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A study of the Correlation Between Dragline Diggability and Blasting Technology at Syferfontein Colliery

Mensah Frimpong

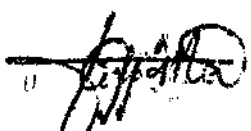
A project report submitted to the Faculty of Engineering, University of the Witwatersrand, Johannesburg, in partial fulfilment of the requirements for the degree of Master of Science in Engineering.

Johannesburg, 1994.

DECLARATION

I declare that this project is my own, unaided work, except where otherwise acknowledged. It is being submitted for the degree of Master of Science in Engineering in the university of the Witwatersrand. It has not been submitted before for any degree or examination in any other university.

Signed this 25 day of July 1994



Mensah Frimpong

ABSTRACT

Given the high capital intensity of the dragline operation, the mine can not afford to tolerate fragmentation problems with its concomitant low dragline productivities. At Syferfontein colliery very large boulders and completely unfragmented sections resulting in low digging efficiency of the dragline were encountered in the interburden blast.

This project investigates the causes of the blasting related problems with an intent of improving dragline productivity at the minimum possible cost. This is achieved through blast monitoring, evaluation of the existing practice (blast design) using empirical relations, fragmentation assessment using a model, analysis of operators performance and dragline availabilities. Dragline productivity is evaluated in the light of the various productivity indices, viz, fill factor, fill time and BCM/H. The results indicated explosive incompatibility with the blasting environment and questionable practices regarding blast design and Delay performance. Engineering field controls such as reduction in drill pattern, reduction of VOD of explosive and improvement in dragline utilisation are found necessary.

Contributing 61% to the total cost, concentrating on reducing dragline cost must receive preference over drilling and blasting cost as it would produce the most tangible advantage in any cost reduction effort. Regression equations relating cost and productivity are established.

Dedicated to my mother Veronica Addai

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LIST OF SYMBOLS

Quantity	Symbol
Allowable deviation	X_A
Burden	B
Average fragment size	X_c
Charge length	L
Degrees of freedom	V_f
Delay interval	D_i
Detonation pressure	P_d
Explosive bulk density	B
Explosion pressure	P_e
Hole diameter	d
Hole length	H
Kilograms of Explosive	Q
Mean deviation	M
Nonel Type detonator	NTD
Number of samples	N
Operators' productivity	P
Percentage availability	A
Percentage utilisation	U
Performance index	PI
Population mean	u
Powder factor	K
Relative weight strength of explosive	E
Relative Weight strength of TNT	E_{TNT}
Rock Hardness Index	A
Screen size (fragment size)	X
Spacing	S
Standard deviation	S_d
Standard deviation of drilling accuracy	W
Uniaxial compressive strength	UCS
Uniformity index	n
Velocity of detonation	VOD
Young Modulus	Y

CHAPTER 1

INTRODUCTION

1.1 Definition of the Problem and Objectives

The blasting operation at Syferfontein Colliery at times results in fines, or big boulders and unfragmented sections. A high proportion of fines indicates excessive use of explosives which, in addition to escalating the cost, also unnecessarily increases the violence of blast on the mine (Brinkman, 1990). For the dragline operation this situation culminates in a low bucket fill factor. Concomitant to this is low dragline productivity (Bcm/H) and higher production costs. In a similar vein boulders, which are usually the result of under utilisation of explosives and poor blast design are also undesirable as they lead to low bucket fill factors and at times necessitate secondary breakage.

The following problems encountered in one way or another by the mine are also worthy of mention:

- misfires or "table top" effect as well as blocky fragmentation and oversize encountered in some blasts
- high vibrational levels resulting from excessive confinement of blast or high charge mass detonated per delay
- low dragline digging penetration efficiency resulting in rapid wear and tear of bucket teeth and high energy consumption of the dragline which is a detrimental to dragline economics

- choice of explosive type and form to ensure efficient blasting and hence good dragline productivity
- high cost of production

Given the capital intensity of the dragline operation, the mine is often confronted with the problem of which design to use to achieve optimum fragmentation in order to yield efficient dragline diggability.

This project addresses the foregoing problems with specific focus on:

- assessment of the entire operation with respect to dragline productivity
- review of the blasting operation on the mine considering the type of explosive used and results of fragmentation analysis
- blast monitoring through velocity of detonation (VOD) and high speed photographic measurement
- operating cost analysis with specific emphasis on the economic implications of changes in design and the obvious cost reduction through improved productivity.

1.2 Scope of the Work

The study was confined to interburden blasting at the west bank on strips 5 and 6, and only one dragline's productivity has been studied. It must, however, be emphasised that this project does not attempt in any way to design a blast. It merely reviews the existing practice and suggests changes that could enhance dragline productivity with concomitant reduction in the overall cost. Cost consideration exclude labour costs and estimates are based purely on the author's own judgement from the available information. The project does not

cover the technology of the dragline.

1.3 Organisation of the Work

The text is divided into eight chapters. The first three chapters are introductory in nature with the first chapter highlighting the background of the problem, sources of data and treatment of results. The second chapter is a literature survey reviewing the previous work done, shock wave theory of blasting, blast monitoring, fragmentation analysis and dragline productivity. The third chapter discusses the operation of the test facilities such as the VODR - 1 and the high speed camera.

The blasting operation at Syferfontein Colliery is critically reviewed in the fourth chapter. Drilling, blast tie-in and explosive loading are the areas of interest. Equally important is fragmentation assessment through the use of KUZ-RAM model and sensitivity analysis of some major blasting parameters such as burden spacing ratio and powder factor which impact heavily on fragmentation.

Certain blasts were monitored through VOD measurement and high speed photography. This is the subject of concern in chapter five which discusses the results of such experimental measurements.

The sixt chapter which is essentially central to the project focuses on dragline performance analysis. Dragline productivity is analyzed in the light of various indices, viz, fill factor (bucket factor), fill time and bank cubic metre per production hour (BCM/Pr.Hr). Furthermore dragline performance is related to blasting in this same chapter. In particular, this chapter considers general discussion, design factors related to

productivity, deductions from VOD measurements and statistical analysis of the results. Operators' performance is also evaluated in this chapter.

Detailed cost analysis is dealt with in chapter seven. The various cost centres, viz. drilling, blasting and dragline cost are the main focus within the confines of this study. Regression analysis has been conducted to establish a linear relation between cost and some productivity indices. Economic implications of the proposed changes in design are analyzed. The text ends with a conclusion and recommendations in chapter eight.

1.4 General Information about the Mine

The Syferfontein Colliery currently mines two coal seams (4# and 5# coal) buried at depths of 30.1m and 62.1m respectively below surface. Two draglines, Marion 1 (2570w) and Marion 2 (8200w), are equipped with 76m^3 and 74m^3 buckets respectively. The overburden, a band of sandstone which is 22.5m thick, is overlain by a topsoil layer 5.5m thick. The overburden blast does not pose much problem compared to that of the inter-burden.

The interburden between #4 and #5 seam is thick, laminated and consists of 9m of shale sandwiched between two sandstone layers each about 9m thick. Drilling in the interburden is done with a 250mm vertical rotary drill at an average depth of 27 - 28m. It is in this area (interburden) that the study is confined. Figure 1.1 shows a typical cross-section of the strata. Drill patterns in the interburden vary from 11 x 10.5 m to 10.2 x 10m for an average hole depth of 27.5m.

The rock is averagely weak with a Young modulus varying between 8 and 45 GPa and uniaxial compressive strengths

varying between 25 and 88 MPa. Table 1.1 below shows the variation of some rock properties with depth. The in situ and after blast specific gravity of the rock in the overburden are 2.52 and 1.87 respectively, and 2.57 and 2.0 respectively for the interburden. The rock is quite massive in the western part and vertically jointed in the northern section of the mine where it is composed predominantly of dolerite.

TABLE 1.1 Variation of Rock Properties with Depth

ROCK PROPERTY	DEPTH				
	15m	35m	45m	80m	109m
Uniaxial Compressive Strength (GF	80	77	81	88	25
Young Modulus (MPa)	10	8	20	45	8
Poisson's Ratio	0.15	0.16	0.19	0.13	0.14

The mine employs four types of loading shovels , namely P & H 2800, H285 DEMAG, H85 DEMAG and H65 DEMAG. Haulage is carried out using four types of trucks, the CAT 789 (177T), CAT 777 (147T), CAT 773 (92T) and D400 (64T). The shovel truck combination is illustrated in Table 1.2 below.

Generally the mine operates an 8 hour shift with 17 shifts per week. There is no production on Sundays. However, the dragline operation is a continuous one with 3 shifts per day, each of 8 hours duration, and 7 days a week.

