s)	Cl (m/s)	GEOMETRY (K)	G (m/s)	Id.	2015-4-2009
7446	1	103.8	165.2	10	30.6	81.4	2568.307	1427.385	3382.65	27.14188	0.278958	66.2004
	2	334.3	189.2	10	43.1	52.8	2181.554	1101.57	3878.16	28.79967	0.248075	62.0875
	3	85.7	198.8	4	37.9	81.4	2382.333	890.75	4423.25	13.9891	0.344273	37.9186
	4	268	236.7	10	56.4	58.5	2052.539	1480.5	1269.28	28.72875	0.25163	63.1764
7447	1	154.7	332.2	10	42.8	48.7	2448.387	1485.186	1369.34	13.03793	0.274886	74.3455
							2533.89	2287.28	1391.64	15.02478	0.314884	64.2772

GRAVIMETRIC ANALYSIS

SAMPLE ID			DEPTH	H (mm)	S1=12.2	S2=0.3	S3=6.7	S4=4.75	S5=2.59	S6=1.38	S7=0.9	S8=0.33	S9=0.15	TEMP	IMPVULSION TIME CONC (%)	APPLIMT CONC (%)	CONC (%)
TIME	1	100	100	WGT WISCONT (%)	2.1	26.97	30.93	35.35	43.9	25.57	12.37	1.99	7.47	105.49	5.024842	5.000000	49.3719
				% WISCONT	1.506005	10.72695	16.62362	18.09647	22.66049	13.66135	6.88273	0.806652	3.747261	100			10.91534
				CUMUL. %	1.506005	12.233917	28.84018	46.93665	69.60014	83.26727	91.9471	98.20876	100				
				WISCONT (%)	55.67282	87.98139	71.86691	53.77334	31.31229	18.22673	9.89003	3.794255					
	2	110	110	WGT WISCONT (%)	31.89	34.25	28.22	24.7	34.05	37.91	13.47	3.51	5.05	106.49	5.022142	5.000000	51.03735
				% WISCONT	10.79136	16.33799	11.80235	13.16414	16.47406	0.342271	0.816736	1.423766	2.579187	100			
				CUMUL. %	10.79136	26.92338	40.35148	53.51562	69.98968	67.22135	68.04755	67.32391	100				
				WISCONT (%)	41.26948	67.17687	71.89892	76.96438	72.74934	12.24989	5.423251	2.626187					
	3	111	111	WGT WISCONT (%)	52.5	45.8	28.28	25.83	24.9	10.54	6.24	3.49	5.12	2.425108	5.004005	10.79125	50.86
				% WISCONT	28.11276	22.65066	14.58551	13.19153	12.91556	0.016437	0.066293	1.722674	1.867673	100			
				CUMUL. %	28.11276	50.76336	65.34887	78.54040	91.45596	93.79622	93.86427	95.61629	100				
				WISCONT (%)	73.89734	91.2966	98.70728	73.37379	11.21892	6.20318	3.116727	1.938719					
				CONC (%)	105.26	5.017731	5.013931	32.43886	0.0652	16.73407	171.2527						

APPENDIX B

Results of laboratory investigation: Strength properties of coal

DATE	TIME	LOCATION	WIND	TEMP	SEA	REMARKS
1964	01	10	10	10	10	10
1964	02	10	10	10	10	10
1964	03	10	10	10	10	10
1964	04	10	10	10	10	10
1964	05	10	10	10	10	10
1964	06	10	10	10	10	10
1964	07	10	10	10	10	10
1964	08	10	10	10	10	10
1964	09	10	10	10	10	10
1964	10	10	10	10	10	10
1964	11	10	10	10	10	10
1964	12	10	10	10	10	10
1964	13	10	10	10	10	10
1964	14	10	10	10	10	10
1964	15	10	10	10	10	10
1964	16	10	10	10	10	10
1964	17	10	10	10	10	10
1964	18	10	10	10	10	10
1964	19	10	10	10	10	10
1964	20	10	10	10	10	10
1964	21	10	10	10	10	10
1964	22	10	10	10	10	10
1964	23	10	10	10	10	10
1964	24	10	10	10	10	10
1964	25	10	10	10	10	10
1964	26	10	10	10	10	10
1964	27	10	10	10	10	10
1964	28	10	10	10	10	10
1964	29	10	10	10	10	10
1964	30	10	10	10	10	10
1964	31	10	10	10	10	10

APPENDIX C

Results of laboratory investigation:

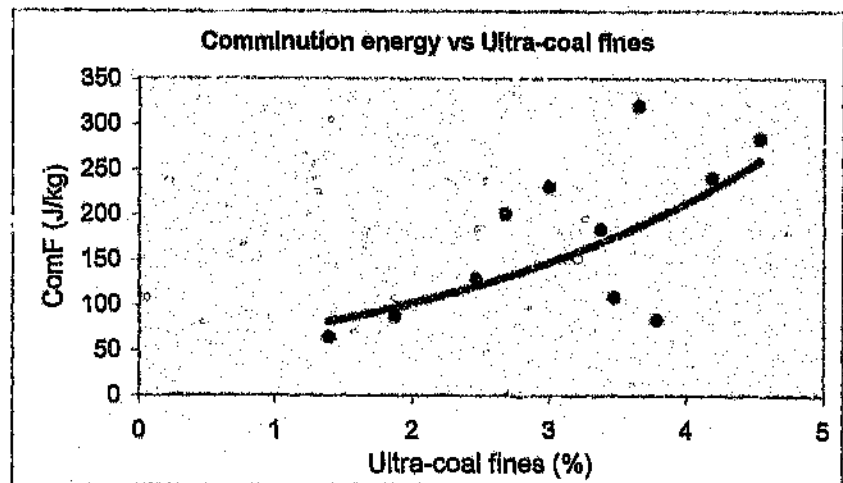
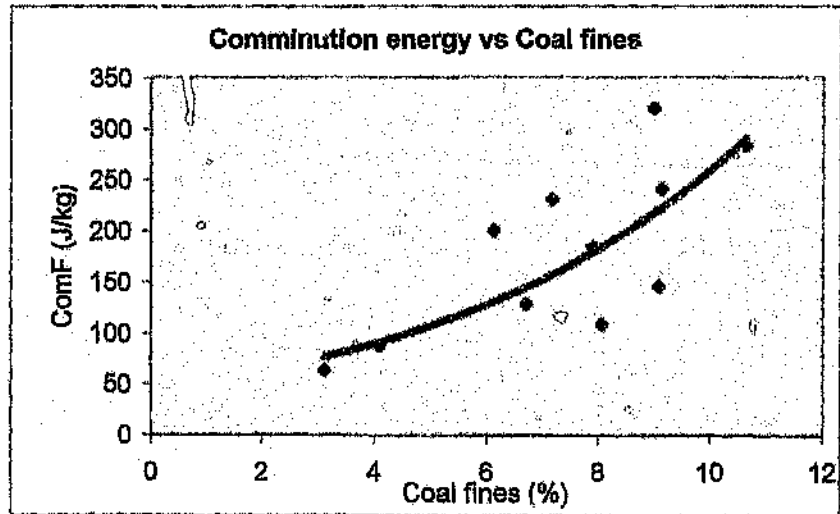
Crushability functions to model comminution of coal

APPENDIX C

Results of laboratory investigation:

Crushability functions to model comminution of coal

1. Observed crushability function for coal of Seam 4 at Tavistock, ATC

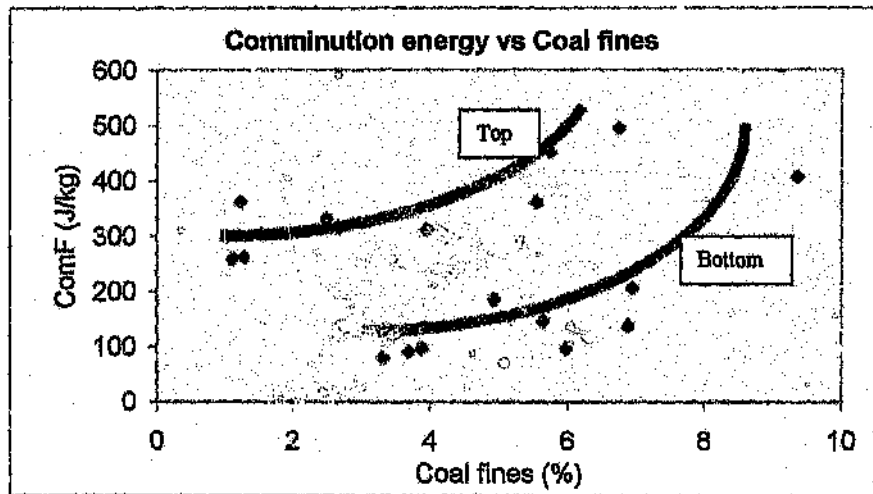


APPENDIX C

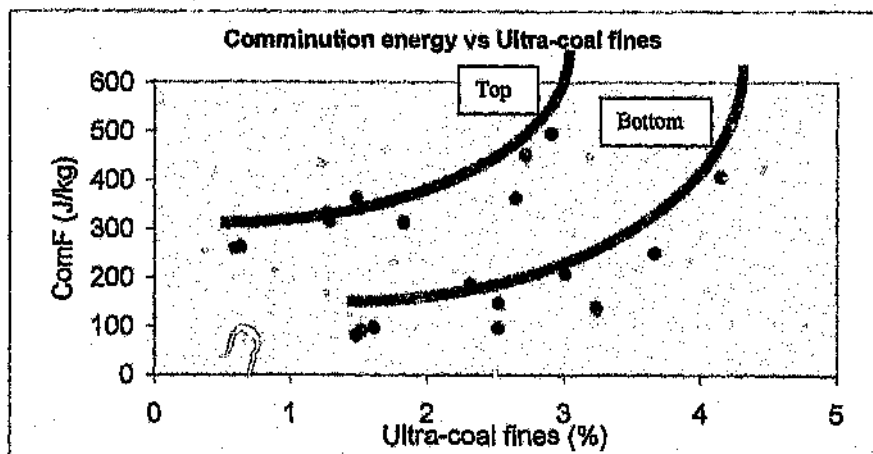
Results of laboratory investigation:

Crushability functions to model comminution of coal

2. Observed crushability function for coal of Seam 2 at ATC, Tavistock



APPENDIX C

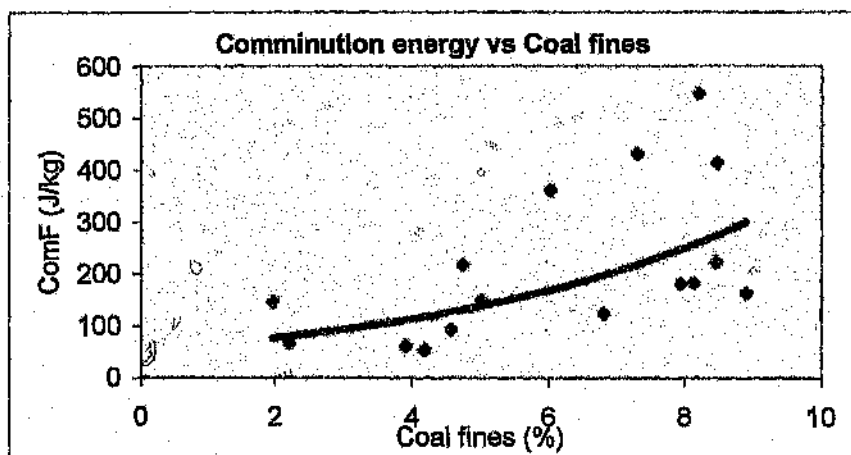


Appendices

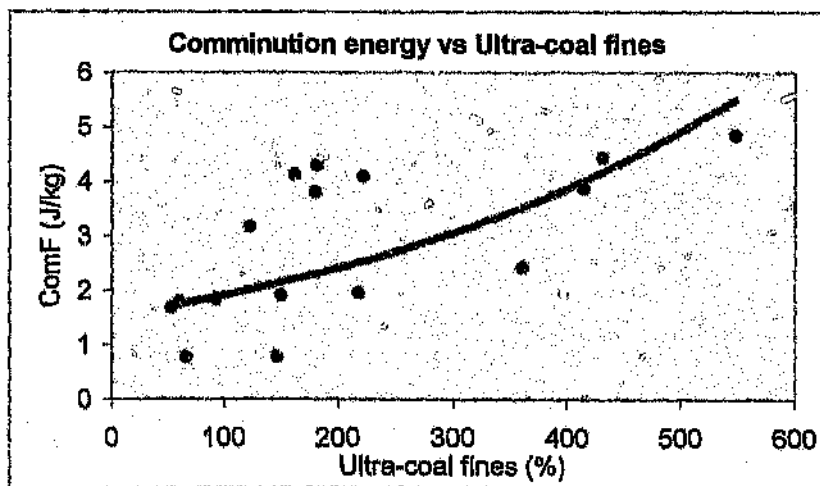
Results of laboratory investigation:

Crushability functions to model comminution of coal

3. Observed crushability function for coal of Seam 1 at A.T.C. Tavistock



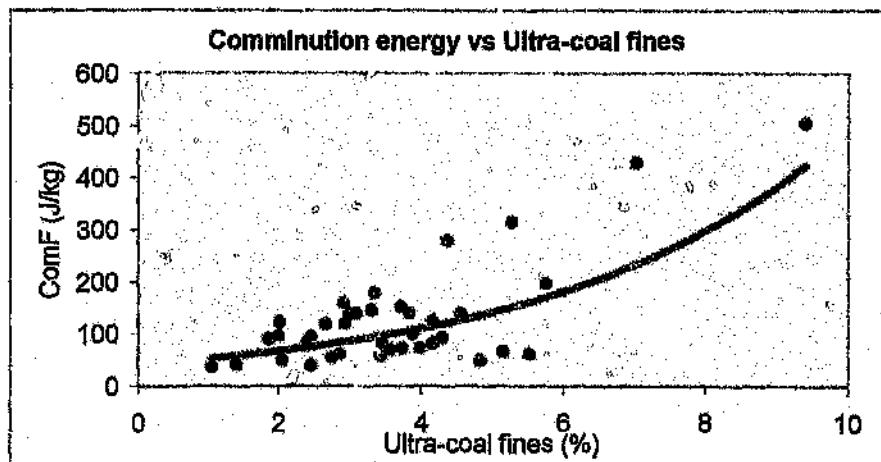
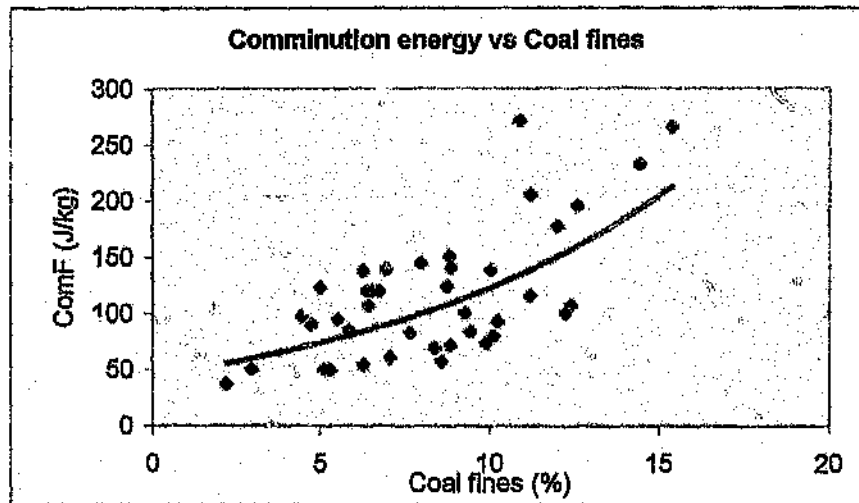
APPENDIX C



Results of laboratory investigation:

Crushability functions to model comminution of coal

4. Observed crushability function for coal of Seam 4 at Greenside

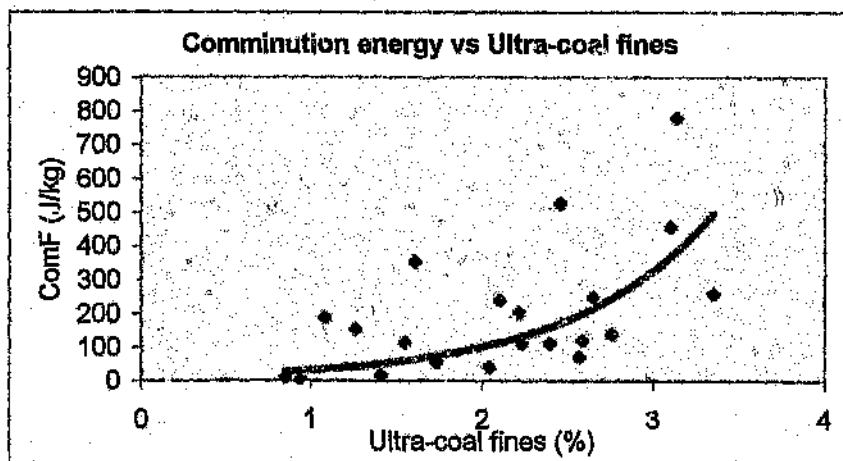
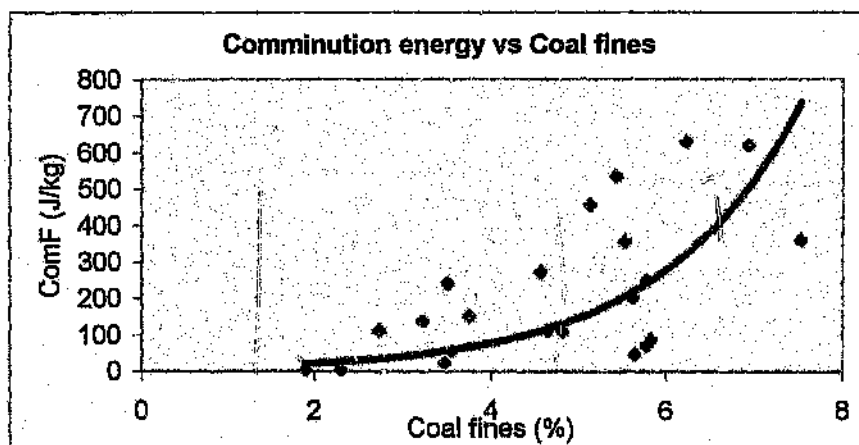


APPENDIX C

Results of laboratory investigation:

Crushability functions to model comminution of coal

5. Observed crushability function for coal of Seam 2 at Greenside



APPENDIX D

Results of field investigation: Surface blasting

Geotechnical logging of fractures in coal

Appendices

APPENDIX D

Results of field investigation: Surface blasting

Geotechnical logging of fractures in coal

1. Observed FF/m and RQD (%) in different coal seams at Kleinkopje

SEAM		Block 2A		Block 5W		PIT 3	
		x	s	x	s	x	s
4	FF(/m)	15	2.65	-	-	-	-
	RQD(%)	61	10.31	-	-	-	-
	n	23		-		-	
2	FF(/m)	18	12.18	20	4.38	35	6.34
	RQD(%)	46	7.60	43	11.09	20	12.68
	n	24		22		29	
1	FF(/m)	23	4.19	30	8.42	50	-
	RQD(%)	33	14.60	25	17.41	0	-
	n	23		16		5	

Legend:

x : the mean,

s : standard deviation and

n : the number of observations

Appendices

APPENDIX D

Results of field investigation: Surface blasting

Geotechnical logging of fractures in coal

2. Observed FF/m and RQD (%) for different coal types at Kleinkopie

SEAM	PIT		COAL TYPE	
			Mixed bright	Dull/lustrous
4	Block 2A	FF	15	18
		RQD	64	48
		n	16	15
2	Block 2A	FF	19	24
		RQD	44	35
		n	59	75
	Block5	FF	21	22
		RQD	50	33
		n	52	74
	Block 3A	FF	35	30
		RQD	18	18
		n	60	67
1	Block 2A	FF	15	22
		RQD	17	35
		n	1	23
	Block 5	FF		30
		RQD		27
		n		18
	Block 3A	FF		50
		RQD		0
		n		1

APPENDIX E

Results of field investigation: Surface blasting

Fragmentation obtained at Kleinkopje

APPENDIX E

Results of field investigation: Surface blasting

Fragmentation obtained at Kleinkopie

1. Blasting data and characteristics

Drilling diameter: 130 mm

Blasthole depth: variable 2 to 4 m according to the seam thickness,

Explosive: AECL emulsion explosive R 100

Mass per hole (kg): variable 5 to 15

Pattern Burden x spacing : 4 m x 4 m

Powder factor: 0.2 kg/m³

2. Legend of acronyms used

Frag Size	: Fragment size
2A2 S	: Coal from Block 2A in No. 2 Seam
2A4 S	: Coal from Block 2A in No. 4 Seam
5W1 S	: Coal from Block 5 West in No. 1 Seam
5WF S	: Coal from Block 5 West, Full Seam blasted
5WM2T	: Coal from Block 5 West Middle No. 2 Seam, Top horizon blasted
5WM2S	: Coal from Block 5 West Middle No. 2 Seam, Select (Low Ash Coal of the bottom seam blasted separately)
3AM2T	: Coal from Block 3A Middle No. 2 Seam, Top horizon blasted
3AM2S	: Coal from Block 3A Middle No. 2 Seam, Select (Low Ash Coal of the bottom seam blasted separately)

Appendices

APPENDIX E

Results of field investigation: Surface blasting

Fragmentation obtained at Kleinkopje

3. Observed fragmentation in blasted coal (Calculated average values)

Frag size (mm)	2A2 S	2A4 S	5W1 S	5WF S	5WM2T	5WM2S	3AM2T	3AM2S
< 0.5	0.6	1.1	0.7	1.2	1.1	0.9	2	1.3
0.5	3.9	5.1	4.2	5	5.4	4.9	10.4	7.8
3	3.9	4.6	3.7	4.1	4.7	4.1	8.6	6.6
6	10.4	12.7	9.6	12	15	12.3	26.3	20.3
15	16.2	14	11.3	12.8	12.6	12.2	13.2	11.9
25	4.8	7.1	7.2	5.8	6.2	6.1	5.4	8.2
32	7.9	20.3	18.3	14.9	13	15.2	7.3	9.2
50	22.4	18.5	20.6	17.8	23.7	21	17	19.8
75	11.5	6.5	11.2	14	7.2	10.8	4.1	6.6
100	18.4	10.1	13.2	12.4	11.1	12.5	5.7	5.9

4. Cumulative fragmentation

Frag size (mm)	2A2 S	2A4 S	5W1 S	5WF S	5WM2T	5WM2S	3AM2T	3AM2S
< 0.5	0.6	1.1	0.7	1.2	1.1	0.9	2	1.3
0.5	4.5	6.2	4.9	6.2	6.5	5.8	12.4	9.1
3	8.4	10.8	8.6	10.3	11.2	9.9	21	15.7
6	18.8	23.5	18.2	22.3	26.2	22.2	47.3	36
15	35	37.5	29.5	35.1	38.8	34.4	60.5	47.9
25	39.8	44.6	36.7	40.9	46	40.3	65.9	56.1
32	47.7	64.9	55	55.8	58	55.7	73.2	65.3
50	70.1	83.4	75.6	73.6	81.7	76.7	90.2	85.1
75	81.6	89.9	86.6	87.6	88.9	87.5	94.3	91.7
100	100	100	100	100	100	100	100	100

Appendices

APPENDIX E

Results of field investigation, Surface blasting

Fragmentation obtained at Kleinkopie

5. Detailed data of fragmentation observed

Percent passing of samples from two seam of Block 2A

FRAGMENT SIZE (mm)	PERCENT PASSING									
	K12/FS/E85H	K12/FS/E85L	K12/FS/F30H	K12/FS/F30L	K12/FS/F80H	K12/FS/F80L	K12/FS/G35H	K12/FS/G35L	K12/FS/G70H	K12/FS/G70L
-0.5	0.5	0.4	0.5	0.8	0.6	0.8	0.5	0.6	0.7	0.8
+0.5	4.3	3.6	4.3	3.6	3.4	4.4	3.5	3.1	4.5	4.1
+3	4.5	3.5	5.1	3.1	3.7	4.1	3.4	3.1	4.2	3.8
+6	11.2	10.2	12.5	9.7	9.5	10.9	9.7	10.1	10.3	9.8
+15	18.8	18.8	18.2	17.4	14.5	15.1	14.3	15.9	13.9	16.8
+25	3.6	5.5	4.9	6.6	3.4	4.7	3.3	5.0	4.4	6.8
+32	8.2	9.1	7.2	9.4	6.6	5.6	7.3	9.3	6.7	9.2
+50	25.2	29.9	22.4	28.1	12.9	20.5	17.3	18.4	24.1	28.3
+75	4.0	10.2	12.6	4.3	22.6	18.0	11.5	10.1	14.3	9.7
+100	21.7	8.6	12.3	17.0	22.8	17.9	29.2	24.4	16.9	12.6

Cumulative percent passing of samples from two seam of Block 2A

FRAGMENT SIZE (mm)	CUMULATIVE PERCENT PASSING									
	K12/FS/E85H	K12/FS/E85L	K12/FS/F30H	K12/FS/F30L	K12/FS/F80H	K12/FS/F80L	K12/FS/G35H	K12/FS/G35L	K12/FS/G70H	K12/FS/G70L
-0.5	0.5	0.4	0.5	0.8	0.6	0.8	0.5	0.6	0.7	0.8
+0.5	4.8	4.0	4.8	4.4	4.0	5.2	4.0	3.7	5.2	4.9
+3	9.3	76.5	9.9	7.5	7.7	9.3	7.4	6.8	9.4	8.7
+6	20.5	17.7	22.4	17.2	17.2	20.2	17.1	16.9	19.7	18.6
+15	37.3	36.5	40.6	34.6	31.7	35.3	31.4	32.8	33.6	35.4
+25	40.9	42.0	45.5	36.2	35.1	40.0	34.7	37.8	38.0	42.2
+32	49.1	51.1	52.7	45.6	41.7	45.6	42.0	47.1	44.7	51.4
+50	74.3	81.0	75.1	73.7	54.6	66.1	59.3	65.5	68.8	77.7
+75	78.3	91.2	87.7	76.0	77.2	82.1	70.8	75.6	83.1	87.4
+100	100	100	100	100	100	100	100	100	100	100

Percent passing of samples from the four seam of Block 2A

FRAGMENT SIZE (mm)	PERCENT PASSING							
	K13/4S/H0	K13/4S/I0	K13/4S/S50H	K13/4S/S50L	K13/4S/T0H	K13/4S/T0L	K13/4S/X40	K13/4S/Y30
-0.5	2.2	2.2	1.7	0.4	0.3	0.3	0.5	0.8
+0.5	8.9	8.0	3.2	4.9	3.6	3.9	3.3	5.0
+3	6.3	6.1	3.1	4.2	3.2	3.9	4.3	5.6
+6	12.3	13.4	11.1	12.8	11.7	10.9	13.9	15.6
+15	10.7	11.9	14.8	16.5	13.5	12.5	15.2	16.8
+25	5.3	7.0	7.8	7.9	6.5	7.6	7.6	7.4
+32	18.2	18.6	21.5	19.6	24.2	20.9	21.0	18.7
+50	15.4	17.4	21.2	18.3	23.0	27.3	14.8	10.9
+70	8.7	6.7	5.4	5.5	7.6	3.0	6.5	8.6
+100	12.0	8.7	10.2	9.9	6.4	9.7	12.9	10.8