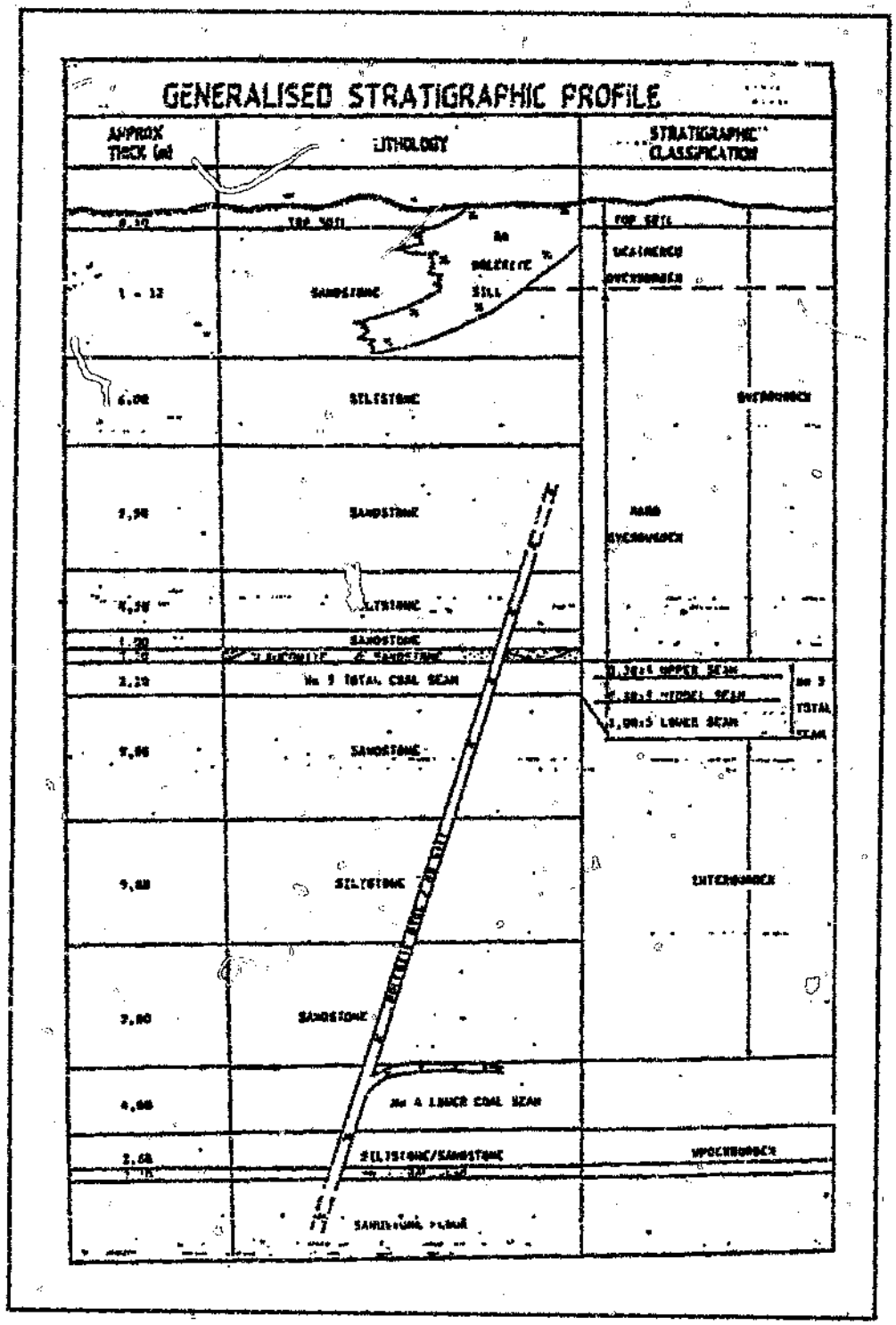


Figure 1.1 Typical cross-section of the rock strata

TABLE 1.2 Truck and Shovel Combinations

Loading Shovel	Bucket Capacity (BCM)	Haulage Truck for shovel	Operating Medium
P & H 2800	31.3	CAT 789, 777	Overburden, interburden and weathered overburden
H285 DEMAG	8.99	CAT 789, 777	Overburden, interburden and #4 seam (coal)
H285 DEMAG	2.38	CAT 777, 773	Top soil, weather overburden, #5 seam and general excavation
H65 DEMAG	2.00	D400	General excavation, top soil, weathered overburden and #5 seam coal

1.5 Sources of Data, Procedures and Treatment of Results

The data for this project was gathered from the Syferfontein Colliery (open cast) on interburden blasts at the west bank, strips 5 and 6. Each strip was divided into 6 blocks. The dragline studied is the Marine 2 which uses a full arc bucket with a capacity of 74m^3. Twelve separate blasts (blocks) have been analyzed. Information was gathered on blast tie-in, loading and dragline production and cost data on drilling, blasting and dragline operations. The work was divided into three major components: fragmentation analysis, dragline

productivity assessment and cost analysis.

On fragmentation, a spreadsheet was developed for a model for fragment size analysis. Blast monitoring was conducted on six separate blasts to verify the cause of poor blast results. The dragline productivity was assessed using the productivity indices of fill factor, fill time and bank cubic metre per production hour (BCM/Pr Hr).

The percentage contribution of the various cost centres, namely drilling, blasting and dragline cost, to the total cost was the main focus of the cost analysis. Regression analysis was conducted to establish a relation between dragline cost, total cost and productivity.

Each of the twelve blasts are analyzed on the basis of fragmentation, dragline productivity and cost. The best blast giving maximum productivity and minimum cost per BCM is selected. The reasons for good blast results as well as poor blast results are identified for future corrections. Sensitivity analysis was performed on the best blast on key blast parameters that influence fragmentation. Eventually the existing practice (design) was critically reviewed and some changes suggested which can help improve fragmentation and dragline productivity and minimize production cost.

CHAPTER 2

FORMULATION OF THE PROBLEM

2.1 Review of the previous work

Blasting remains, in most cases, the only means by which rock is prepared for extraction. Often when poor blast results occur the tendency is to increase drilling and explosive use because the real cause of the problem is not known. This chapter gives the theoretical overview of blasting. Also in focus are explosive assessment, fragmentation analysis, dragline productivity evaluation, blast monitoring and cost analysis from theoretical perspective.

2.1.1 Shock wave theory of blasting

Under ideal conditions, i.e. very large hole diameter and perfect confinement, Chapman and Jouguet postulated and it was further advanced by Zeldov, Von Newman and Doring (Cook, 1974) that explosives undergo ideal detonation. In this case, a plane shock wave travels through the explosives which heat up and causes a chemical reaction that supports the shock wave which travels with a speed termed the velocity of detonation (VOD). The detonation pressure (P_d) in the primary reaction zone is given by the expression

$$P_d = 0.25BV^2 \qquad (1)$$

where B = explosive bulk density (kg/m^3) and V = velocity of detonation.
The pressure of the detonation products in the explosion state, i.e. explosion pressure (P_e) is about half of the detonation pressure and is given by

$$P_e = 0.13BV^2 \qquad (2)$$

Upon liberation, the chemical energy of the explosive is converted to various forms of energy, such as heat, seismic energy (fragmentation), kinetic energy of spoil (throw), concussion and noise (air blast).

The model proposed by Lownds (1984) and Kirby and Lieper (1985) offers the framework to conceptualise the work done by the explosive during a blast. They consider that following detonation the blasthole expands radially to a quasi-static equilibrium state where the explosion pressure is matched by the stress in rock surrounding the hole. The energy delivered up to this point is the shock energy and appears mainly as shock waves crushing near the hole, and as stored elastic strain energy in the rock surrounding the hole. Subsequent expansion of the gases into the fracture network; until they escape to the atmosphere is termed the heave energy. Further expansions, after gases have vented represents wasted energy which appears as airblast and residual heat.

2.1.2 The Mechanism of Rock Breakage

The mechanism of rock breakage as conceptualised by the the fundamental idea of the shock-wave theory is this (Hino, 1959):

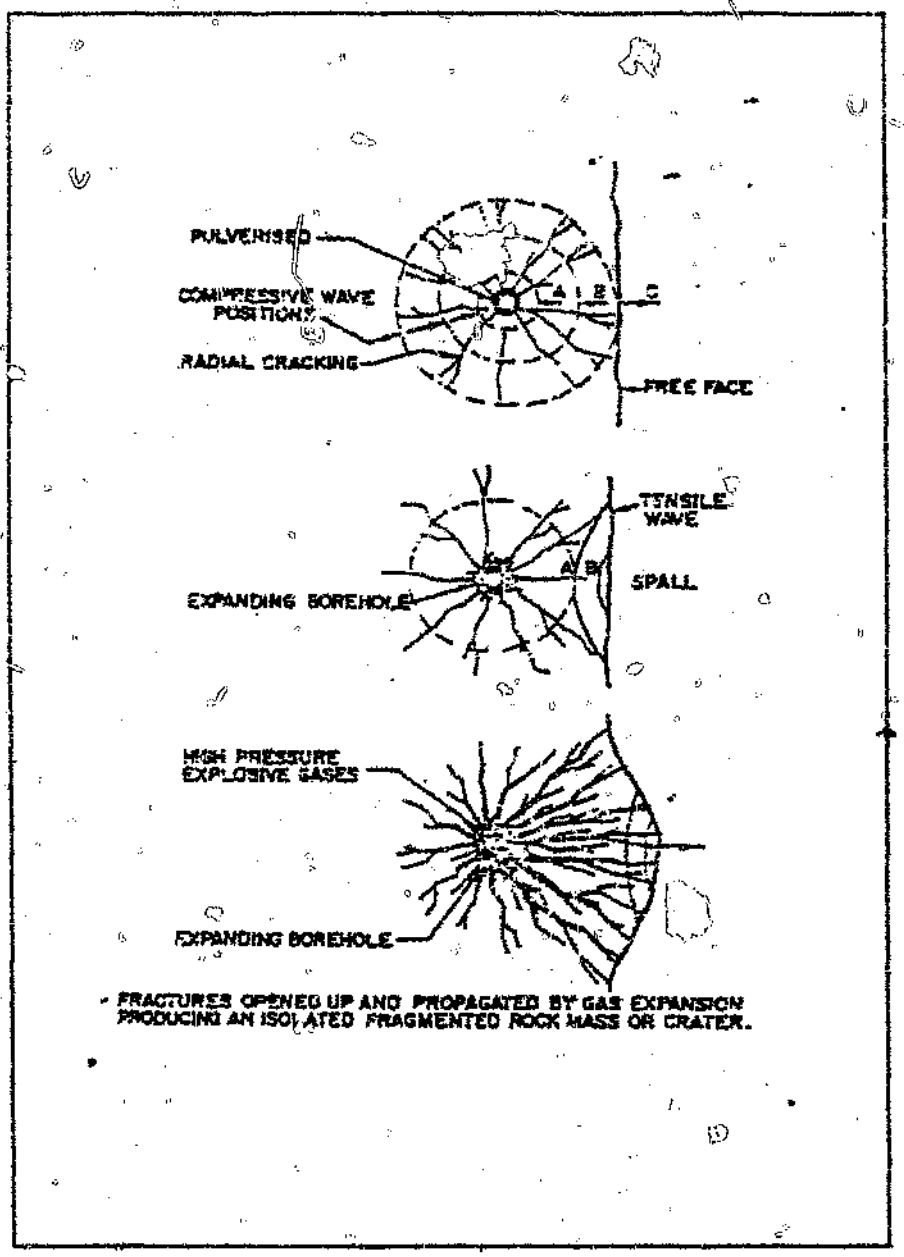
1. Detonation of explosives produces a "crushed zone" around the hole insofar as the intensity of the shock wave produced by an explosive charge is greater than the compressive charge

of the rock. As the compressive strength of the rock is rather high this range of crushed zone is limited to the vicinity of the charge, while a short wave with high peak pressure but of long duration propagates outwards as a compressive wave without producing any breakage of rock (Figure 2.1).

2. Beyond the crushed zone, there can be no compressive failure due to shock wave; however the shock wave (compressive wave) is reflected as a tension wave at the free face because at the free face there can be no force. As the tensile strength of the rock is much smaller than the compressive strength of the rock a fracture occurs at the point where the effective tension, that is, the difference between the intensity of the tension wave and the remaining compression, exceeds the tensile strength of rock.

3. The range of breaking extends from the free face towards the centre of the charge. The distance between this point of the first main fracture and the first free face is defined as the "thickness of the first slab" (Hino, 1959). This dimension defines the size of the fragments. If there remains a compressive wave after the detachment of the first slab which moves outward with a momentum entrapped within its thickness from the original shock wave, then similar processes are repeated at the newly produced free face until the intensity of the remaining compressive wave is reduced to below the tensile strength of the rock.

4. Only a part of the total energy of an explosive goes into shock wave and the gases from detonation at high pressure expand doing work against the resistant forces of the rock



(After Chiappetta, 1983)

Fig 2.1 Sequence of events occurring in a horizontal section of a rock mass surrounding a borehole in which a charge is fired.

and against the inertia of a big mass of rock previously detached by the shock wave of the ground.

2.1.3 Explosive assessment

In assessing explosives one must bear in mind that the total energy contained in an explosive, i.e. raw calculated energy, pressure and detonation velocities are not good indicators of performance. The term "ideal" as applied to explosives refers to the quality of detonation and not necessarily the rockbreaking results. Explosives must detonate at the characteristic VOD in order to perform useful work. Nevertheless oversize boulders and difficult digging conditions after a blast can not usually be ascribed with confidence to poor explosive performance unless measurements are made. Possible problems experienced in a blast may include

- full column deflagration
- cable severance prior to detonation
- Explosive failure to initiate
- Primer failure to initiate
- Poor in-hole delay and surface delay performance

Explosives may have low or high VOD depending upon the particle sizes or chemical formulation of the explosive. The large particle sizes or constituents used in dynamite leads to majority of its energy being released in the secondary reaction zone which takes place behind the primary reaction front. Dynamite therefore has low VODs (1800 - 2500 m/s). On the other hand emulsion contains mainly finely divided constituents. The greater surface area of the reactants leads to rapid combustion with most of its energy being released in the primary reaction zone as the detonation front moves through the explosive

column and less energy being released in the secondary reaction zone behind the detonation front. Hence for an Emulsion, stable detonation is only achieved at a much higher VOD of about 5000 to 6000 m/s. As a consequence, emulsions with a high VOD usually produce finer fragmentation and greater backbreak due to greater shock. Dynamite and ANFO for obvious reasons undergo slow reaction because of their larger grain size producing less shock energy and greater heave energy (Brinkmann, 1990).

2.1.4 Fragmentation analysis

The problem areas in any blasting model which is specifically, rather than comparatively predictive are
- definition of the rock properties
- selection of the appropriate explosive performance index
- determination of actual blasting fragmentation

The KUZ-RAM Model (Cunningham, 1983) is an empirical fragmentation model used for estimating fragmentation, throw, damage and advance. It is based on Rosin-Rammler curve and Kutnetsov equation. C.M Lownds (1983) also did an extensive work on the fragmentation model based on crack growth theory. The KUZ-RAM Model shows the relationship between blasting parameters and mean fragment size. The relevant equations are:

(a) Rosin-Rammler equation

$$R = \exp. - (X / X_c)^n$$

where

R = proportion of material retained on screen X cm (mass fraction of fragment larger than X)

X = screen size (fragment size), cm

X_c = characteristic size or average fragment

size (50 per cent mass passing), derived from Kutnetsov.

n = uniformity index, 0.5 - 2.0.

(b) Kutnetsov equation

$$X_c = A\ (\ K\ )^{-0.8}\ Q^{0.144}\ \ (\ E/E_{TNT}\ )^{-0.633}$$

where

X_c = mean size, cm

A = Rock hardness index, 1 - 22.

K = Powder factor, kg/m^3

Q = kg explosive

E = Relative weight strength of explosive

E_{TNT} = Relative weight strength of TNT

The KUZ-RAM model has been found to give a good correlation where blasting results have been assessed (Cunningham, 1991). Dick *et al* 1982, have measured the fragmentation from blasting in limestone using small diameter blasthole (22mm). Their results correlated very well with the KUZ-RAM model and agreement was obtained over the whole size range. For large scale blasting such as that conducted by Cunningham (1983) using the photoplanimetric technique for fragmentation assessment, the KUZ-RAM gives extremely good correlation in the coarser section of the curve but indicates rather more fines than are given by the photographic analysis. Used as a model predictive tool, the KUZ-RAM model is good for the application of deriving cost-effective drilling patterns within a blasting property. The limitation however is the lack of prediction on overbreak, heave, toe pulling or other important blasting considerations.

Fragmentation of rock according to Brinkmann et al (1990) is affected principally by four categories of factors: rock characteristics (extent of geological and mining

induced structural discontinuities), explosive characteristics, geological relationship between explosive and burden rock (factors that determine the powder factor) and dynamic interactions which occur during millisecond delay firing. They also showed that fragment size reduces with increasing explosive consumption and that the sizes become more uniform toward smaller sizes as specific energy increases.

For a given material type and explosive charge per metre of hole, there is a maximum size of burden that can be used and produce a full crater. To produce a satisfactory digging condition the rock mass should be isolated and broken back to the charge as overburdening may result in boulders. Craters produced by one hole should also overlap those produced by the next (Crosby, 1981). Smith and Ash (1977) in their investigation into how blasthole burden, spacing and hole length affect rock breakage showed that

- coarser fragmentation resulted from simultaneous firing of blastholes compared to that produced when holes are delayed.
- best overall breakage results were experienced when boreholes were delayed using a spacing to burden ratio of one.
- as the spacing to burden ratio was increased for a given length to burden condition there was an associated decrease in the smaller sizing fraction.

2.1.5 Dragline Productivity

Dragline productivity or performance, according to Crosby (1981) is assessed through its digging time, swell factor, fill factor and BCM per production hour.

* The digging time is the aspect of the cycle

time which is mainly related to fragmentation. The digging time represents 25% of a typical cycle time.

* The swell factor is the ratio of the material weight per loose cubic meter to material weight per bank cubic meter. Rocks like sandstone, shale and limestone usually have swell factor values between 0.55 and 0.8 in the bucket depending on the size distribution of the fines and coarse materials. Either extreme, all fines or all coarse, does not give the optimum swell factor. In fact, the optimum lies between the extremes depending on size and shape. Improvement in the swell factor should result in:
  - increase of capacity in BCM per work cycle
  - reduction of the height of the spoil piles which should reduce rehandling where applicable.

* Fill factor is the ratio of loose cubic meter to the bucket rating in cubic meters. It usually varies from 0.4 to 1.1 and depends on the ease with which material flow into the bucket.

Ideally one expects dragline productivity to be increased through improved fragmentation with a concomitant reduction in the overall cost. Crosby et al (1981) advocates that dragline productivity can be improved through

- improving fragmentation based on a relatively low drilling unit cost by improving explosive distribution. This is achieved by reducing the drilling yield (BCM/m). A pragmatic approach

is to reduce the drilling pattern with a scaled reduction in the scaled diameter such that the powder factor remains constant and

- increasing explosive energy input per BCM.

The Drury and Westmaas (1975) study confirmed Mackenzie's argument for reducing total cost by increasing the blasting cost. The result was better fragmentation which increased shovel loading efficiency, reduced maintenance and repair cost and increased crusher output. Mackenzie achieved a superior "degree of fragmentation" primarily by using more energetic or higher strength explosives.

2.1.6 Blast monitoring

Velocity of detonation of explosives (VOD) is an important parameter in blast monitoring. Brinkmann et al, 1990 have outlined a number of reasons for measuring the VOD of explosives. They assert that

- The explosive pressure (the peak pressure applied to the borehole walls) in ideal detonation bears a square relation to VOD (Refer to equation 2, section 2.1.1). As the pressure decreases with the square of VOD, a decreased pressure results in significant drop-off in VOD and hence peak pressure applied to the borehole walls. Rock breakage is then jeopardised.
- The detonation stability decreases at VODs below characteristic values. At characteristic VOD, detonation is at its most stable.
- The measurement provide useful information on how explosive performance is affected by such factors as detonation cord desensitisation, water and sludge, booster size, hole depth and hole diameter. VOD increases with hole diameter. Below the critical diameter, detonation will fail to occur.
- Information is provided on initiation system

performance. Actual detonation times are obtained thus allowing the assessment of both in-hole and surface delays.

- Ground displacement and its effect on cut-offs can also be obtained.

High speed filming has been employed by Ladds et al (1991) in their investigation into the blasting problem at the ATCOM open cast to check whether misfires detected by the in-hole VOD measurements were caused by surface failures of initiation or the in-hole initiation system. The results revealed failures in the delay element and out-of-sequence firing.

Rock assessment is a useful tool if meaningful fragmentation analysis of a blast is to be conducted. The rock conditions to be considered are density, mechanical strength (uniaxial compressive and tensile strength), elastic properties (Young Modulus) and rock structures including jointing. Explosives can and do fragment rock across joints, but the extent of fracturing depends upon the rock strength, blasting geometry, explosive pressure and joint quality.

2.2 Specific aims of the project

In the light of the above, the experimental project is intended to achieve, inter alia, the following objectives:

1. To assess dragline diggability as related to blasting performance.

2. To quantitatively assess the explosive requirement (energy input) for efficient blasting and hence improved dragline productivity.

3. To assess the productivity of the dragline under different blast conditions such as the use of different decking materials and explosives (emulsion) in different formulations.

CHAPTER 3

TEST FACILITY

3.1 VODR-1

VODR-1 is a continuous velocity of detonation (VOD) measuring system which works on a time reflectometry principle with a recording capability (Vather, 1990). Electrical pulses are transmitted down coaxial cables and reflected at discontinuities in the cable. The time interval between the transmission of each pulse and the reception of its reflection are measured. By repeating the single pulse measurement a time history of the cable length is created. This data is used to generate processed data. Some of the cable types in normal use are FSJ1-50A coaxial cable and RG-58 coaxial cable with a minimum cable length of 230m.

Sample intervals as short as 5μs can be selected giving a resolution of 30mm at 6000m/s. A series of sequential holes can be observed from one cable by looping the cable from one hole to the other. It is possible to monitor a hole of length 70m with this system. The VODR-1 provides an on-site graphics display and hard copy of recorded data both as numerical listings and graphics plots. Figure 3.1 shows a simplified VODR data curve and illustration of how the curves are interpreted.