Cumulative percent passing of samples from the four seam of Block 2A

FRAGMENT SIZE (mm)	CUMULATIVE PERCENT PASSING							
	K13/4S/H0	K13/4S/I0	K13/4S/S50H	K13/4S/S50L	K13/4S/T0H	K13/4S/T0L	K13/4S/X40	K13/4S/Y30
-0.5	2.2	2.2	1.7	0.4	0.3	0.3	0.5	0.8
+0.5	11.1	10.2	4.9	5.3	3.9	4.2	3.8	5.8
+3	17.4	16.3	8.0	9.5	7.1	8.1	8.1	11.4
+6	29.7	29.7	19.1	22.3	18.8	19.0	22.0	27.0
+15	40.4	41.6	33.9	38.8	32.3	31.5	37.2	43.8
+25	45.7	48.6	41.7	46.7	38.8	39.1	44.8	51.2
+32	63.9	67.2	63.2	66.3	63.0	60.0	65.8	69.9
+50	79.3	84.6	84.4	84.6	86.0	87.3	80.6	80.8
+70	88.0	91.3	89.8	90.1	93.6	90.3	87.1	89.4
+100	100	100	100	100	100	100	100	100

Percent passing of samples from the one seam of Block 5W

FRAGMENT SIZE (mm)	PERCENT PASSING										
	F16/IS/Q25	F16/IS/Q50	F16/IS/Q75	F16/IS/P0	F16/IS/P25	F16/IS/P50	F16/IS/P75	F16/IS/Q0	F16/IS/Q25	F16/IS/Q50	F16/IS/Q75
-0.5	0.8	1.0	0.8	0.5	0.8	0.9	0.5	0.3	0.5	0.7	1.0
+0.5	4.5	5.0	5.2	3.3	4.3	3.9	3.8	4.4	4.4	5.2	2.7
+3	3.6	3.6	4.8	3.4	3.9	3.2	3.0	3.6	3.8	3.7	4.5
+8	10.2	11.5	11.7	7.9	12.1	9.0	9.3	10.8	9.8	9.5	3.3
+15	10.3	12.8	11.4	8.9	14.1	11.0	11.1	12.3	14.5	9.5	7.9
+25	7.8	8.0	6.8	4.7	6.3	5.7	6.3	7.4	7.3	7.0	13.8
+32	22.6	18.3	19.9	15.9	21.4	16.1	18.8	22.2	19.8	17.8	8.1
+50	25.2	15.9	18.0	14.2	17.3	18.8	17.4	20.6	26.7	18.4	34.5
+70	8.0	10.0	11.2	24.2	12.8	14.3	10.2	8.2	5.9	7.8	11.0
+100	7.4	15.9	10.2	17.0	7.0	17.3	19.6	10.2	7.3	20.4	13.2

Cumulative percent passing of samples from the one seam of Block 5W

FRAGMENT SIZE (mm)	CUMULATIVE PERCENT PASSING										
	F16/IS/Q25	F16/IS/Q50	F16/IS/Q75	F16/IS/P0	F16/IS/P25	F16/IS/P50	F16/IS/P75	F16/IS/Q0	F16/IS/Q25	F16/IS/Q50	F16/IS/Q75
-0.5	0.8	1	0.8	0.5	0.8	0.9	0.5	0.3	0.5	0.7	1.0
+0.5	5.1	6	6	3.8	5.1	4.9	4.3	4.7	4.9	5.9	3.7
+3	8.7	9.8	10.8	7.2	9.0	8.0	7.3	8.3	8.7	9.6	8.2
+8	18.9	21.1	22.5	15.1	21.1	17.0	16.6	19.1	18.5	19.1	11.5
+15	29.2	33.9	33.9	24.0	35.2	28.0	27.7	31.4	33.0	28.6	19.4
+25	36.8	39.9	40.7	28.7	41.5	33.7	34.0	38.8	40.3	35.6	33.2
+32	59.4	58.2	60.6	44.6	62.9	49.8	52.8	61.0	60.1	53.4	41.3
+50	84.6	74.1	78.6	58.8	80.2	68.4	70.2	81.6	86.8	71.8	75.8
+70	92.6	84.1	89.8	83.0	93.0	82.7	80.4	89.8	92.7	79.6	86.8
+100	100	100	100	100	100	100	100	100	100	100	100

Percent passing of samples from M2S (2 seam) of Block 5W

FRAGMENT SIZE (mm)	PERCENT PASSING								
	G23/2S/N30	G23/2S/P10	G23/2S/R25	G23/2S/S30H	G23/2S/S30L	G23/2S/T25H	G23/2S/T25L	G21/2S/C30	G21/2S/N15
-0.5	1.0	1.4	1.0	0.7	0.5	0.4	0.8	1.1	0.9
+0.5	6.4	5.9	4.5	4.0	3.8	4.3	5.0	4.9	4.8
+3	5.2	4.3	5.5	4.0	3.7	3.4	3.4	4.3	4.7
+6	19.9	15.7	17.7	10.0	12.1	9.6	9.1	11.5	12.0
+15	11.0	9.0	16.7	17.0	12.7	9.0	12.6	12.1	11.7
+25	5.7	5.8	7.7	4.8	6.6	7.6	5.0	6.3	6.6
+32	10.0	8.6	15.3	6.8	19.0	18.6	17.3	18.5	17.3
+50	30.0	25.8	17.6	21.4	19.8	21.8	29.3	24.8	10.3
+70	7.2	7.7	3.6	10.6	9.3	16.4	14.0	4.5	14.5
+100	3.6	15.8	10.4	20.7	12.7	8.9	3.5	12.0	17.2

Cumulative percent passing of samples from M2S (2 seam) of Block 5W

FRAGMENT SIZE (mm)	CUMULATIVE PERCENT PASSING								
	G23/2S/N30	G23/2S/P10	G23/2S/R25	G23/2S/S30H	G23/2S/S30L	G23/2S/T25H	G23/2S/T25L	G21/2S/C30	G21/2S/N15
-0.5	1.0	1.4	1.0	0.7	0.5	0.4	0.8	1.1	0.9
+0.5	7.4	7.3	5.5	4.7	4.1	4.7	5.8	6.0	5.7
+3	12.6	11.6	11.0	8.7	7.8	8.1	9.2	10.3	10.4
+6	32.5	27.3	28.7	18.7	19.9	17.7	18.3	21.8	22.4
+15	43.5	38.3	45.4	35.7	32.6	26.7	30.9	33.9	34.1
+25	49.2	42.1	53.1	40.5	39.2	34.3	35.9	40.2	40.7
+32	59.2	50.7	68.4	47.3	58.2	52.9	53.2	58.7	58.0
+50	89.2	76.6	86.0	68.7	78.0	74.7	72.5	83.5	68.3
+70	96.4	84.2	89.6	79.3	87.3	91.1	96.5	88.0	82.8
+100	100	100	100	100	100	100	100	100	100

Percent passing of samples from the M2S (2 seam) and Full (2) seam of Block 5W (contd)

FRAGMENT SIZE (mm)	PERCENT PASSING						
	G23/2S/U35H	G23/2S/U35L	G23/FS/V50H	G23/FS/V50L	G23/FS/W0H	G23/FS/W0L	G21/FS/A25
-0.5	0.5	0.5	0.4	1.0	1.3	2.2	1.2
+0.5	4.7	4.3	4.9	4.3	5.2	3.7	7.0
+3	3.7	3.1	3.9	3.0	3.3	4.1	6.0
+6	10.6	8.5	10.7	8.8	7.6	11.0	22.1
+15	8.6	10.4	13.0	11.4	13.2	15.0	11.6
+25	7.0	7.0	5.6	6.0	3.0	7.0	6.7
+32	17.3	19.3	19.9	17.3	8.4	20.4	8.7
+50	22.6	23.6	24.4	11.7	10.6	15.7	26.5
+75	8.8	5.5	7.2	23.7	27.5	7.4	4.4
+100	16.2	17.8	10.0	12.8	18.9	13.5	5.8

Cumulative percent passing of samples from the M2S (2 seam) and Full (2) seam of Block 5W (contd)

FRAGMENT SIZE (mm)	CUMULATIVE PERCENT PASSING						
	G23/2S/U35H	G23/2S/U35L	G23/FS/V50H	G23/FS/V50L	G23/FS/W0H	G23/FS/W0L	G21/FS/A25
-0.5	0.5	0.5	0.4	1.0	1.3	2.2	1.2
+0.5	5.2	4.8	5.3	5.3	6.5	5.9	8.2
+3	8.9	7.9	9.2	8.3	9.8	10.0	14.2
+6	19.5	18.4	19.9	17.1	17.4	21.0	36.3
+15	28.1	26.8	32.9	28.5	30.6	36.0	47.9
+25	35.1	33.8	38.5	34.5	33.6	43.0	54.6
+32	52.4	53.1	58.4	51.8	42.0	63.4	63.3
+50	75.0	76.7	82.8	63.5	52.6	79.1	89.8
+75	83.8	82.2	90.0	87.2	80.1	86.5	94.2
+100	100	100	100	100	100	100	100

Percent passing of samples from the M2T (2 seam) of Block 5W

FRAGMENT SIZE (mm)	PERCENT PASSING								
	G23/2T/N25	G23/2T/O30	G23/2T/P20	G23/2T/R20	G23/2T/S30	G23/2T/T30	G23/2T/U40	G21/2T/C25	G21/2T/N25
-0.5	0.9	1.3	1.0	0.9	1.3	1.5	1.1	1.2	0.7
+0.5	6.5	6.5	5.1	5.4	6.1	5.5	3.8	5.6	4.3
+3	5.0	5.1	5.0	5.6	5.1	5.0	3.6	4.1	3.6
+6	22.5	20.2	18.6	11.5	14.7	13.6	10.7	13.5	9.7
+15	10.0	11.0	9.1	18.0	15.5	12.4	12.4	7.1	18.0
+25	6.8	4.9	6.8	5.0	8.3	7.4	7.6	4.1	5.2
+32	8.9	9.8	8.2	9.8	22.3	19.7	20.8	6.4	10.9
+50	22.3	26.6	22.7	27.3	14.6	26.4	25.7	20.2	27.2
+70	9.3	6.6	8.2	7.5	5.8	2.2	4.3	13.4	8.0
+100	7.8	8.0	15.3	9.0	6.5	6.3	10.0	24.4	12.4

Cumulative percent passing of samples from the M2T (2 seam) of Block 5W

FRAGMENT SIZE (mm)	CUMULATIVE PERCENT PASSING								
	G23/2T/N25	G23/2T/O30	G23/2T/P20	G23/2T/R20	G23/2T/S30	G23/2T/T30	G23/2T/U40	G21/2T/C25	G21/2T/N25
-0.5	0.9	1.3	1.0	0.9	1.3	1.5	1.1	1.2	0.7
+0.5	7.4	7.8	6.1	6.3	7.4	7.0	4.9	6.8	5.0
+3	12.4	12.9	11.1	11.9	12.5	12.0	8.5	10.9	8.6
+6	34.9	33.1	29.7	23.4	27.2	25.6	19.2	24.4	18.3
+15	44.9	44.1	38.8	41.4	42.7	38.0	31.6	31.5	36.3
+25	51.7	49.0	45.6	46.4	51.0	45.4	39.2	35.6	41.5
+32	60.6	58.8	53.8	56.2	73.3	65.1	60.0	42.0	52.4
+50	82.9	85.4	76.5	83.5	87.9	91.5	85.7	62.2	79.6
+70	92.2	92.0	84.7	91.0	93.5	93.7	90.0	75.6	87.6
+100	100	100	100	100	100	100	100	100	100

Percent passing of samples from the M2S (2 seam) of Pit 3A

FRAGMENT SIZE (mm)	PERCENT PASSING								
	B78/2S/C30	B78/2S/D30	B78/2S/E90	B78/2S/F90	B78/2S/G90	B78/2S/H65	B78/2S/I30	A72/2S/C85	A72/2S/C0
-0.5	1.1	3.1	2.6	1.5	1.0	0.8	2.6	0.3	2.2
+0.5	5.7	15.2	8.4	7.5	5.2	5.7	6.6	15.4	12.5
+3	8.6	12.5	5.9	5.4	4.8	5.2	5.1	15.7	9.8
+6	27.1	31.3	21.3	15.3	21.8	17.1	14.6	31.0	31.6
+15	11.0	9.9	11.9	8.4	12.2	18.7	18.9	31.4	10.2
+25	5.5	4.7	5.6	3.4	5.5	8.9	8.7	5.7	7.5
+32	5.0	5.5	10.2	7.6	9.2	22.4	26.4	0.5	5.8
+50	12.9	11.6	22.6	20.7	22.2	12.6	10.5	0	13.2
+70	6.4	4.8	6.3	10.2	7.6	6.4	3.7	0	1.6
+100	16.7	1.4	5.2	20.0	10.3	2.2	4.9	0	5.5

Percent passing of samples from the M2S (2 seam) of Pit 3A (contd)

FRAGMENT SIZE (mm)	PERCENT PASSING				
	A72/2S/A80	A72/2S/A60	A72/2S/A40	A72/2S/A25	A72/2S/A0
-0.5	1.2	2.6	0.2	1.0	0.4
+0.5	7.4	12.3	1.9	4.2	0.4
+3	7.5	7.6	2.2	1.3	0.5
+6	33.6	23.2	9.7	6.3	0.9
+15	12.1	10.0	5.9	4.8	1.8
+25	13.1	5.0	12.5	15.7	13.0
+32	4.2	6.9	9.5	5.7	9.4
+50	14.9	18.2	38.8	36.4	42.9
+70	3.4	5.3	7.5	14.8	13.8
+100	2.6	8.9	11.8	9.8	16.9