VOD measurements are an indispensable tool in blast monitoring in that they help to assess:

- stemming performance i.e. selection of

appropriate stemming material

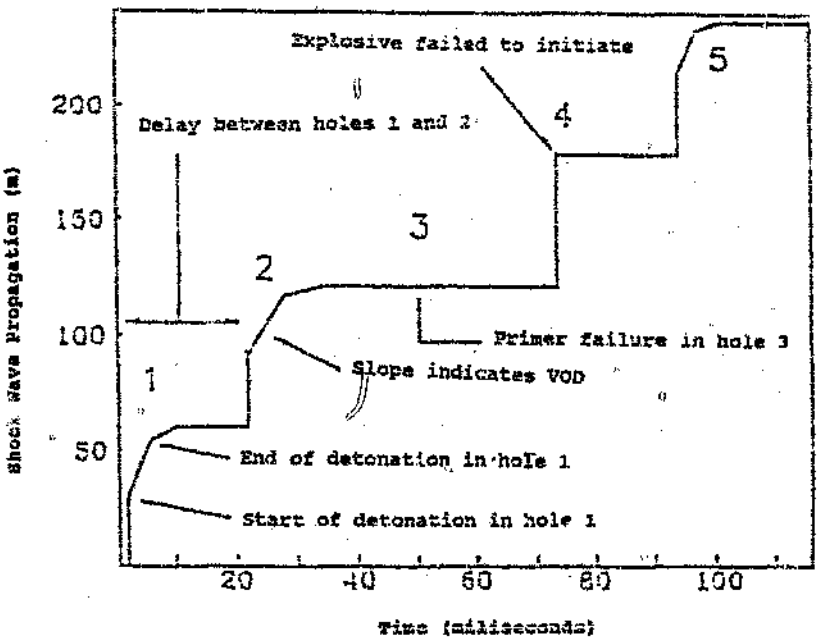


Fig 3.1 Typical VOD output with curve interpretation (After Ladds et al, 1991)

- detonation performance of explosives. A VOD less than the characteristic values implies that some of the ingredients are not reacting at all. This may be caused by a host of factors including the presence of borehole water, poor mixing or loading, inadequate priming, desensitisation due to shock from earlier fired holes (time scatter effect) and insufficient booster size,
- ground displacement and its effect on cut-off

can also be determined

- information on initiation system performance. Actual detonation times are obtained thus allowing an assessment of both surface and in-hole delays. Compatibility of the initiation system and/or explosive with the blasting environment can also be verified.

3.2 High Speed Camera

The LOCAM II high speed camera is a 16mm rotating prism type camera usually positioned at a safe vantage point and using daylight as the sole source of illumination. Filming is carried out at a speed of 500 frames per second and this provides a detailed record of the performance of the surface initiation system. The camera is usually positioned at the back of each blast between 100 and 400m from the blast. High speed photography assists in

- evaluating blast design alternatives
- identifying the causes of unacceptable blast results
- checking whether the surface initiation system or in-hole initiation system is responsible for misfires detected by in-hole VOD measurements
- checking out-of-sequence firing.

3.3 Geophones

A geophone is a sensor which feeds information to a recorder. It consists of a digital recorder in the form of a multi-channel high sample rate unit used for close-in seismic measurements. The high sampling rate is necessary to obtain the required resolution of the near field frequency activity. Geophones provide useful

information on the overall blast performance and timing design. Brinkman et al (1990) have recommended close-in seismic monitoring as a means of obtaining the following information:

- overall blast timing can be determined as initiation in each row or hole is detected by the nearby geophones
- areas of poor blast performance are clearly seen as the amplitude dips in the sequence of seismic wavelets. Where three dimensional monitoring is applied, the dips can be correlated with in-hole measurements thus allowing assessment of other areas in the blast where poor performance is noted but no in-hole data is available.
- by comparing sequential wavelet amplitudes and durations from close-in seismic traces, it is possible to detect rows or holes where overburdening or poor burden relief has occurred by the existence of abnormally high amplitudes.

3.4 The Dragline

The dragline is a stripping machine used mostly for stripping overburden and shallow interburden. There is a growing tendency towards the use of the dragline as opposed to other stripping machines such as shovels and bucket wheel excavators because of their high production rates and versatility. Other reasons for the preferred use of the dragline include:

- flexibility in operation or ease of movement
- large digging depth capability
- ability to handle or stack material having poor stability
- complete safety from spoil pile slides or pit

flooding during normal operation
- high percentage of coal recovery or less coal damage
- low maintenance costs
- ability to dig deeper box cuts and rehandle partings well
- ability to move in any direction

Figure 3.2 is a schematic showing the major components of dragline. Figure 3.3 is a typical dimensional diagram of a walking dragline, the major dimensions being designated by letter identifications. Table 3.1 identifies the terminology for the various dimensions designated by the letter code in Figure 3.2. Of all these dimensions the two of most importance are B, operating radius and G, dumping height. These two dimensions describe the physical stacking capability of a particular machine. At Syferfontein Colliery, the dragline is used for both overburden and interburden stripping but this study is based on the use of the dragline in the interburden only as mentioned earlier. The various digging methods are discussed below:

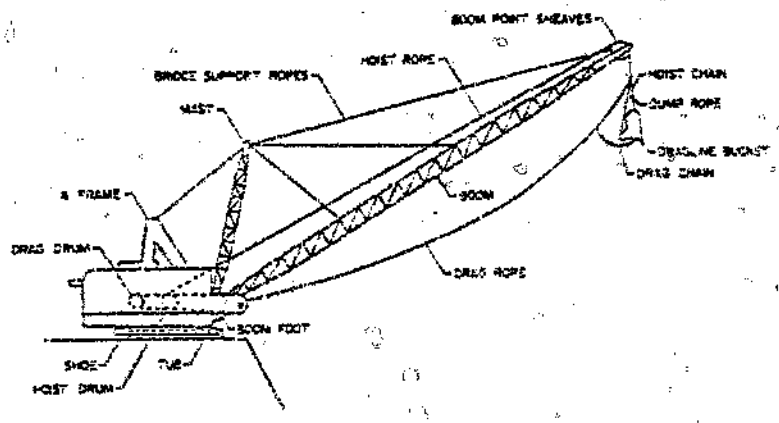


Fig 3.2 Schematic of Dragline showing major components

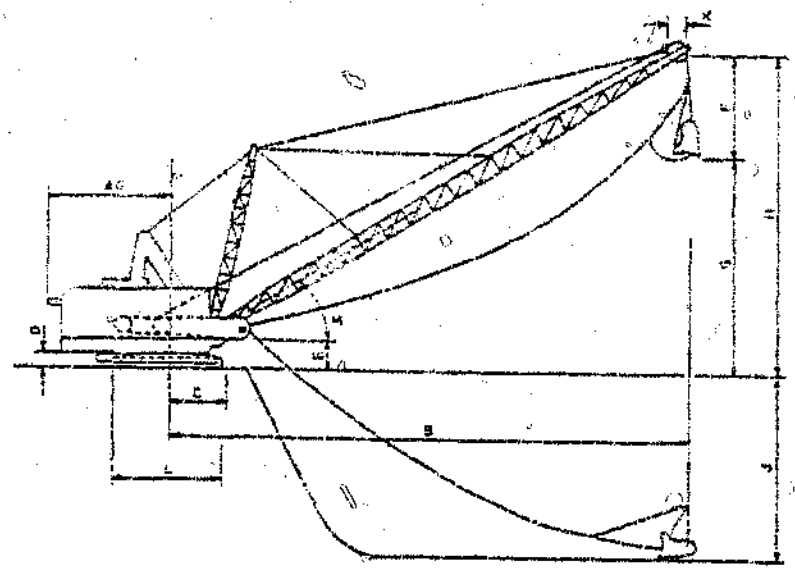


Figure 3.3 Dimensional diagram of a walking dragline

LETTER CODE	DIMENSION TERMINOLOGY
A	Clearance Radius
B	Operating Radius
C	Boom Foot Radius
D	Clearance Height
E	Boom Foot Height
F	Dumping Clearance
G	Dumping Height
H	Boom Point Height
J	Digging Depth
K	Point Sheave Pitch Diameter
L	Tub Diameter
M	Boom Angle

Table 3.1 Dragline Dimension Terminology as illustrated in Figure 3.3

3.4.1 Dragline digging modes

This section discusses the various digging modes of the dragline. A dragline stripping method is dictated by such factors as bank height, the nature of the overburden, highwall angle and spoil pile angle of repose as well as stability. The dragline dimensions also greatly influence the stripping method. Although there are several stripping methods with many diversifications, this section emphasises on the main ones:

- Box pits
- Simple side casting
- Advance Benching
- Extended Benching
- Pull back method

Box Pits

The first cut used to begin a new mine is quite often a box cut. This is necessary to produce an initial highwall. This pit gets its name from the long box like shape of the completed cut. There are two distinct methods of making the box pit without rehandling material, these being (a) End cut and (b) Side cut.

For very thick overburdens relative to dragline size, methods may be necessary which entail certain quantities of rehandling. These methods involve either:

(a) Spoiling on both sides of the box cut, or
(b) First producing a borrow pit on the spoil side of the box cut.

(a) End Cut

The end cut box pit is produced with the dragline positioned at the end of the pit. Figure 3.4 shows a

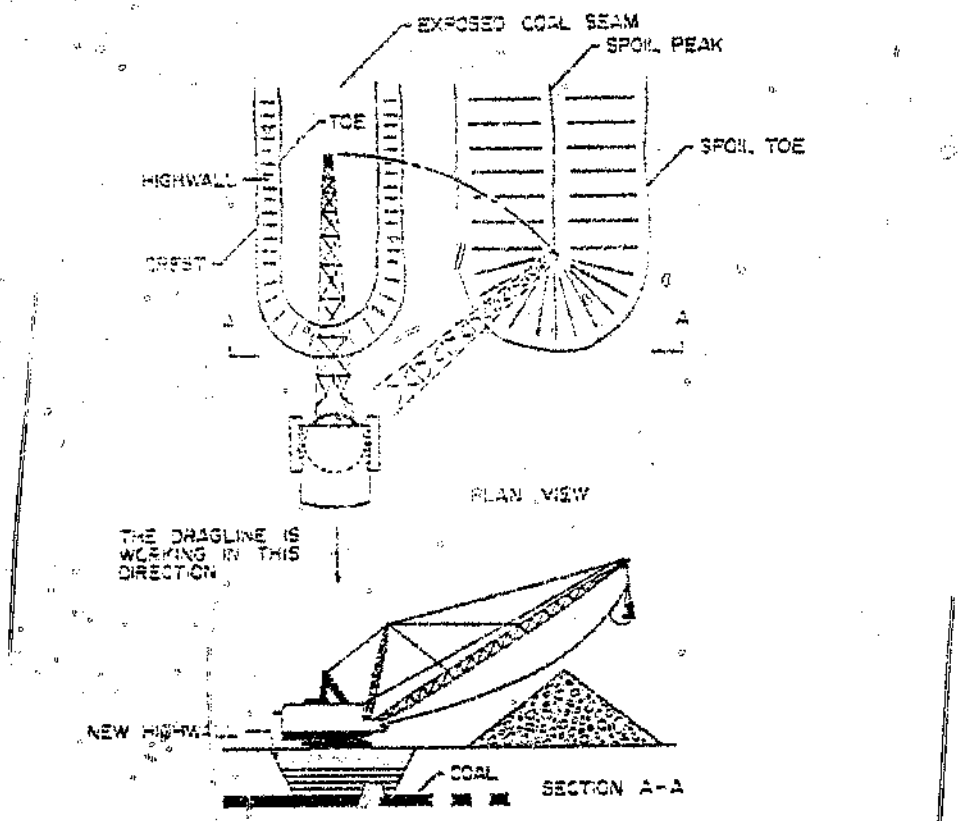


Figure 3.4 Plan and cross section view of Box Pit, end cut method.

typical end-of-box pit being produced. This method is used in overburden covers that are relatively thin compared to the dragline dimensions. Thus it is possible for the machine to spoil all the material to the side away from the coal deposit, or to what becomes the opposite side of the cut from the new highwall. Figure 3.5 shows that all the material can be spoiled in this case using a maximum dragline swing angle of 90 degrees. The end cut method has a very favourable dragline swing angle (maximum 90 degrees) especially when compared with that of the side cut method. However, the end cut method is restricted to relatively shallow overburden thicknesses because of the limit on the dragline dumping radius. As the cut increases in depth the waste pile increases in volume until it can no longer be spoiled completely clear of the box pit.

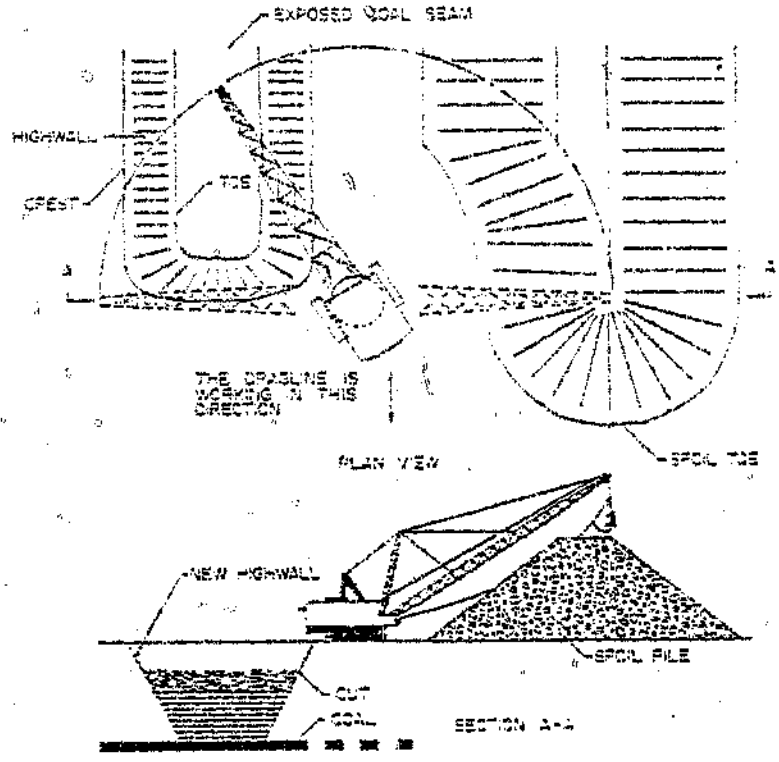


Figure 3.5 Plan and cross sectional view of Box Pit, rehandle End cut method.

(b) Side Cut

The side cut box pit is produced with the dragline positioned at the side of the pit. This method is used when the overburden is of such a depth that it is impossible for the dragline to use the end cut position. The side cut position allows the spoil to be dumped at a greater distance from the pit making it possible to widen the spoil pile at the maximum dumping height, as required.

Appendix A1 shows that the swing angle can be increased to the theoretical maximum of 180 degrees for certain parts of the cut. In practice the highwall side of the pit would be allowed to lag behind the near side reducing the maximum swing angle to approximately 130 degrees. This is still considerably larger than for the end cut method but for thicker overburdens and this given machine size, it is the only way the box pit can be produced without rehandle.

Simple Side Casting

Simple side casting is the most common stripping method and is the most straightforward. This method, or variations of it, can be seen as part of most other methods even in the most complex spoiling sequences. Basically the dragline sits on the overburden surface and digs down to the coal surface spoiling into a pile at one side. Appendix A2 illustrates a dragline using the simple side casting technique.

There are two basic cuts that make up the simple side casting method. The first is the key cut. This cut, illustrated in Appendix A3, is designed to excavate a wedge of material, termed the key, at the inside limit of the complete cut for the particular pit width. To dig this material, the dragline is positioned directly in line with the key at such a point that the spoil can be dumped directly onto the pit floor of the previous cut. The key cut is the only way to establish a highwall slope.

Advance Benching

It is not unusual for the simple side casting method to

be modified into the advance benching method. This is done by forming a bench working level for the dragline at an elevation somewhat lower than the original ground surface. Such an operation is illustrated in appendix A4.

This method is particularly useful in areas where the surface conditions are soft and thus unsuitable for dragline support. A level is therefore selected in the bank so that the bearing material is of the required strength. Another use is in rolling or hilly terrain where there are frequent ridges or valleys. The advance benching technique allows a stabilization of the dragline working level by cutting through ridges and reducing fill requirements in depressions to a minimum.

Extended Benching

Extended benching is the first method so far considered that is specifically designed to include rehandling. This rehandling is necessary to allow the dragline to handle overburden covers that are beyond its capability using the previous methods. This type of situation can develop for a number of reasons. Usually the dragline has been purchased sometime previously for a shallower overburden condition and has been moved to a new location with increased cover. Another situation may be isolated hilly areas of a property for which it would have been uneconomical to design a dragline specifically. Thus the dragline is designed for the average cover and is used in a rehandle situation in some isolated areas to prevent the loss of coal that could result if these islands were not taken. Appendix A5 illustrates a typical extended bench operation.

This bench extension allows the dragline to move out towards the spoil thus making it possible to reach the

required spoil peak. The material used to make the extension must be excavated later, this being the designed rehandle. The material for the extension is dug from two main locations, the key or the key and advanced bench.

Pull Back Method

The pull back method is an alternative to extended benching when the dragline reach is insufficient to handle the required overburden cover without rehandling. Its use results from similar pit conditions and for similar reasons as were outlined in the section on extended benching. A typical pull back operation is illustrated in Appendix A6.

The pull back method can be adapted using a single dragline or using two draglines. In the latter case both machines are being assigned a specific part of the operation. The method entails first either side casting the initial cut from the original ground surface or from an advance bench. In the pull back method the dragline is unable to spoil all the overburden clear of the coal. As a result this material fills over the coal seam at the toe of the spoil pile and is the material that must be later rehandled to completely uncover the coal.

CHAPTER 4

BLASTING OPERATIONS AT SYFERFONTEIN COLLIERY

Rock breaking by drilling and blasting is the first phase of the production in most mining operations. Optimisation of this process is very important because the fragmentation obtained in this operation affects the costs of the whole group and inter-related mining operations including loading. In a strict sense, optimisation of rock breaking by drilling and blasting would mean minimisation of the cost of these two individual operations. However a minimum cost for breaking rock may not be in the best interest of the overall mining system. The blasting related problems usually encountered at Syferfontein Colliery include production of excessive fines and/or boulders, a table top effect resulting from misfires, hard toes and undiggable areas and high vibrational levels. A high proportion of fines indicates excessive use of explosive consumption which in addition to escalating cost, increases the violence of the blast unnecessarily (Brinkmann, 1990). The Table top effect (unfragmented sections) caused by misfires may be the result of under utilisation of explosives (low powder factor) and/or poor blast design. This situation has necessitated secondary breakage in some cases on the mine. Either extreme, predominantly fines or boulders, is undesirable as it leads to low dragline productivity in terms of BCM per hour, low bucket or fill factor and longer digging period. The aforementioned problems justify a critical

analysis of the existing practice on the mine from design perspective as well as fragmentation analysis. This chapter therefore briefly reviews blast design as it occurs on the mine and fragmentation assessment using a model. Figure 4.1 shows the summary of the existing design parameters.

4.1 Review of the existing practice (design) on the mine

In the sections that follow the existing practice on the mine is reviewed from a design perspective using empirical relations where appropriate. The blast design in question is categorised under drilling, blast tie-in and explosive loading.

4.1.1 Drilling

Drilling in the interburden between #4 and #5 seam where this study is confined, is done using a 250mm vertical rotary drill. Hole lengths vary between 26.3 and 28m with an average of 27.5m (Table 4.1). However, the length of the hole is determined by the stratigraphy of the number 4-seam. The number of holes per blast may vary between 62 and 166. Crest burden varies from 5 to 6m and a toe burden of 8.5m is not uncommon. As can be seen from Table 4.1, various combinations of spacing burden ratio were used in different blasts. These include 10 x 10.2m, 11 x 10.2m, 11 x 10.5m and 10.5 x 10.5m. As it is the most important design parameter which greatly impacts on fragmentation and which can be easily varied in the field for optimum fragmentation, the discussion that follows focuses mainly on the spacing burden ratio.