Cumulative percent passing of samples from the M2S (2 seam) of Pit 3A

FRAGMENT SIZE (mm)	CUMULATIVE PERCENT PASSING								
	B78/2S/C30	B78/2S/D30	B78/2S/E90	B78/2S/F90	B78/2S/G90	B78/2S/H65	B78/2S/I30	A72/2S/C85	A72/2S/C0
-0.5	1.1	3.1	2.6	1.5	1.0	0.8	0.6	0.3	2.2
+0.5	6.8	18.3	11.0	9.0	6.2	6.5	7.2	15.7	14.7
+3	15.4	30.8	16.9	14.4	11.0	11.7	12.3	31.4	24.5
+6	42.5	62.1	38.2	29.7	32.8	28.8	28.9	62.4	56.1
+15	53.5	72.0	50.1	38.1	45.0	47.5	45.8	93.8	66.3
+25	59.0	76.7	55.7	41.5	50.5	53.4*	54.5	99.5	73.8
+32	64.0	82.2	65.9	49.1	59.7	78.8	80.9	100	79.6
+50	76.9	93.8	88.5	69.8	81.9	91.4	91.4	100	92.8
+70	83.3	98.6	94.8	80.0	89.7	97.8	95.1	100	94.4
+100	100	100	100	100	100	100	100	100	100

Cumulative percent passing of samples from the M2S (2 seam) of Pit 3A (contd)

SIZE (mm)	CUMULATIVE PERCENT PASSING				
	A72/2S/A60	A72/2S/A80	A72/2S/A40	A72/2S/A25	A72/2S/A0
-0.5	1.2	2.6	0.2	1.0	0.4
+0.5	8.6	14.9	2.1	5.2	0.8
+3	16.1	22.5	4.3	6.5	1.3
+6	49.7	45.7	14.0	12.8	2.2
+15	61.8	55.7	19.9	17.6	4.0
+25	74.9	60.7	32.4	33.3	17.0
+32	79.1	67.6	41.9	39.0	26.4
+50	84.0	85.8	80.7	75.4	69.3
+70	97.4	91.1	88.2	90.2	83.1
+100	100	100	100	100	100

Percent passing of samples from M2T (2 seam) of Pit 3A

FRAGMENT SIZE (mm)	PERCENT PASSING							
	B78/2T/E75	B78/2T/F90	B78/2T/G90	B78/2T/H50	B78/2T/I20	A72/2T/C75	A72/2T/C0	A72/2T/A90
-0.5	1.3	1.9	1.0	1.2	4.0	0.8	3.2	2.2
+0.5	9.5	6.1	5.2	6.3	11.1	17.2	15.1	13.4
+3	8.8	5.6	4.8	4.5	7.9	19.3	8.9	9.2
+6	27.9	21.6	21.8	20.0	22.6	35.4	31.0	30.0
+15	13.0	11.3	12.2	12.3	9.8	24.0	12.3	10.8
+25	5.7	7.3	5.5	7.4	4.4	2.9	5.2	4.4
+32	8.5	8.4	9.2	12.4	8.4	0.4	5.8	5.1
+50	21.5	21.8	22.2	29.5	15.9	0	12.9	11.9
+70	2.5	7.6	7.8	5.4	5.1	0	0.9	3.9
+100	1.3	8.4	10.3	1.0	10.8	0	4.7	9.1

E12

Cumulative percent passing of samples from M2T (2 seam) of Pit 3A

FRAGMENT SIZE (mm)	CUMULATIVE PERCENT PASSING							
	B78/2T/E75	B78/2T/F90	B78/2T/G90	B78/2T/H50	B78/2T/I20	A72/2T/C75	A72/2T/C0	A72/2T/A90
-0.5	1.3	1.9	1	1.2	4	0.8	3.2	2.2
+0.5	10.8	8	6.2	7.5	15.1	18	18.3	15.6
+3	19.6	13.6	11	12	23	37.3	27.2	24.8
+6	47.5	35.2	32.8	32	45.6	72.7	58.2	54.8
+15	60.5	46.5	45	44.3	55.4	96.7	70.5	65.6
+25	66.2	53.8	50.5	51.7	59.8	99.6	75.7	70
+32	74.7	62.2	59.7	64.1	68.2	100	81.5	75.1
+50	96.2	84	81.9	93.6	84.1	100	94.4	87
+70	98.7	91.6	89.7	99	89.2	100	95.3	90.9
+100	100	100	100	100	100	100	100	100

Appendix

APPENDIX F

Results of field investigation: Underground blasting

In-hole VOD measured insitu at ATC, Tavistock

Appendices

APPENDIX F

Results of field investigation: Underground blasting

In-hole VOD measured insitu at ATC, Tavistock

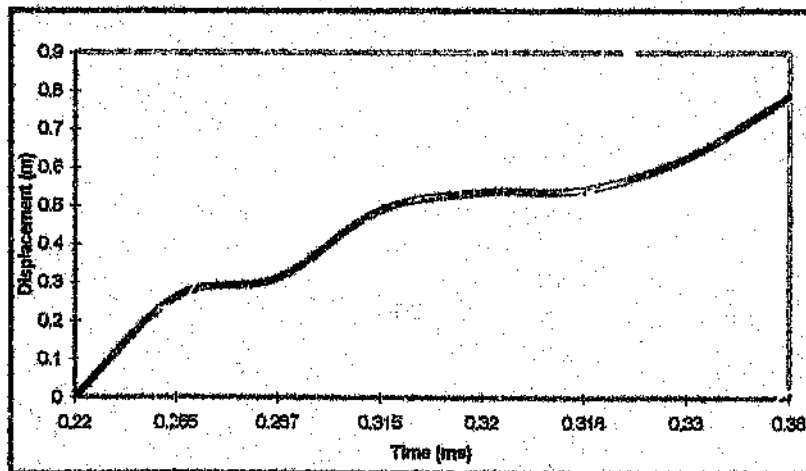
In-hole VOD measurement in coal blasting

Blast MX82#251 of 26/09/1996

Data:

Location: Section 2, Road 3, Face: left
Seam 2 at Tavistock, Arthur Taylor Colliery
Charge/Drilled diameter: 25 mm / 36 mm
Charge length
 . theoretical: $4 \times 360 \text{ mm} = 1.44 \text{ m}$
 . measured : 0.8 m
Cut depth: 2.25 m
Number of holes: 22 (7,5,5,5)

Result: VOD of 5071 m/s



Appendices

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Results of field investigation: Underground blasting

In-hole VOD measured insitu at ATC, Tavistock

In-hole VOD measurement in coal blasting

Blast MX84#251 of 17/09/1996

Data:

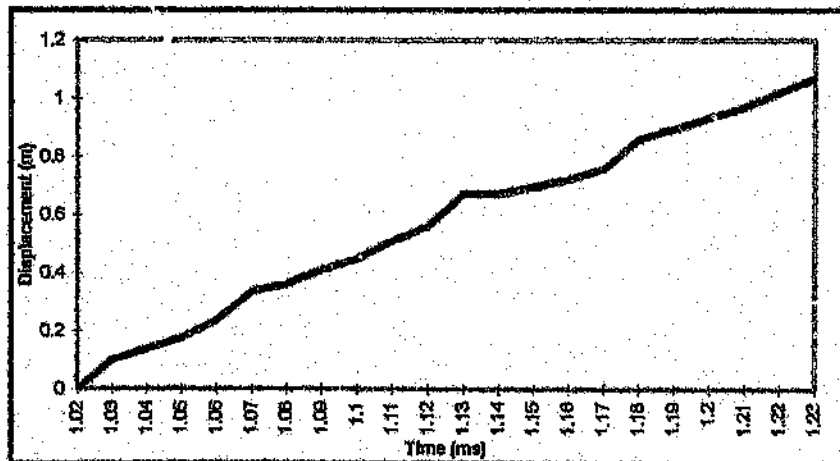
Location: Section 3, Road 6, Face: right
Seam 4 at Tavistock, Arthur Taylor Colliery
Charge/Drilled diameter: 25 mm / 40 mm
Charge length

- . theoretical: $4 \times 360 \text{ mm} = 1.44 \text{ m}$
- . measured: 1.071 m

Cut depth: 2.25 m

Number of holes: 13 (5,3,5,5)

Result: VOD of 5100 m/s



Appendices

APPENDIX F

Results of field investigation: Underground blasting

In-hole VOD measured insitu at ATC, Tavistock

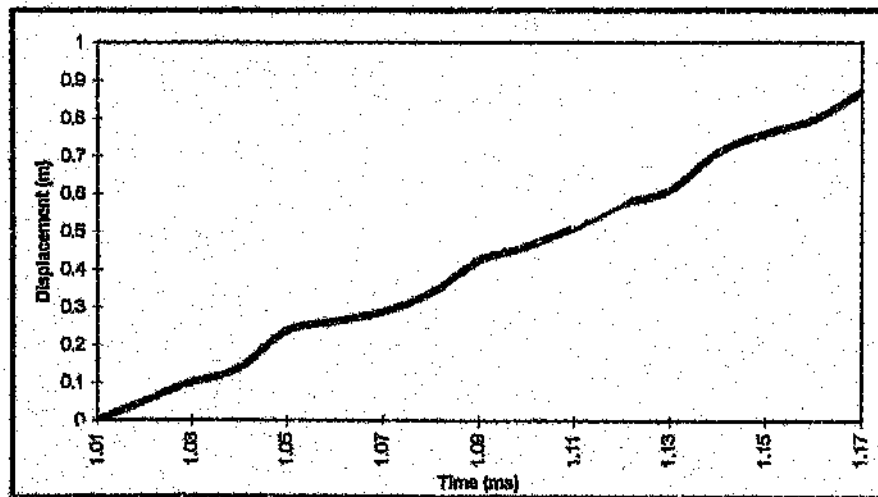
In-hole VOD measurement in coal blasting

Blast MX84#252 of 17/09/1996

Data:

Location: Section 3, Road 5, Face: left
Seam 4 at Tavistock, Arthur Taylor Colliery
Charge/Drilled diameter: 25 mm / 40 mm
Charge length
 . theoretical: 4 x 360 mm = 1.44 m
 . measured : 0.872 m
Cut depth: 2.25 m
Number of holes: 18 (5,3,5,5)

Result: VOD of 5450 m/s



Appendices

APPENDIX F

Results of field investigation: Underground blasting

In-hole VOD measured insitu at ATC, Tavistock

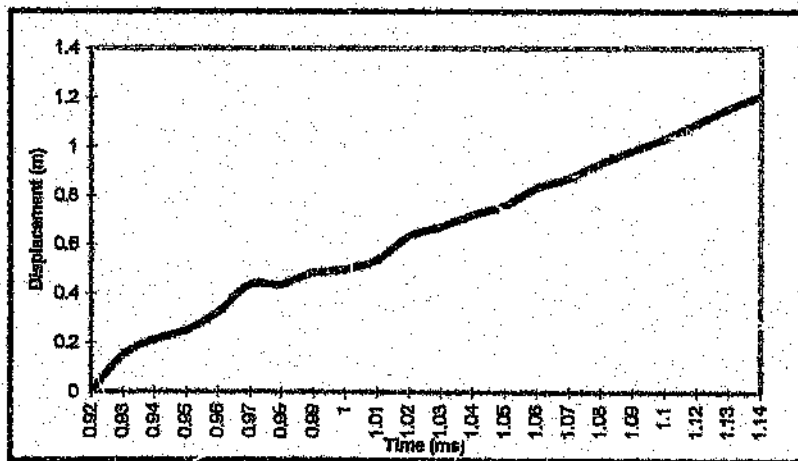
In-hole VOD measurement in coal blasting

Blast MX84#253 of 7/09/1996

Data:

Location: Section 3, Road 4, Face: left
Seam 4 at Tavistock, Arthur Taylor Colliery
Charge/Drilled diameter: 27 mm / 40 mm
Charge length
 . theoretical: $4 \times 330 \text{ mm} = 1.32 \text{ m}$
 . measured : 1.295 m
Cut depth: 2.25 m
Number of holes: 18 (5,3,5,5)

Result: VOD of 5396 m/s



APPENDIX F

Results of field investigation: Underground blasting

In-hole VOD measured insitu at ATC, Tavistock

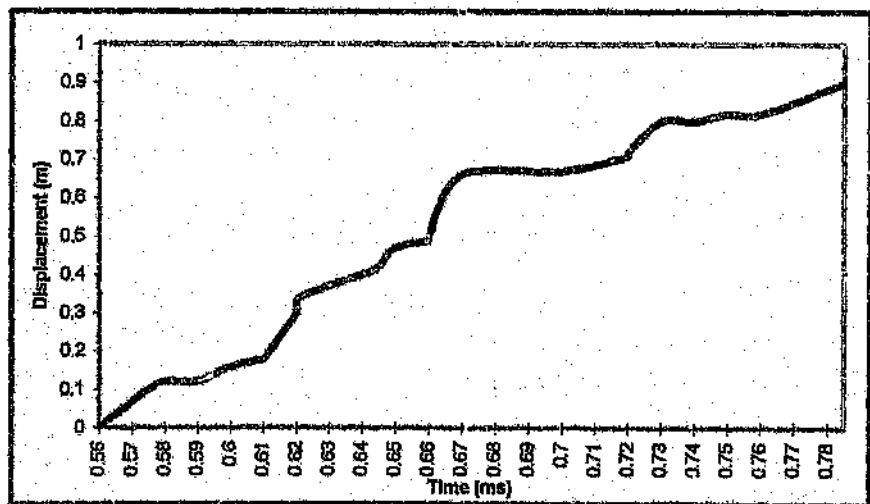
In-hole VOD measurement in coal blasting

Blast MX82#291 of 26/09/1996

Data:

Location: Section 2, Road 1, Face: front
Seam 2 at Tavistock, Arthur Taylor Colliery
Charge/Drilled diameter: 29 mm / 36 mm
Charge length
 . theoretical: $4 \times 270 \text{ mm} = 1.08 \text{ m}$
 . measured : 0.824 m
Cut depth: 2.25 m
Number of holes: 22 (7,5,5,5)