Burden

The selection of the burden is the most vital decision in

PARAMETERS	EXISTING PRACTICE
1. Drill Pattern (S x B)	11X10.5m, 10.5x10.5m, 11 x 10.2m, 10 x 10.2
2. Hole Diameter (d)	250m
3. Standard Deviation (w)	0.3
4. Hole Length (H)	27 - 28m
5. Charge Length (L)	18 - 22m
Rock Properties	
6. Young Modulus (Y)	15
7. Rock density	2.57
8. Poison ratio	0.13 - 0.44
9. Uniaxial Compressive Strength	77 - 88MPa
10. Rock Mass Description (RMD)	50
11. Density influence (RDI)	14.25
12. Hardness factor	5
13. Rock Hardness index	4.16
Explosive Properties	
14. Powder factor (Kg/m3)	0.39 - 0.49
15. Kg Explosive / hole	1116 - 2004
16. RWS of explosve	83.3 @ 1.31 RD 82.5 @ 1.15RD
17. RWS of TNT	131
18. Bulk density	1.398 @ 1.31 RD 1,323 @ 1.15RD
Other Parameters	
19. Explosive type	E-6000 (RD of 1.31 & 1.15
20. Decking material	3 - 5 Air, Ash &Full Column
21. Number of Holes /blast	62 - 168
22. Delays	
(a) Between rows	42ms, 67ms, 84ms
(b) In-hole	Top = 525, Bottom = 500 Top = 500, Bottom = 500
23. Boosters	(170 - 336) x 454kg
24. Stemming	5m drill Chippings
25. Crest burden	5, 5.75, 6m
26. Toe burden	8.5
27. Presplit	3m
28. Tie-in	V1 configuration
29. Detaonator	1 x 6D

Figure 4.1 Design Parameters: Existing practice at Syferfontein Colliery

Table 4.1 Drilling data

BLAST DATE (1)	BLOCK NUMBER (2)	HOLE DIAMETER (mm) (3)	HOLE LENGTH (m) (4)	CREST BURDEN (m) (5)	TOE BURDEN (m) (6)	NUMBER OF HOLES (7)
23/2/93	3205109	250	27	6	8.5	164
26/2/93	320587	250	28	6	8.5	139
03/3/93	320576	250	27.5	6	8.5	97
10/3/93	320575	250	27.5	6	8.5	114
18/3/93	320543	250	27.5	6	8.5	85
25/3/93	320531	250	27.5	6	8.5	168
30/3/93	320501	250	27	6	8.5	62
01/6/93	3206108	250	27.5	5	8.8	195
08/6/93	320678	250	27	5	8.5	66
11/6/93	320676	250	27.8	5	8.5	114
18/6/93	320665	250	27	5.75	8.5	96
29/6/93	320643	250	26.3	5.75	8.5	166
		250				
AVG		0.00	27.30	5.71	8.53	122
STD		0.00	0.45	0.43	0.09	45

a blast design. The dimensions selected must suit the characteristic of the rock being used. This parameter is chosen so that satisfactory fragmentation and displacement are obtained. Dubnie's (1964) study of more than 100 mining operations shows that irrespective of rock material, explosive material and initiation system, the burden (B) is practically related to hole diameter (d) as follows:

$$B = 0.024d + 0.85 \qquad (3)$$

with correlation co-efficient of 0.7988.

Applying this relation to the Syferfontein operation with

a hole diameter of 250mm,

$$B = 0.024\ (250) + 0.85 = 6.994m \approx 7m$$

Hence a burden of 10.2 to 10.5 as used on the mine appears to be too much. With such a large burden, the unit volume of rock broken increases and fragmentation may not be satisfactory. On the other hand the reduced burden means more holes and an increase in drilling cost per unit volume. A trade-off is required between closer drilling pattern necessitating more explosive input and good fragmentation affording easy dragline diggability.

Hagan (1982), through a series of studies, also recommends burden in the range of 20d-35d with the optimum around 25d. that is

$$B = 25d \qquad (4)$$

where B = Burden in meters and d = hole diameter in millimetres. With a hole diameter of 250mm as used on the mine, this empirical relation gives a burden such that

$$B = 25 \times 250mm = 6.25m.$$

In all certainty this does not deviate very much from the value obtained using Dubnie's approach and hence confirms the argument that the burden of 10.2 to 10.5 used on the mine appears to be too much.

Hagan justifies his argument by proving that excessive burden (greater than the optimum value) might result in one or more of the following:

(a) Inability of the strain-wave to fracture over a greater distance thus preventing efficient utilisation of heave energy. High pressure

gases then jet into a network of short radical cracks created, which is inadequate for efficient breakage. Heave energy is unable to provide adequate displacement and its associated breakage, and this results in a tight muckpile resisting dragline bucket penetration.

(b) Explosive gases can bottle up within the blast hole for a period of time which is longer than the optimum, resulting in ground vibrations. In their effort to escape to the atmosphere these gases are obliged to stream into, wedge open and extend both natural discontinuities and strain-wave induced cracks all the way around the blasthole. Forward-looking fissures and cracks are not preferentially extended. (Forward-looking cracks are extended preferentially in a blast with a well-balanced burden)

(c) With a much larger burden, explosion gases behave as though they are working to only the horizontal face, i.e. the natural ground surface; the shot becomes a crater blast. The degree of fracturing, loosening and loss of cohesion within the strata tends to be symmetrical about the longitudinal axis of the blasthole. This encourages excessive overbreak and face-loose rock, a situation often encountered on the mine.

Spacing

Spacing is an important parameter since it controls mutual stress effects between holes. This dimension is dependent mainly upon the burden, hole depth, relative

primer locations between adjacent charge columns and the delay interval used for adjacent charges. The spacing (S), according to Dubnie (1964), is related to burden as follows:

$$S = 0.9B + 0.91 \qquad (5)$$

with correlation co-efficient of 0.86.

Hence from equation (4) with burden B of 7, the spacing is

$$S = 0.9\ (7) + 0.91 = 7.21m.$$

The spacing used on the mine varies between 10 and 11m. With such large spacing, craters due to one hole may fail to overlap those produced by the next resulting in toes and hence difficulty in digging by the dragline. This situation is not uncommon on the mine and is apparently one of the reasons for fragmentation problems resulting in low dragline digging efficiency. The spacing-burden ratio is discussed again under fragmentation assessment using a computer model.

Furthermore where the spacing is too large, the rock between and just in front of the blastholes in the back row may be poorly broken. Such solid pinnacles or buttresses could cause relatively low digging efficiencies, especially at or near grade level where the unbroken buttresses would be manifested as toes.

The selected spacing, therefore, must be a compromise: large enough to prevent excessive overlap of overbreak zones behind adjacent blastholes, but small enough to a give relatively even distribution of explosive energy in the rock to be broken and removed.

4.1.2 Blast Tie-in

Table 4.2 summarises the items discussed in this section about blast tie-in. These include initiation, delays and boosters. A square drill pattern is used on the mine with tie-in the form of V0 configuration (see figure 4.3).

Delays

A basic requirement of multiple line blasting is that a reflecting plane or free face must be provided in front of each row of holes in order to pre-condition the rock mass of the next row to detach or displace it horizontally. This requires delays in the sequence of detonation of blast holes or groups of blast holes. For the property being investigated a 42 ms, 67 ms and 84 ms delay between rows are in common use. Shocktube is used as a trunkline and there are no delay intervals between holes as such. Crosby et al (1981) have found that the time it takes for the burden to start to move in a bench blast is proportional to the size of the burden. They advocate that for optimum blast displacement a surface delay interval between rows of holes should be 5 - 7 ms per meter of burden.

i.e $D_i = 5 - 7$ ms / meter (6)

For a burden of 10.5 used on the mine, the delay interval requirement will be as follows:

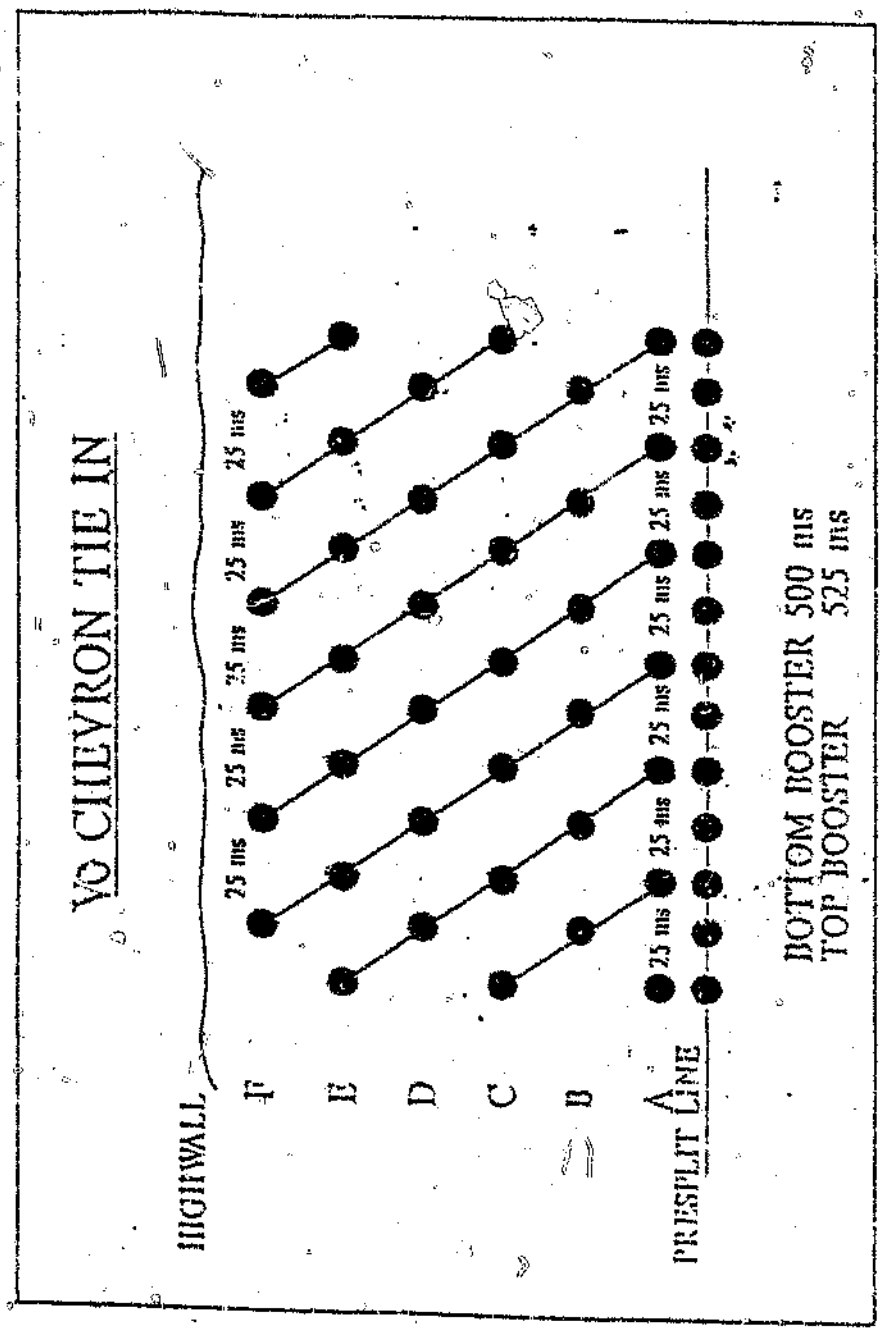
$D_i = 5 \times 10.5\text{ms}$ to $7 \times 10.5\text{ms} = 52.5$ to 73.5 ms

It can be inferred from the above analysis that the 42ms delay interval used on the mine is simply inadequate considering the burden used. With such a short delay, there may be insufficient horizontal displacement of the muckpile, which in turn means muckpile height will be

Table 4.2 Blast tie-in

BLAST DATE (1)	BLOCK NUMBER (2)	S X B (m) (3)	SHOCKTUBE (NONEL) (4)	TRUNKLINE LENGTH (CASES, m) (5)	DELAYS BETWEEN HOLES (ms) (6)	IN HOLE DELAY (ms) (7)	BOOSTER WEIGHT (gm) (8)
23/2/93	3205109	10 X 10.2	N1=164X12m N2=164X30m	4	37 X 42	T = 525* B = 500*	328 X 454g
26/2/93	320587	10 X 10.2	N1=139X10m N2=139X30m	6	29 X 42	T = 525 B = 500	278 X 454g
03/3/93	320576	10 X 10.2	N1=97X10m N2=97X30m	10	23 X 42	T = 525 B = 500	194 X 454G
09/3/93	320575	10 X 10.2	N1=114X10m N2=114X30m	5	21 X 42	T = 525 B = 500	228 X 454g
18/3/93	320543	11 X 10.2	N1=85X10m N2=85X10	4	17 X 42	T = 525 B = 500	170 X 454g
25/3/93	320531	11 X 10.2	N1=168X10m N2=168X30m	5	40 X 42	T = 500 B = 500	336 X 454g
30/3/93	320501	11 X 10.2	N1=62X10m N2=62X30m	3	30 X 42	T = 500 B = 500	124 X 454g
01/6/93	3206108	11 X 10.5	N1=183X10m N2=183X30m	4	45 X 42	T = 500 T = 500	390 X 454
08/6/93	320678	11 X 10.5	N1=65X10m N2=65X30m	5	15 X 67	B = 500 T = 500	130 X 454g
11/6/93	320676	11 X 10.5	N1=114X30m N2=114X12m	4	37 X 42	T = 500 B = 500	228 X 454
18/6/93	320665	11 X 10.5	N1=96X30m N2=96X12m	8	38 X 67	T = 500 B = 500	192 X 454g
29/6/93	320643	10.5X10.5	N1=166X30m N2=166X12m	13	64 X 42	T = 500 B = 500	332 X 454g
08/7/93	320622	10.5X10.5	N1=194X30m N2=194X12m	8	93 X 42	T = 500 B = 500	776 X 454g

T = Top Charge Firing Time B = Bottom Charge Firing Time



(Source: Syferfontein Colliery; BLASTATS, Interburden Blast, June 1993).

Figure 4.2 Blast tie-in: V0 configuration

more, blast can be choked, formation will be tight and the floor will be full of toes. This situation partly contributes to the difficult dragline diggability and hence low productivity experienced in a number of blasts. It should nonetheless be noted that if the delay interval is too long, cut-off of the primacord line or in the extreme case the disruption of the explosive column can result.

Two types of in-hole delays are in common use on the mine: top 525ms, bottom 500ms in some cases and 500ms for both top and bottom charges in other cases. Investigation of in-hole delays is best done through velocity of detonation measurements and this is covered in chapter 5 under blast monitoring.

4.1.3 Explosive Loading

The explosive loading data has been summarised in Table 4.3 for the blast under consideration. Two formulations of emulsion have been used, viz. emulsion E-6000 (relative density 1.31 g/m^3) in some cases and in other cases gassed emulsion of relative density 1.15g/m^3. In effect, Emulsion E-6000 is a heavy ANFO consisting of 60% Matrix or Emulsion and 40% ANFO. Powder factors range from 0.39 to 0.49 kg/m^3 with an average value of 0.45kg/m^3. With such powder factors fragmentation problems should not be anticipated but this is hardly the case.

The high shock energy of an emulsion is also responsible for the crushing of rock within the immediate vicinity of a borehole thus contributing to excessive production of fines. This situation is exacerbated by the lack of horizontal displacement of blasted rock which results in a tight muckpile which offers resistance to bucket penetration and thus culminates in low dragline

Table 4.3 EXPLOSIVE LOADING DATA

BLAST DATE (1)	BLOCK NUMBER (2)	POWDER FACTOR (Kg/m3) (3)	Kg of Explosive per Hole (4)	CHARGE LENGTH (m) (5)	RELATIVE DENSITY (g/m3) (6)	DECKING		STEMMING LENGTH (m) (9)	RELATIVE WEIGHT STRENGTH (10)	BULK DENSITY (g/cm3) (11)
						MATERIAL (7)	LENGTH (8)			
23/2/93	3205109	0.48	1171	18	1.3	Ash	3	5	83.3	1.398
26/2/93	320587	0.44	1273.8	20	1.3	Ash	4	4	83.3	1.398
03/3/93	320576	0.49	1310	19.5	1.3	Ash	4	4	83.3	1.398
09/3/93	320575	0.44	1200.7	17.5	1.3	Ash	5	5	83.3	1.398
18/3/93	320543	0.39	1116.7	17.5	1.3	Ash	5	5	83.3	1.398
25/3/93	320531	0.48	1234.3	17.5	1.3	Air	5	5	83.3	1.398
30/3/93	320501	0.43	1289.5	19	1.3	Air	3	5	83.3	1.398
01/6/93	3206108	0.45	1265.5	17.5	1.3	Air	5	5	83.3	1.398
08/6/93	320678	0.48	1449.4	17	1.3	Air	5	5	83.3	1.398
11/6/93	320676	0.4	1288.6	18.5	1.3	Air	5	5	83.3	1.398
18/6/93	320665	0.49	2004.7	23	1.15	-	-	5	82.5	1.323
29/6/93	320643	0.48	1306.5	21.3	1.15	-	-	5	82.5	1.323
AVG		0.45	1325.89	18.86	1.28		3.67	4.83	83.166667	1.386
STD		0.03	219.38	1.75	0.06		1.80	0.37	0.2981424	0.028

productivity. The selection of the appropriate surface delay interval could eliminate tight muckpiles and as a consequence improve the dragline digging efficiency.

Decking

Deck loading is performed by the mine to counteract the hard banded formations of sandstone between which is sandwiched a soft siltstone layer, and also to achieve better vertical energy distribution for a given powder factor. Figure 4.3 illustrates a typical section of a blast hole with deck

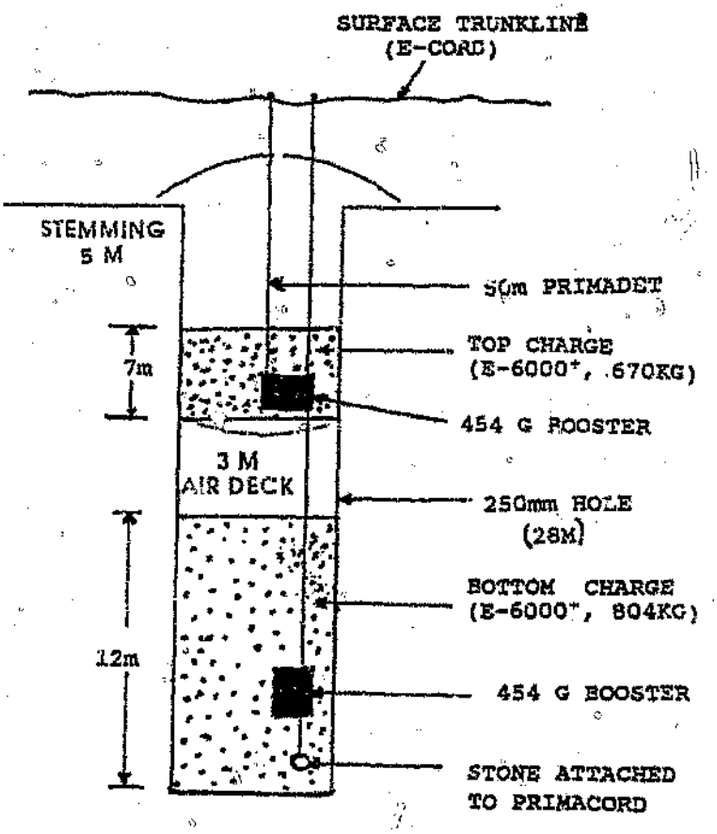


Figure 4.3 Deck loading in a blast hole

loading. The mine employs either ash or air of length between 3m to 5m as deck stemming material between the top and bottom
charges. Full column charging has been a recent practice in an attempt to redress fragmentation problems as well as digging difficulties observed with the dragline operation. The effect of decking on fragmentation and dragline performance is addressed in the subsequent chapters. Where deck charges are employed separate primers are used for each deck. As mentioned earlier, charges are either delayed in the hole at 25ms delay intervals, or not delayed in some cases. In the former case, initiation is from bottom to top to avoid the upper part of the hole being weakened before the lower explosive column generates its full velocity. Upper fracturing creates an escape route for the gases which should remain confined to accomplish the work of breaking and fragmenting. Furthermore bottom priming results in better breakage in the area which is the zone of rock most difficult to break. A 454g booster is inserted in both bottom and top deck to bring the explosive to detonation quickly.

4.2 Fragmentation Analysis

Fragmentation analysis is one of the major considerations when assessing blast results. Owing to the nature of blasting and the variation of geological factors it is difficult to predict blasting results either precisely or consistently. Any attempt to predict blasting is handicapped by two obstacles, viz:

- geological conditions which can not easily be mapped for every blast,
- implementation of blasting plans is subject to the vagaries of day-to-day problems such as lost holes, short holes, inconsistent water levels, drilling inaccuracy and cut-offs in

the explosive column.

Nevertheless there are reasonable indicators for assessing blasting effectiveness. These include

- fragmentation, what percentage oversize
- hard toe and undiggable areas necessitating secondary breakage
- multiple profile
- loading rates as well as wear rates on bucket teeth
- others such as backthrow, flyrock, vibration and air blast.

While all of these factors may be termed 'satisfactory' or 'good' the question still remains, "Are we blasting too heavy or too tight". The relative importance of the above-mentioned factors is debatable. A factor such as the loading rate is quite subjective as it is inextricably linked to the indefinable differences in the operators' abilities (Hino, 1956). This subject (operators' performance) is covered in Chapter 6 under dragline performance analysis. A visual observation of muckpile as a means of assessing blast effectiveness may result in erroneous conclusions. This is evidenced by the earlier studies by Brinkman (1990) where the muckpile profile looked very good but a lot of boulders were observed and difficulty was encountered when digging commenced. This method, although employed by many blasters, inevitably underestimates fines and cannot also determine the buried dimensions of protruding boulders. In the next section emphasis is placed on fragmentation as a means of assessing blasting performance through the use of a model which has been tested and proved as a satisfactory predictive tool of blast results.