Result: VOD of 4709 m/s



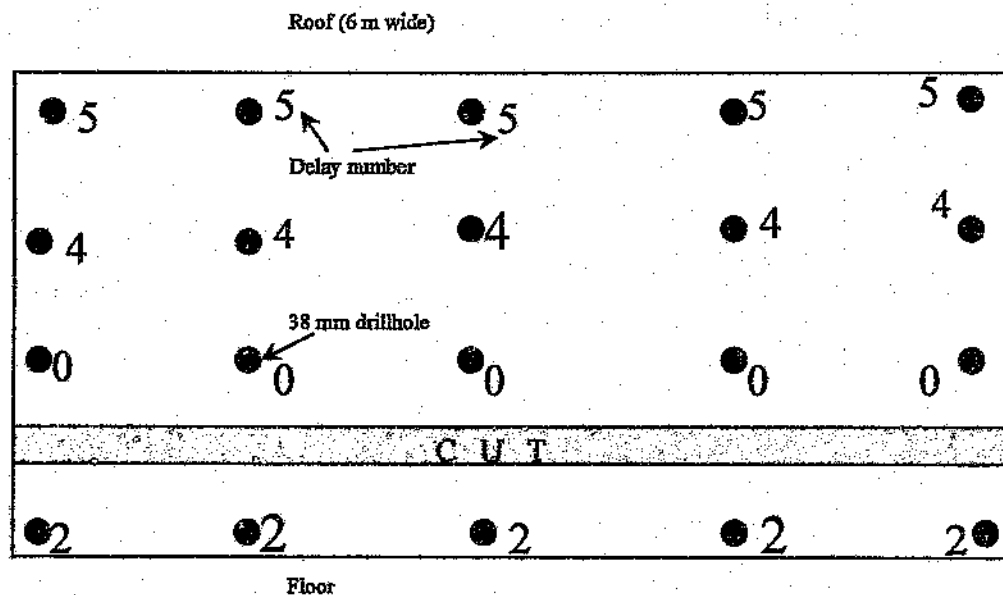
APPENDIX G

Results of field investigation: Underground blasting
Timing patterns used to investigate the optimum inter-row delays
and recent development of this project

APPENDIX G

Results of field investigation: Underground blasting

Timing patterns used to investigate the optimum inter-row delays

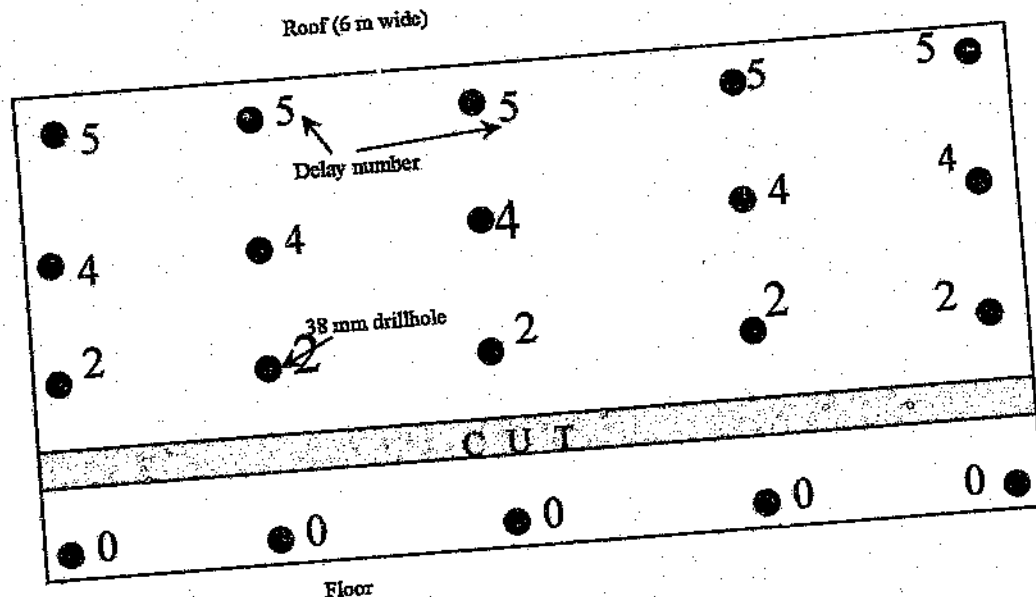


Conventional pattern of the Cut Blasting technique used at Kriel Colliery

APPENDIX G

Results of field investigation: Underground blasting

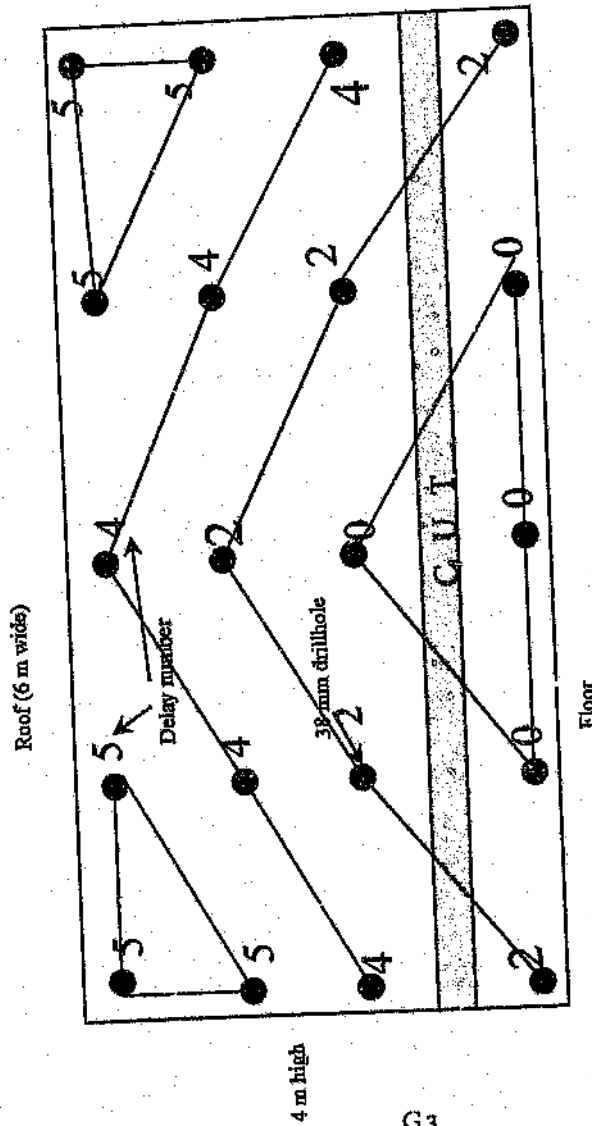
Timing patterns used to investigate the optimum inter-row delays



A Coal Cut Blasting pattern with Reversed firing of blasthole rows (REV)

APPENDIX G

Results of field investigation: Underground blasting
Timing patterns used to investigate the optimum inter-row delays



A Coal Cut Blasting pattern tied in a V1 chevron (CHEV)

APPENDIX G

Results of field investigation: Underground blasting

Recent development of the project

BLASTING EFFICIENCIES AT KRIEL COLLIERY

A paper extracted from the project report by
C. Du Plessis¹ under supervision of K. K. Kabongo²
Department of Mining Engineering
University of the Witwatersrand

ABSTRACT

The purpose of this report is to investigate the reasons for the poor face advances being achieved at Kriel Colliery and to establish whether trial blasts conducted with longer delays may be part of a solution.

Very poor face advances per blast have resulted since the introduction of the watergel based explosive, Enercoal. Because of the superior strength of previously used Nitroglycerin based explosives, cutting and drilling operations were less critical. However, when blasting with Enercoal, precision work is required in order to achieve acceptable face advances.

The trials commenced in November 1996 at one of the blasting sections of the colliery. The trials were a follow up of the previous work done by C.H. Lubbe on the optimization of delay timing in coal blasting at the same mine. The objective of this renewed effort was to determine if better face advances might be achieved with longer delayed timing of the blast. Substantial improvements resulted from these trials to the extent that the management has decided to the introduction of this timing system in all standard blast work at the colliery.

¹ Student in the Department of Mining Engineering at the University of the Witwatersrand

² Lecturer of Blasting Technology

INTRODUCTION

Kriel Colliery forms part of Amcoal Colliery and Industrial Operations Limited and is situated in the northern margin of the Highveld coalfield. It is approximately midway between Ogies and Bethal, 12 km north-west of Kriel town. The colliery was established in 1976 to supply coal to Eskom's 3000 MV Kriel Power Station.

Both underground and opencast extraction methods are employed to produce a target of 8,5 million tons of coal per annum, of which the underground produces approximately 4 million tons and the opencast section the remainder. The underground mine at North West Shaft currently utilises both conventional and continuous mining methods, with two Voest Alpine AM 80 road headers and five blasting sections.

GEOLOGY

Sediments of the Vryheid and Dwyka formations deposited on a glaciated pre-Karoo basement consisting of Rooiberg felsites underlie the Highveld coalfield. The three principal seams are the number 2, 4 and 5 seams. The no. 4 seam is the most important and covers most of the reserve area. The depth of this seam varies from sub-outcrops to 70 metres below surface. The depth limitation for extraction by opencast method is 40 metres. Coal deeper than 40 metres is mined by bord and pillar method.

The seam thickness varies from 3,0 to 7,0 metres, with an average of 4,9 metres. Three main quality zones may be identified as following. The upper poorer quality that consists mainly of dull to dull shaly coal with interbedded carbonaceous shales. The middle good quality section comprises massive lustrous coal with thin bright bands. The lower intermediate quality zone consists of dull to lustrous coal with minor thin bright bands.

The full coal seam is mined in the opencast section, whereas underground mining is confined to the lower 3,5 to 4 metres. The fossil marker, which is a natural cleavage plane on a bright band at the base of the upper dull shaly coal zone, forms a competent clean roof to the underground workings. A persistent shale band or shale marker, 10 to 15 centimetres in thickness, can be easily identified approximately 15 centimetres below the fossil marker.

The remaining approximately 1,4 metres of poor quality coal is left behind in the roof which is then supported by a set pattern of roof bolts and forms a buffer to protect the overlying sediments from rapid weathering, thereby reducing the risk of fall of ground.

Appendices

The area has been intruded by a number of dolomite sills and dykes. Large areas of reserves have been burnt and rendered useless for exploitation. Intrusions have also caused significant displacements of the seams.

BREAKING A COAL FACE

The Energy Of Explosives

Energy is required to break a coal face. All commercial explosives are largely composed of oxidisers and fuels which, when detonated, react extremely rapidly to produce gases at very high temperatures and pressures. This rapid pressures increase supply the energy needed to break the coal. In coal mining, where the risk of an explosible atmosphere exists, limitations are placed on the strength of the explosives to minimise the production of incandescent particles and flame on detonation. The strength of an explosive is a measure of its capacity to break material, coal for instance.

Explosives Used At Kriel Colliery

Traditionally the main explosives used at Kriel Colliery were Nitroglycerin based explosives like Ajax and Coalex. Powergel was used at one stage but became too expensive as Kriel was left as the only user. After an explosion at the AECl plant in September 1994, the manufacturer decided to cease the production of Nitroglycerin based explosives and Enercoal is now used.

Trials attempted by the mine in conjunction with AECl to determine the feasibility of using pumpable emulsion explosives were stopped in May 1995 because of unsatisfactory results.

A Comparison Between Ajax And Enercoal

Ajax is a Nitroglycerin based compound consisting mainly of Ammon gelignite, while Enercoal is a watergel sensitised by Monomethylamine Nitrate. The relative weight strength (RWS) of explosives is defined as the energy per unit mass relative to ANFO, which is used as a base and taken as 100%. The RWS of Ajax is 96% compared to the RWS of 79% of Enercoal. Ajax is clearly a much stronger explosives.

The Elements Of A Blast

Introduction

When the breaking energy is applied by electricity through the cutting drum of a mechanical miner, the breaking action can be continuously controlled and varied to

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achieve the desired effect. In contrast, when explosives are used, the breaking energy is produced almost instantaneously on detonation of the charges. Each hole contains only sufficient explosives to provide the energy required for breaking its designated burden. If the burden is too big due to incorrect marking, drilling or some other reason, blowouts will occur with the resultant poor face advance. It is thus of the utmost importance that every element of the blast is correctly done, especially when a relative weak explosive like Enercoal is used.

Provision of the cut

The purpose of the cut is to provide a free face to which the shot holes can break sequentially to excavate to the full length drilled. A free face is probably the most critical element of a blast, as the face cannot advance further than the cut provided. It is suspected that under-cutting of the cut is probably the single greatest reason for poor face advances. The jib length of the coal cutters in use at Kriel is 2,5m that enables a cut depth of 2,4m.

Controls must be instituted to ensure that cuts are provided to this full depth over the complete width of the heading and that duff is removed from the cut before the coal cutter is taken from the heading.

It is vital that the position of the cut in relation to the floor, i.e. 1m above the floor, be correct. If the cut is located too close to the floor, the burden of the bottom holes will be too little, while the burden of the row above the cut will be excessive. A poor advance is inevitable.

Stelling bars must be used to stabilise the coal cutter. These were previously in use but were removed.

Drilling the round

The correct drilling of each shot hole with respect to placement, angle and depth is essential for efficient breaking. It is important to relate the burden to the mass of explosives in the hole; overburdening will result in inadequate fracturing or blown out shots. Under burdening leads to excessive fracturing and flying debris. Although a guide based on the strength of the various explosives is available, the ideal hole pattern is so dependant on local conditions that is best established by trial and error. Whatever the required pattern, it is vitally important that drilling is accurately done. A hole meant to be drilled horizontally inclined at 5° at the collar, will end 220mm from its intended position and poor face advance will result.

A facedrill is used to drill 20 holes in 4 rows at a burden of 1 m spaced 1,5 m apart. According to the requirements of the law, the hole must be 15 cm shorter than the cut. Stoppers have recently been installed on the face drills to prevent over drilling.

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Charging the round

Shot holes must be cleaned from all drill cuttings before charging up commences. Apart from making it difficult to get the cartridges into the hole, drill cuttings between the cartridges can lead to misfires. To prevent deflagration of the charge, it is important to position the detonator at the toe of the hole and to trainload the charge to prevent air gaps or duff coming between the cartridges.

The base of the detonator should point in the direction of the charge column and it is important to attach the detonator lead wires securely around the cartridge to prevent the detonator being pulled out of the cartridge. Care must be taken to ensure that the correct charge is placed in each hole. At Kriel permission has been obtained to load each hole with a maximum charge of 1000 g.

Sufficient tamping, done with clay ampoules, is vital since it, not only prevent the ignition of a possible explosive atmosphere, but also it ensures containment of the hot high pressure gases in order to obtain maximum breaking power.

Timing the round

The last element of the blast is the correct timing of the round. It is important to obtain sequential firing of the holes in order to obtain a good face advance. Often one hears the expression "the face does not want to break". If, however, every step of the operation is correctly done, the face will break.

PRODUCTION FROM THE BLASTING SECTIONS

The target for each of the five blasting sections, operating on double shifts, is to produce 45 000 tons per month. In order to achieve this it is required that each section must blast, at an advance of at least 2,2 metres per blast, 15 faces during each shift. Face advances being attained, however, are far short from this requirement. Table 1 below depicts the face advance per blast achieved in the various sections during the past year (1996).

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Table 1 Face advance per blast achieved in 1996

MONTH	Sect 1	Sect 2	Sect 3	Sect 4	Sect 5	Average
Jan	1.85	2.00	2.24	1.96	1.86	1.98
Feb	2.01	2.08	1.97	2.09	1.78	1.99
Mar	1.71	1.91	1.87	1.88	1.64	1.80
Apr	1.70	1.96	1.66	2.17	1.73	1.83
May	2.06	1.83	1.79	2.28	1.85	1.95
Jun	1.58	1.68	2.06	1.97	1.69	1.80
Jul	1.95	2.00	1.85	1.71	1.77	1.86
Aug	2.13	1.93	1.89	1.76	1.78	1.92
Sep	1.98	1.68	1.96	1.61	1.74	1.78
Oct	2.13	2.15	1.94	1.90	1.90	2.00
Nov	1.62	1.68	1.31	1.35	1.59	1.50
Dec	1.95	2.41	1.81	2.15	1.96	2.04

The mine's average face advance for 1996 is shown on Figure 1 below:

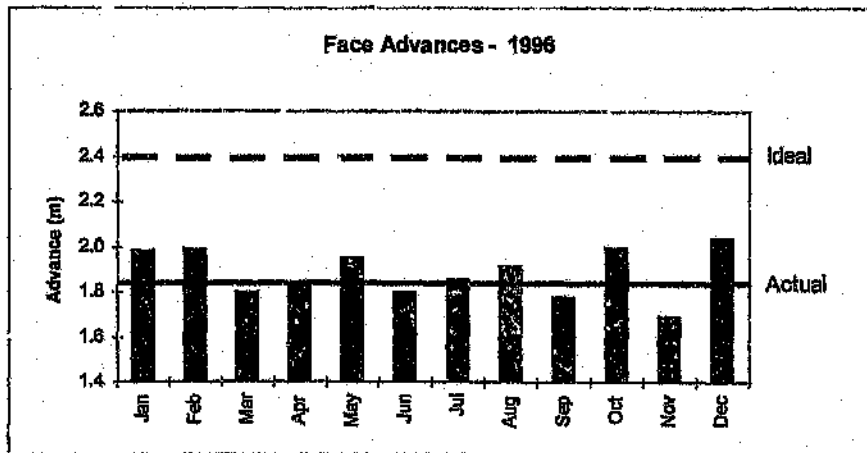


Figure 1 Average face advance in 1996

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The average face advance for 1996 was only 1,83 m. The production achieved from these five blasting sections during the past four calendar years is reflected below in Table 2.

Table 2 Production achieved from the blasting sections over the last four years

YEAR	TARGET	PRODUCTION	% ACHIEVED
1993	2 700 000	2 300 000	84,9 %
1994	2 250 000	2 100 000	90,8 %
1995	2 700 000	2 300 000	85,5 %
1996	2 700 000	2 300 000	86,9 %
TOTAL	10 300 000	9 000 000	86,9 %

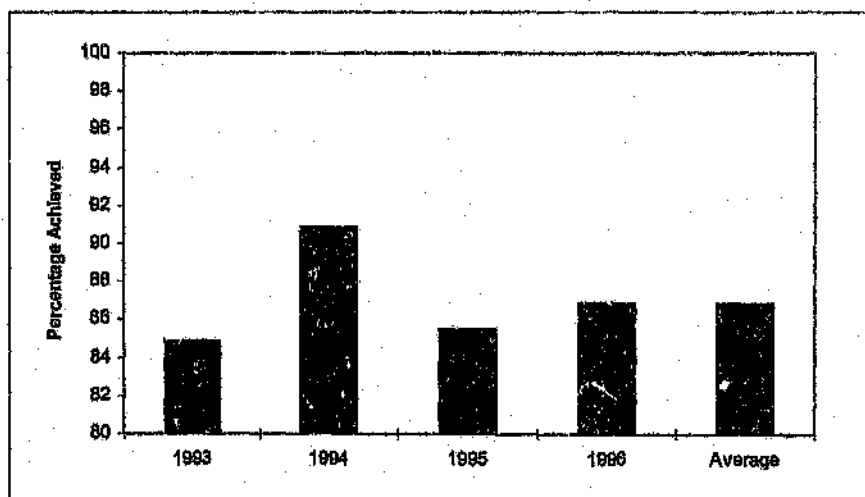


Figure 2 Production achieved from the blasting sections over the last four years

From the Table 2 and the Figure 2 above, it is evident that the blasting sections are not achieving the production targets. A major contributing factor for failing to reach the production targets, is the poor face advances being achieved.

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During 1996 the blasting sections produced a total of 2 300 000 tons against a target of 2 700 000 tons, a shortfall of 400 000 tons. The average face advance per blast for 1996 was 1,83m. During this time, 28 270 faces were blasted. If the average face advance could be increased to 2,3m, an increase in the annual production, as calculated below, can be attained:

No. of blasts:	28 270
Average road width:	6,0m
Average road height:	4,0m
Density:	1,5
Increased advance:	0,47m
	$= 28\,270 * 6,0 * 4,0 * 1,5 * 0,47$
Tonnage	$= 477\,000\text{ tons}$

This figure is well in excess of the shortfall of 400 000 tons experienced during 1996.

The tonnage of 2 400 000 obtained during 1996 at an average face advance of 1,83m was achieved with 28 270 blasts. This same tonnage could have been achieved by 23 300 blasts of 2,3m advance. This means that 5 000 faces were cut, drilled, charged and blasted additionally. At an average of 12 blasts per section per shift, this represents more than 40 working days.

TRIALS

Trial blasts were recently conducted in order to investigate the ways of improving the face advance situation. Originally, with Ajax, Coalex and Powergel in use, the traditional "line timing" that was used is shown in Figure 3. This timing pattern (standard) is characterised by identical detonators in each row of the face. From the bottom upward the following delay detonators are used:

- No. 0 detonator, which is instantaneous.
- No. 2 detonator, with a delay of 55 milliseconds
- No. 4 detonator, with a delay of 110 milliseconds
- No. 5 detonator, with a delay of 150 milliseconds

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5	5	5	5	5
4	4	4	4	4
2	2	2	2	2
0	0	0	0	0

Figure 3 Standard timing pattern

With the introduction of the less powerful Enercoal this timing pattern resulted in unsatisfactory face advance per blast. The major contributing factor for this drop in output was believed to be the inaccurate in drilling and cutting. Operators of the face drills and coal cutters were used to a system where less attention paid to drilling and cutting still produced satisfactory face advances due to the strength of Ajax.

A timing pattern similar to that used in a drag round was tried in all the blasting sections (see Figure 4). With this timing, the blast is detonated from the middle of the face, with instantaneous detonators, outwards to the corners, where the No. 5 detonators are initiated last. Some improvement in coal outputs was recorded but still the blasting efficiencies (ratio of the cut depth to the breaking depth after the blast) were not optimal.

5	4	2	4	5
4	2	0	2	4
2	0	0	0	2
5	2	0	2	5

Figure 4 Timing pattern attempted (drag round)

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A request for an exemption to use longer delays was introduced to office of the Inspector of Mines. After the exemption was obtained, detonators with delays longer than the maximum permitted 150 milliseconds were used in 200 trial blasts. The No. 6, No. 7 and No. 8 detonators were introduced (see Figure 5). These detonators have delays of 200, 250 and 300 milliseconds respectively. The longer delays are intended to allow for greater relief of the burden. A number of configurations of placing the individual delays in blastholes was studied and tried. The pattern given in Figure 5 below emerged as the most suitable and optimal.

8	7	7	7	8
6	4	4	4	6
4	3	3	3	4
2	0	0	0	2

Figure 5 Optimal timing pattern achieved

RESULTS

After the introduction of the above longer delay-timing pattern in the pilot blasting section chosen for the trial (M11) at the end of November, the face advance per blast increased drastically. After each blast, the advance was carefully measured and recorded. During the investigation, 153 blasts were measured at an average advance of 2,34 metres. Face advances of 2,4m or more were frequently obtained as shown in Figure 6 below.

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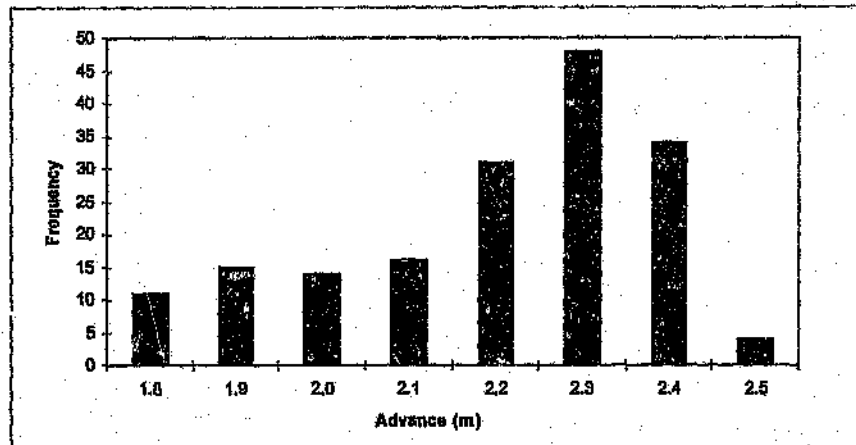


Figure 6 Distribution of the resulting advances per blast recorded

The average face advance per blast in this section during December 1996, as measured by the surveyors was 2,41 metres.

It is worth noting that while the trials were being conducted, a very high level of supervision was being maintained. The section was visited regularly by the managers. Everybody concerned was aware that deviations from requirements would not be tolerated. Care was taken to ensure that the cut was provided to the full depth and that drilling was accurately done. Although the new timing pattern also contributed to the improvement in the face advance, it is contended that the increase in the level of supervision was another major factor that played in favour of the better face advances achieved.

CONCLUSIONS

- Current production targets can be achieved without additional costs if proper face advances can be attained.
- Sub-standard cutting and drilling is always the major reasons for the poor face advances.
- The existing standard of supervision needs attention, as it is inadequate.
- The trials in M11 section, with the delayed timing of the blast, proved successful.

RECOMMENDATIONS

- Operators of coal cutters and face drills need to be re-trained. A course in the "getting back to basic mining principles" program is recommended. Initial training could be conducted in the training centre, where the importance of correct cutting and drilling should be stressed. Practical training should be done in a designated section where the best operators should be employed as trainers.
- Gangs need to be motivated and the importance of quality work rather than quantity must be emphasised. They need to understand that incorrect drilling and cutting lead to poor face advances and production, and thereby decrease their bonuses.
- The standard of supervision must improve. Sub-standard advances must not be accepted.
- All coal cutters should be equipped with stelling bars to stabilise the machine during cutting.
- The new method timing of the blast should be introduced in all blasting sections.

LASTEST DEVELOPMENTS

The satisfactory results obtained from the 200 trial blasts discussed in this paper have motivated an application to the Department of Mineral and Energy Affairs for a permanent exemption to use the longer delay detonators. The Inspector of Mines has approved the application (a copy of the exemption is appended to this paper). This timing round has been introduced in all the blasting sections at Kriel Colliery.

ACKNOWLEDGEMENTS

The assistance of the staff at Kriel Colliery and Mr.K. Kabongo, lecturer at the University of Witwatersrand, in preparing this report is greatly appreciated.

REFERENCES

- AECI (1985), *Blasting Practices in Collieries, Explosives Today*. AECI, Series 2, No.26, (Revised Edition).
- Lubbe, C. H. (1995), *Optimisation of delay timing in underground coal blasting*. M.Sc. Thesis, University of the Witwatersrand, Johannesburg.

Appendices

APPENDIX G

Results of field investigation: Underground blasting

Recent development of the project

Notes

The work described in this thesis and in the preceding pages has resulted in a change in interpretation of the Mining Regulations. The letter of exemption, which follows, shows that longer delays are now permitted under specified conditions.

The Mine Manager
Kriel Colliery
P O Box 27
KRIEL
2271

M.O. POULTNEY

1997-03-20

OT11/5/2/9-S107

Sir

EXEMPTION FROM MINERALS ACT REGULATION 9.27

In reply to your letter of application, referenced mc/ff/HOD/mc7, dated 1996-12-30, partial exemption in terms of Minerals Act Regulation 3.15.1 is hereby granted from the requirements of Regulation 9.27 read with the First Schedule, to the extent that short period delay detonators ranging between 0 and 8 may be used, subject to the following conditions being observed:

1. The charge per shot hole may not exceed 800 grams.
2. Suitable arrangements must be made and maintained to ensure a sufficient quantity of intake air in the face region to prevent the build up of methane. Any heading being advanced beyond 20 metres must be positively ventilated with at least 0,2 cubic metres per second for each square metre of the average cross sectional area of each of the faces being advanced of fresh air at a distance of not more than 10 metres from the face.
3. Every shot hole must be properly tamped with an approved tamping material in accordance with the requirements of the Regulations.
4. The shift boss appointed in terms of Regulation 2.15.1 must ensure by personal observation that the following Regulations are strictly adhered to :

Regulation 8.10.12
8.10.14
8.10.30
10.6.6
10.8.1.1
10.8.2
10.24.4

5. Such shift bosses must daily, at the end of the shift, report in their log books observations regarding compliance with the foregoing Regulations. The Section Manager or Mine Overseer must by daily inspection of the log books satisfy themselves that the stipulated conditions are properly adhered to.

This exemption is valid for a period of one year from the date of this letter and may be cancelled or amended at any time should such action be considered necessary.

All persons concerned must be made fully conversant with the terms of this exemption copies of which shall be readily available to them.

Please acknowledge receipt of this letter.

Yours faithfully


PRINCIPAL INSPECTOR
MPUMALANGA REGION

TIKSTERSNCHAP-09-27ALETTERALET-09-27

The Senior Inspector
MPUMALANGA REGION

**APPLICATION FOR PARTIAL EXEMPTION TO USE OF LIMITED
DETONATORS MINERALS ACT REGULATION 9.27 KRIE COLLIERY**

1. The Mine Manager's letter of application, referenced mc/ff/HOD/mc7, dated 1996-12-30 refers.
2. Application is made for partial exemption from the requirements of Minerals Act Regulation 9.27 to the extent that short period delay detonators ranging between 6 and 8 may be used.
3. A previous partial exemption has been granted for a trial period of 200 faces and these trials have now successfully been completed.
4. The results were such that the average face advance increased by 0,5 m, and at the same time damage to the roof and sidewalls decreased due to the better timing which in turn increased the overall safety conditions.
5. During the trial period nothing untoward was reported.
6. This application was discussed with the various employee organisations and their consent has been obtained.
7. Normal clay tamping will be used.
8. Similar exemptions have been granted in the past with no untoward occurrences.
9. Considering the above, it is recommended that exemption be granted as per the attached draft letter.


M.O. POULTNEY
INSPECTOR OF MINES
MPUMALANGA REGION

G19

1997-03-20

an **AMCOAL** company

Appendices

Kriel Colliery

Co. Reg. No. 010673006

A Division of Amcoel Colliery and
Industrial Operations Limited

Incorporated in the Republic of South Africa

P.O. Box 27

Kriel

2271

Telephone Kriel 017-6171111

Facsimile Kriel 017-6483910

In reply please quote
AK/111111/1111

30 December 1996

Department of Mineral and Energy
Regional Director
Private Bag X7279
WITSBANK
1035

Attention: Mr D. Du Plooy

TO CARRY OUT BLASTING OPERATIONS USING DETONATORS NO. 6, NO. 7
AND NO. 8 ON A PERMANENT BASES IN ALL THE BLASTING SECTIONS AT
KRIEL COLLIERY

In terms of Regulation 3.15.1, exemption from Regulation 9.2.7
"Permitted Explosives," permission is requested to use the
following detonators No. 6, 7 and 8 as part of our initiating
system in the Blasting Sections.

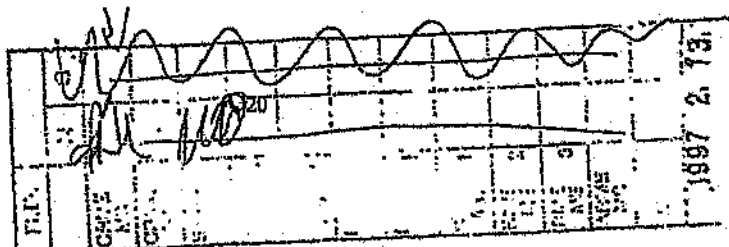
The motivation behind this is as follows:

Trials were conducted in M11 Section 2 at North West Shaft from
the 18 November 1996 to the 30 December 1996. During this time
198 faces were blasted using a pattern recommended by ASCI. (All
Dets were used)

The results were such that the average face advance increased
from 1.8m to an average of 2.3m, giving an additional face
advance of $\pm 0.5m$, this in tons, added an extra 20 tons to each
blast for the same amount of effort and cost as before. For a
22 shift month and an average of 20 faces blasted per shift this
increased the tons produced by $\pm 8\ 000$ tons.

Due to the better timing, the rib side conditions improved and
also less damage was evident to the roof thus overall safety
conditions improved with the use of these detonators.

The long term advantage will be in cost per ton mined and tons
per case of explosive will increase proportionally.



DIRECTORS: D Rankin (Chairman), J P Bloemestein, I W Boye (Treasurer), H N Dwyer, D Haskins, J A Laybourne (Ez. Int.), J S Loria, A E Pearson (Ez. Int.),
M R Prid, D J Rison (Ez. Int.), M B Stacey, C Sutherland, K J Trueman (Ez. Int.), R R L G Wicks (Ez. Int.), D B Wicks

QCC

Appendices

As requested in your initial Exemption Reg. No. 0711/5/2/9-5107,
the following information is submitted.

Trial Results

<u>Working Place</u>	-	M11 Sect 2 North West Shaft Kriel Colliery
<u>Test Period</u>	-	18-11-1996 to 30-12-1996
<u>No. of Faces Blasted</u>	-	198 (All detonators were used up)
<u>Mining Dimensions</u>	-	Mining Height Avg 4.1m Bord Width Avg 6.5m Face Adv Avg 2.3m

The average face advance for M11 since April 1996 was 1.63m.

The average face advance for all blasting sections was 1.83m for 1996.

As can be seen from these statistics the increase in face advance using the detonators was $\pm 0.47m$.

Test Patter Used - As per diagram

Problems Encountered - None

Employees Comments - Happy to use them as the production increased so their bonus also increased.

ABCI Comments - During the trial a Technical Representative checked on the use of these detonators and had no problems.

Safety - Improved Roof and Rib side conditions.

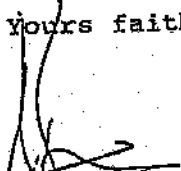
Costs - Overall cost saving on explosives and detonators.

Explosive Efficiency - Increase proportionally

In conclusion, the trials conducted using these detonators were successful and the overall performance was an increase in production at no additional cost. The implementation of these detonators on a permanent bases in all the blasting sections will not only improve production, reduce costs but will also improve the safety aspect in the sections.

Hoping that this application meets with your approval.

Yours faithfully


R. C. BERRY
Mine Manager

BLASTING ROUND TIMING

8	7	7	7	8
6	4	4	4	6
4	3	3	3	4
New round				
2	0	0	0	2

5	5	5	5	5
4	4	4	4	4
2	2	2	2	2
Previous round				
0	0	0	0	0

5	4	2	4	5
4	2	0	2	4
2	0	0	0	2
Present round				
5	2	0	2	5

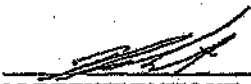
Appendices



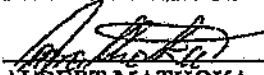
^{Appendix}
The following Unions and Safety Representatives have been consulted and have agreed that the use of detonators 6, 7 & 8 may be used in all the blasting sections on a permanent basis :

UNION REPRESENTATIVE'S SIGNATURES IN AGREEMENT :

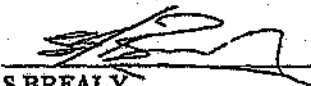
MINE WORKERS UNIONS


M BEZUIDENHOUT

NATIONAL UNION OF MINEWORKERS


ALBERT MATHOKA

OFFICIALS UNION

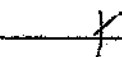

S BREALY

N.E.T.U.


NIC WILKEN

Safety Representatives signatures in agreement :

M NTOSHE



J MOKGOPE

JOSEPH

P NICADIMENG

FRANCIS L. NICADIMENG

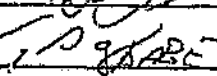
B MAJARA

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M MAKOETLANE



T GRABES



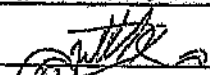
J ZONDI



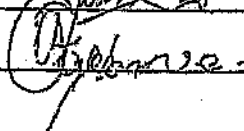
E NDLUVA

ANDRUS LERUE

S MGWADISO



F MAGABANE



References

REFERENCES

References

Aler, J. T. and Wieland, M. S. (1991), *The waiting time recovery test*, Proceedings, 7th Annual Symposium on Explosives and Blasting Research, Las Vegas, Nevada, USA.

Allsman, P. L. (1960), *Analysis of explosive action in breaking rock*, AIME, Vol. 217, pp. 468 - 478.

Anderson, D. A.; Winzer, S. R. and Ritter, A. P. (1984), *Time-history of principal strains generated in rock by cylindrical explosive charges*, Proceedings, Symposium Rock Mechanics in Productivity, Protection, AIME, Illinois, USA.

Ash, R. L. (1973), *The influence of geological discontinuities on rock blasting*, Ph D. thesis, university of Minnesota, USA.

Atlas Powder Company (1987), *Explosives and rock blasting*, Atlas Powder Company, Dallas, Texas, USA.

Badal, R. (1995), *Rock blasting with discontinuities*, International book traders, Delhi, India.

Bieniawski, Z. T. (1979), *Suggested methods for determining the uniaxial compressive strength and deformability of rock materials*, ISRM, International Journal Rock Mechanics Mining Science & Geomechanics, Vol. 16, No. 2, pp. 135 - 140.

Bligh, T. P. (1974), *Principles of breaking rock using high pressure gases*, Proceedings, 3rd International Congress on rock Mechanics, Denver, IIB, US National Academy of Science, pp. 1421 - 1427.

References

Brady, B. H. G. and Brown, E. T. (1985), *Rock mechanics for underground mining*, George Allen & Unwin (Publisher) Ltd., London, UK.

Brinkman, J. R. (1986), *Rock fragmentation and heave from stopping blastholes during controlled gas penetration tests*, Proceedings, 17th International Congress on High speed Photography and Photonics, Pretoria, RSA, SPIE Vol. 674, pp. 365- 380.

Brinkman, J. R. (1987), *Separating shock wave and gas expansion breakage mechanisms*, Proceedings, 2nd International Symposium on Rock Fragmentation by blasting, Keystone, Colorado, USA.

Brinkman, J. R. (1989), *Rock breaking characteristics of various mining explosives in small diameter holes*, Proceedings, 5th Annual Research Symposium on Explosives and Blasting Technique, SEE, New Orleans, Louisiana, USA.

Brinkman, J. R. (1990), *An experimental study of the effects of shock and gas penetration in blasting*, Proceedings, 3rd International Symposium on Rock Fragmentation by Blasting, Brisbane, Australia.

Britton, R.; Konya, C. J. and Skidmore, D. R. (1984), *Primary mechanism for breaking rock with explosives*, Proceedings, Symposium on Rock Mechanics in Productivity, Protection, AIME, Illinois, USA.

Bond, F. C. (1952), *The third theory of comminution*, Transactions AIME 193, 484.

Borovikov, V. A. and Vanyagin, I. F. (1995), *Modelling the effects of blasting on rock breakage*, Balkema, Rotterdam, Netherland.

References

Boukharov, G. N. (1972), *Theory of rock blasting*, Lecture notes, Rock Blasting, El-Tabbin Mining and Metallurgical Institute for higher studies, Cairo, Egypt.

Chadwick, J. (1994), *South African coal*, Mining Magazine, September 1994.

Chiappetta, F. R.; Mammelle, M. E. and Postupack, C. (1988), *Causes and recommendations for controlling coal damage when blasting overburden*, Annual Conference on explosives and Blasting Techniques, S.E.E., Anaheim, California, USA.

Chiappetta, F. R. (1994), *Causes and recommendations for controlling coal damage when blasting overburden*, Proceedings 4th Annual BME South African Surface Mining Seminar, Pretoria, RSA.

Coleman, P. (1994), *Notes on drilling monitoring - An example of information technology applied to mining*, 4th Annual BME Southern African Surface Mining Seminar, Pretoria, RSA.

Comeau, W. (1993), *La physique et la mecanique des transferts d' energie lors d' abatage des roches a l' explosif*, Proceedings, 3rd International Summer Seminar on Rock Blasting, Ales, France.

Cook, M. A. (1958), *The science of high explosives*, Rheinhold, New York, USA.

Cook, M. A. (1971), *The science of high explosives*, Robert E. Krieger Publishing Co. Inc., New York, USA.

References

Cunningham, C. V. B. (1970), *A study of strain wave phenomena associated with blasting*, MSc thesis, University of the Witwatersrand, unpublished.

Cunningham, C. V. B. (1987), *Fragmentation estimates and the Kuz-Ram model: Four years on*, Proceedings, 2nd International Symposium Rock Fragmentation by Blasting, Keystone Colorado, USA, pp. 475-487.

Cunningham, C. V. B. (1991), *Blasting-off-the-Solid project*, Report to the AMCOAL steering committee, unpublished.

Da Gama, C. D. and Nelson, C. R. (1971), *The formation of radial cracks in rock blasting analyzed in terms of the modified Griffith's theory*, Proceedings, AIME Symposium On Dynamic Rock Mechanics, Missouri, Rolla, USA.

Delaporte, C. and Heizmann, B. (1995), *The impact of fines at Kreinkopje Colliery and Landau Colliery*, Confidential report, unpublished.

Deree, D. U. and Miller, R. F. (1966), *Engineering classification and Index properties for intact rock*, Technical Report No. AFWL-TR-65-116, Air Force Weapons Lab., Kirtland AFB, New Mexico.

Du Plessis, M. P.; Wallace, B.W. and Tipping, N. (1987), *The performance of small and intermediate diameter explosives*, AECI / ICI Internal report, unpublished.

References

Evans, I. and Pomeroy, C. D. (1966), *The strength, fracture and workability of coal*, National Coal Board, Mining Research and Development Establishment, Pergamon Press Ltd, U.K.

Fogelson, D. E., Atchison, T. C. and Duvall, W. I. (1959), *Propagation of peak strain and strain energy for explosion-generated strain pulse in rock*, Proceedings, Quarterly Colorado School of Mines, Vol. 54, No. 3, pp. 273 - 284.

Hagan, T. N. (1979), *The control of fines through improved blast design*, Proceedings, AIMM No. 271 pp. 9 - 20.

Holmberg, R. (1974), *Charge calculation in bench blasting*, Sve De Fo Reports DS 1974:4, Stockholm, Sweden.

Holmberg, R. (1993), *Rock damage and some recent developments to control rock damage*, Proceedings, International Summer Seminar on Rock Blasting, Ales, France, pp. 194 - 206.

Hukki, R. T. (1975), *The principle of comminution: an analytical summary*, Engineering Mining Journal, Vol. 176, No. 106.

ISRM commission on standardisation of laboratory and field tests (1978), *Suggested methods for the quantitative description of discontinuities in rock masses*, International Journal of Rock Mechanics and Mining Sciences & Abstracts, Vol. 15, No. 6, pp. 319 - 368.

Johansson, C. H. and Persson, P. A. (1970), *Detonics of high explosives*, Academic press, New - York, USA.

References

Just, G. D. (1979), *Rock fragmentation in blasting*, C.I.M. bull, Vol. 72 No. 803, pp. 143 - 148.

Ireland, K. S., and Stern, M. S. (1987), *Report of results of Penobel trials at Goedehoop Colliery*, AECI Blast Consult, unpublished report.

Isakov, A. L. and Sher, E. N. (1983), *Theoretical research and modelling of directed cracks propagation in borehole blasting*, Proceedings, 1st International Symposium on Rock Fragmentation by Blasting, Lulea, Sweden.

Kabongo, K. K. (1985), *Optimization of blasting parameters - A study of rock fragmentation in bench blasting*, MSc thesis, UNZA, unpublished.

Kabongo, K. K. (1993), *Influence of stemming geometry on fragmentation*, Proceedings, International Seminar on Rock Blasting, Ales, France, pp. 431-438.

Kabongo, K. K. (1995a), *Model of fines generation in coal blasting*, Proceedings, 4th International Symposium on Mine Planning and Equipment Selection, Calgary, Canada. pp. 945 - 952.

Kabongo, K. K. (1995b), *Blasting-induced damage in coal*, Proceedings 21st International Conference on Blasting Technique, SEE Tennessee, Nashville, USA, pp. 203 - 213.

Kabongo, K. K. (1995c), *Low density ammonium nitrate fuel oil to improve gold recovery*, SAIMM, Vol. 95 No. 1, pp. 1 - 6.

References

Kabongo, K. K. and Darkwah, L. (1996), *Reduced coal fines in blasting*, SME Preprint 96-125.

Kabongo, K. K. and Du Plessis, J. J. L. (1997), *Blasting technique for reduced coal fines*, CSIR Project report No. ESH 97-0004, Confidential report, unpublished.

Kabongo, K. K. (1997), *Impact of inherent fractures on fragmentation of blasted coal*, IBC Conference on Efficient Blasting, Pretoria, RSA.

Kick, F. (1885), *Des gesetz der proportionalem widerstand und seine anwendung*, Felix, Leipzig.

Kim, D. S. and Mc Carter, M. K. (1993), *Influence of shock damage on subsequent comminution of rock*, Proceedings, Fragblast - 1993, pp. 63 - 70.

Konya, C. J.; Britton, R. and Lukovic, S. (1987), *Charge decoupling and its effect on energy release and transmission for one dynamite and watergel explosive*, Proceedings, 3rd Symposium on Explosives and Blasting Research, S.E.E., Miami, USA, pp. 14 - 25.

Kou, S. Q. and Rastan, A. (1992), *Burden related to blasthole diameter in rock blasting*, International Journal Rock Mechanics Mining Science & Geomechanics Abstracts, Vol.29, No. 6, pp. 543 -553.

Kumao, H. (1961), *Fragmentation of rock through blasting and shock wave theory of blasting*, Proceedings, Quarterly of Colorado School of Mines, Vol. 55, No.1, pp. 190 -209.

References

Kutter, H. K. (1967), *The interaction between stress waves and gas pressure in the fracture process of underground explosion in rock, with particular application to pre-splitting*, Ph D thesis, University of Minnesota, USA.

Kutter, H. K. (1970), *Stress analysis of a pressurized circular hole with radial cracks in an infinite elastic plate*, International Journal of Fracture Mechanics, No. 6, pp. 233 - 247.

Kutter, H. K. and Fairhurst (1971), *On the fracture process in blasting*, International Journal of Rock Mechanics & Mining Sciences, Vol. 8, No. 3, pp. 181 - 202.

Kutter, H. K. and Kulozik, R. G. (1990), *Mechanics of blasting in a discontinuous rock mass*, Mechanics of jointed faulted rock, Balkema, Rotterdam, Netherland, pp. 295 - 299.

Kutuzov, G. N. (1994), *Rock blasting - Industrial blasting practices*, Publishing House of Moscow State Mining University, Moscow, Russia.

Ladds, C. G.; Davey, O. L.; Rorke, A. J.; Jordan, D. J. P. and Cunningham, C. V. B. (1992), *Report of blast monitoring at ATCOM for improved blasting efficiency*.

Landman, G. (1993), *BOTS - Background Information*, Communication at Knel Colliery (unpublished).

Langefors, U. and Kilhstrom, B. (1963), *The modern technique of rock blasting*, John Wiley & Sons Inc., New York, USA.

References

Leiper, G. F. and Du Plessis, M. P. (1990), *Describing explosives in blast models*, ICI PLC Ardeer site, Stevenston, Ayrshire, UK, KA20 3LN.

Lilly, P. A. (1986), *An empirical method of assessing rock mass blastability*, AusIMM/IE, Aust Newman Combined Group, Large Open Pit Mining Conference.

Lizotte and Scroble (1994), *Geological control over blast fragmentation*, CIM Vol. 87 No. 983, pp. 57 - 71.

Lownds, C. M. (1986), *The strength of explosives*, The planning and operation of open-pit and strip mines, Ed. Deetlefs, J. P. SAIMM, Johannesburg, pp. 151 - 159.

Lubbe, C. H. (1995), *Optimisation of delay timing in underground coal blasting*, MSc thesis, University of the Witwatersrand, Johannesburg, RSA.

Lubbe, C. H. and Kabongo, K. K. (1995), *Optimisation of inter-holes timing in underground coal blasting*, Communication, Wits Mining Research Seminars, unpublished.

Macgregor, I. M. (1983), *Preliminary results on the relationship of coal petrology to coal cuttability in some South African coals*, Special publication of the geological society of South Africa, Vol. 7, pp. 117-128.

*Maynard, B. C. (1990), *A blast design mode using the inherent fragmentation of rock mass*, CIM bulletin, pp. 71.

References

- Mc Carter, K. K. and Kim, D. S. (1993), *Influence of shock damage on subsequent comminution of rock*, proceedings, Fragblast-4, Balkema, Rotterdam, Netherlands, pp. 63 - 70.
- Mc Carthy, T. S. (1981), *A Fortran IV program for rapid graphic display of sedimentary borehole log data*, Trans. Geol. Society of South Africa, Vol. 84, pp. 271 - 279.
- Mel'nikov, N. V. (1940), *Utilisation of the energy of explosives and the fragment size of rock in blasting operations*, Gorn.Zh., No. 5.
- Mel'nikov, N. V. and Marchenko, L. N. (1971), *Effective methods of application of explosion energy in mining and construction*, Proceedings 12th Symposium on Rock Mechanics, AIME, pp. 359 - 378.
- Mel'nikov, N. V.; Marchenko, L. N.; Zharikov, I. F. and Selnov, N. P. (1978), *Blasting methods to improve rock fragmentation*, Acta Astronautica, Pergamon Press Ltd, U.K. Vol. 5, pp. 1113 - 1127.
- Morrel, S. et al. (1992), *The prediction of powder draw in comminution machines*, Comminution-Theory and Practice, pp. 133 - 247, SME, Littleton, Colorado, USA.
- Nagy, J., Hartmann, I., Christofel, F. P. and Seller, E. C. (1952), *Experiments on multiple short-delay blasting of coal (part II)*, USBM, Report of Investigations 4875, May 1952.

References

- Nicholls, H.R. and Duvall, W.I. (1963), *Effect of characteristic impedance on explosion-generated strain pulses in rock*, Proceedings, 5th Symposium on Rock Mechanics, Minnesota, USA, Pergamon Press, pp. 331-346.
- Nielson, K. (1995), *Blasting-Crushing-Grinding: Evaluation of an integrated comminution system*, Communication Wits Mining Research Seminar on Rock Blasting, Johannesburg, RSA.
- Ouchterlony, F. (1974), *Fracture mechanics applied to rock blasting*, *Advances in rock mechanics*, 3rd ISRM congress, 11-B Denver Co, pp. 1377 - 1383.
- Ouchterlony, F. and Zongqi, S. (1983), *New methods of measuring fracture toughness on rock cores*, Proceedings, 1st International Symposium on Rock Fragmentation by Blasting, Lulea, Sweden.
- Persson, P.; Hornberg, R. and Lee, J. (1994), *Rock blasting and explosives engineering*, CRC Press, Inc. USA.
- Petrosyan, M. I. (1994), *Rock breakage by blasting*, Elsevier, Rotterdam, Netherland.
- Priest, S. D. and Hudson, J. A. (1976), *Discontinuity spacing in rock*, *International Journal Rock Mechanics Mining Science*, Vol. 13, pp 135 - 148.
- Qian Liu and Katsabanis, P. D. (1993), *A theoretical approach to the stress waves around a borehole and their effect on rock crushing*, Proceedings, Fragblast-4, Balkema, Rotterdam, pp. 9 - 16.

References

- Rossmanith, H. P. (1994), *Fracture mechanics*, Lecture notes, Department of Mining Engineering, University of Witwatersrand, Johannesburg, RSA.
- Rossmanith, H. P. (1997), *The on-set of fractures around a blasthole*, Personal communication.
- Rzhevsky, V. V. and Novik, G. (1971), *The physics of rocks*, Mir Publisher, Moscow, USSR.
- Rzhevsky, V. V. (1978), *Process of openpit mining*, Nedra, Moscow, USSR.
- Sadwin, L.D., Okubo, S. and Kennedy, J.E. (1963), *Explosive performance as related to blasting*, Proceedings, 5th Symposium on Rock Mechanics, Minnsota, USA, Pergamon Press, pp. 307-312.
- Sarracino, R. (1993), *Physics of explosives*, Lecture notes, Department of Mining Engineering, University of the Witwatersrand, Johannesburg, RSA.
- Schultz, T. S (1972), *Investigation into splitting effects in explosively loaded small diameter holes with stress raisers*, MSc thesis, University of the Witwatersrand, unpublished.
- Scott, A.; Chitombo, G. and Kleine, T. (1993), *The challenge of the prediction and control of fragmentation in mining*, Proceedings, Fragblast-4, Balkema, Rotterdam, Netherland.
- Seligmann, P. C. and Shultz, T. S. (1972), *Effects of an applied stress field on fracture propagation from notched boreholes*, Communication,

References

Department of Mining Engineering, University of Witwatersrand, unpublished.

Singh, B. and Pal Roy, P. (1993), *Blasting in ground excavation and mines*, Balkema, Rotterdam, Netherland.

Singh, S. P. (1995), *Mechanism of cut blasting*, Institution Mining & Metallurgy, Transaction, vol. 104 (A125-94) Sep/Dec 1995, pp. A134-A138.

Slaughter, S., Turner, R., Capsain, J. and Grey, D. (1992), *Investigation of coal fines at Coal and Allied Hunter Valley Mine*, Proceedings, 3rd Large open Pit Mining Conference, Mackay.

Stagg, M. S.; Rholl, S. A and Ottermess, R. E. (1989), *The effect of explosive type and delay between rows on fragmentation*. Proceedings, 15th Conference on Explosives and Blasting Technique, New-Orleans, USA, pp. 353-366.

Steverding, B. and Lehnigk, S. H. (1976), *The fracture penetration depth of stress pulses*, International Journal Rock Mechanics, Mining Sciences & Geomechanics., pp. 75 - 80.

Thorpe, L. and Pedroncelli, E. J. (1986), *Coal sizes produced by various mining methods*, COMRO, Research report NO. 8/86 CM1E30.

Udy, L. L. (1994), *Hard rock blasting - Soft rock blasting, which explosive to use*, Proceedings, 4th Annual BME South African Surface Mining Seminar, Pretoria, RSA.

References

Van Zyl, P. G. (1993), *Report on current drilling and blasting practice at Kleinkopje Colliery*, Confidential report, unpublished.

Von Rittinger, P.R. (1867), *Lehrbuch der aufbereitungskunde*, Ernst and Koorn, Berlin, Germany.

Vutukuri, V. S., Lama, R. D. and Saluja, S. S. (1974), *Handbook on mechanical properties of rock*, Vol.1, pp 38.

Whittaker, B. N.; Singh, R. N. and Sun, G. (1992), *Rock Fracture Mechanics Principles, design and applications*, Elsevier, Netherland.

Wieland, S. M. (1987), *The desensitisation and malfunction of coal-mine explosives*, Proceedings 13th Conference on Explosives and Blasting Technique, Miami, Florida, USA.

Williams, K. (1991), *Optimisation of detonator delay in underground colliery blasting*, Internal Report, Bank Colliery, unpublished.

Wills, B. A. (1988), *Mineral processing technology: An introduction to the practical aspects of ore treatment and mineral recovery*, Pergamon press, Singapore.

Zipf, R. K. and Bienlawski, Z. T. (1989), *A fundamental study of respirable dust generation in coal*, *Mining Science and Technology*, Elsevier Science Publishers B. V., Amsterdam, Netherland.

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