4.2.1 Fragmentation Assessment with the KUZ-RAM Model

One of the most important objectives of blasting is to have reasonably good fragmentation for a muckpile easily amenable to shovel loading. Much effort has recently gone into accurately predicting fragmentation, as the degree of fragmentation of rock affects the performance of loading, crushing and the overall economics of the mining operation (Scott, 1992). It is acknowledged that screening is the only reliable, accurate and unbiased method of evaluating fragmentation quantitatively. Nevertheless given the large volume of rock broken per single blast ahead of dragline digging, screening or sieving can not be conducted as a means of assessing fragmentation. The use of a reliable fragmentation model for predicting blasting results is thus justified. Fig 4.4 is a field model illustrating blasting inputs and outputs. The KUZ-RAM model (Cunningham, 1983) is an empirical fragmentation model used for estimating fragmentation. It makes use of the Rosin-Rammler (1933) curve and Kuznetsov (1973) equation. C M Lownds did an extensive work on the fragmentation model based on crack growth theory. In particular it shows the relationship between blasting parameters and mean fragment size.

Fragmentation of rock, according to Brinkmann et al (1987), is affected principally by four categories of factors, viz, rock characteristics (extent of geological and mining induced structural discontinuities), explosive characteristics, geological relationship between explosive and burden rock (factors that determine the powder factor) and dynamic interactions which occur during millisecond delay firing.

In the KUZ-RAM model, rock has been defined by a "Rock factor", 'A', which was recommended to be between 7 and 12, depending on the hardness and jointing of the rock.

The explosive performance is defined by the density and relative weight strength, and substantiation of the actual fragmentation achieved was based on the available published results.

The relevant equations are

(a) Rosin-Rammler equation

$$R = \exp. -(x/x_c)^n \qquad (7)$$

where

R = proportion of material retained on screen Xcm
(Mass fraction of fragment larger than X)

X = screen size (fragment size)

X_c = characteristic size or average fragment size (50 percent mass passing, derived from Kuznetsov equation)

n = Uniformity index, 0.5 - 2.0

(b) Kuznetsov equation

$$X_c = A(K)^{-0.8}\ Q^{0.144}\ (E/E_{TNT})^{-0.633} \qquad (8)$$

where

X_c = Mean size in Rosin-Rammler equation, cm

A = Rock hardness index, 1-22

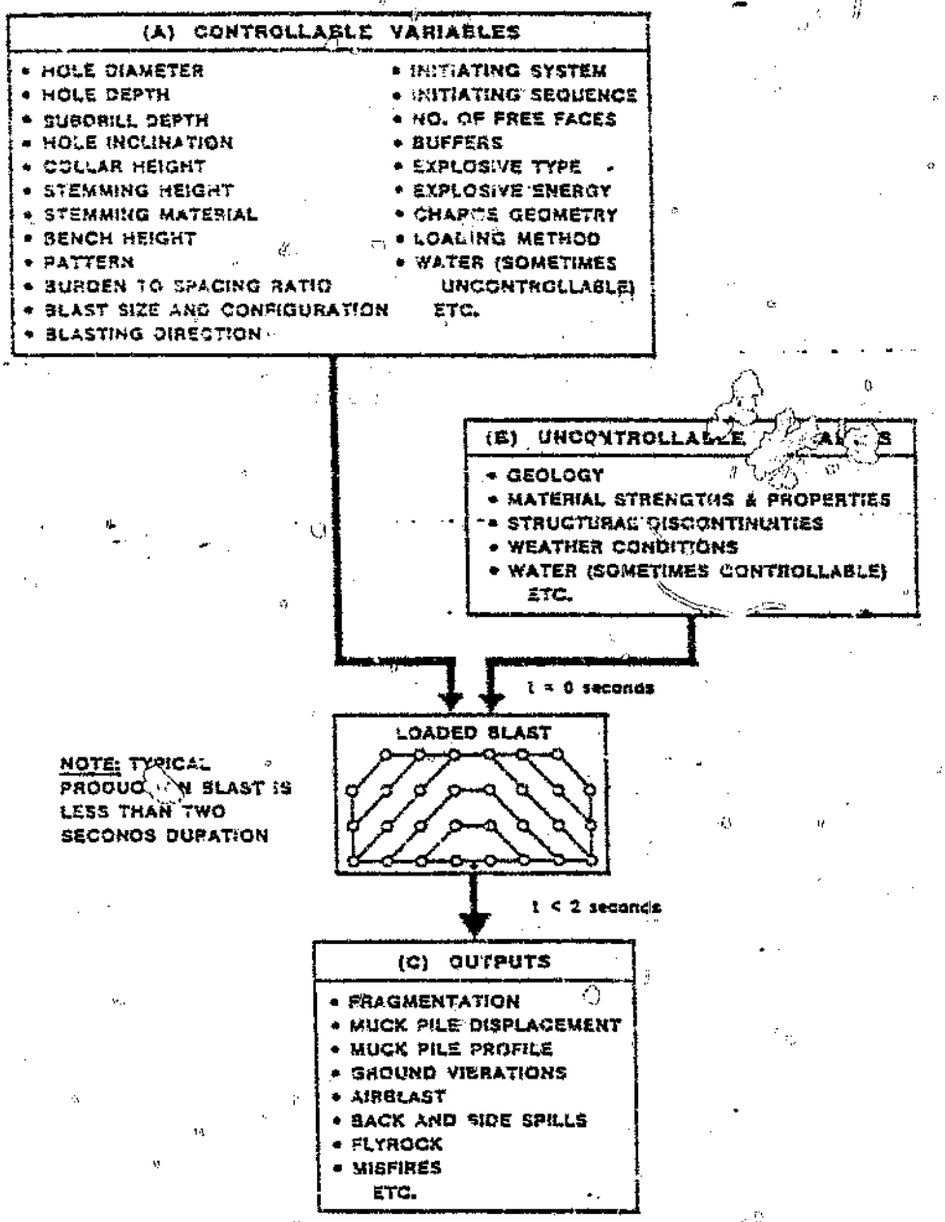
K = Powder factor, Kg/m^3

Q = Kg explosive

E = Relative Weight Strength of explosive.

E_{TNT} = Relative Weight Strength of TNT

The 'ROCK HARDNESS INDEX, A' is defined in terms of rock mass description, density influence and hardness as follows:



(After Chiappetta, 1983)

Figure 4.4 Field model Illustrating Blast design Inputs and outputs.

$$A = (RMD + RDI + HF) \times 0.06 \qquad (9)$$

where

RMD = Rock Mass Description,
(Friable = 10
Massive = 50
Vertically jointed = JF*)

JF = Joint Factor (defined below)
Depends on drill pattern, joint spacing and joint condition.

RDI = Density influence
(RDI = 25 x Density - 50)

HF = Hardness Factor
(if Y > 50 GPa, HF = UCS/3)
(if Y < 50, Gpa, HF = Y/3)[1]

The joint factor, JF is defined as

$$JF = (JCF \times JPS) + JPA \qquad (10)$$

where

JCF = Joint Condition Factor,
(Tight = 1, Relaxed = 1.5,
Gouge filled = 2.0)

JPS = Joint Spacing across Pattern
(<0.1m, JPS = 10
<DP, JPS = 80
>DP, JPS = 20
DP, $(B \times S)^{0.5}$)

JPA = Joint Plain Angle factor,
(Dip out - 20
Dip in - 40
Dip perp- 30)

[1](Y in Gpa, UCS in Mpa, Density in g/cc)

Cunningham (1991) has derived the exponent of uniformity 'n' in the Rosin-Rammler equation from blasting parameters:

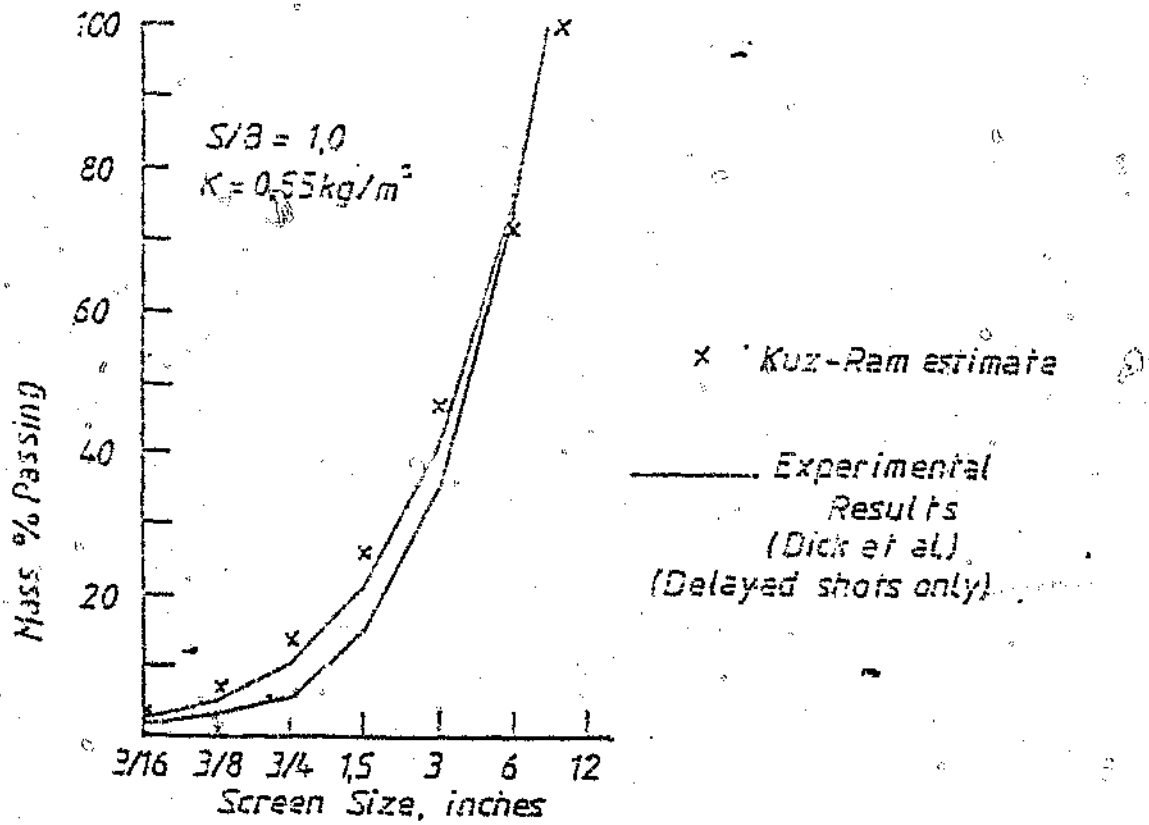
$$n = (2.2-14B/d)\ ((1+S/B)/2)^{0.5}\ (1-W/B)L/H \qquad (11)$$

where

B = Burden, m
S = Spacing, m
d = Hole diameter, mm
W = Std Dev of drilling accuracy, m
H = Hole length, m
L = Charge length, m

The uniformity exponent 'n' of the Rosin-Rammler expression is more critical below 1, and the normal range is between 0.75 and 1.5.

Based on measurements of fragment size at a wide and varied number of blasts, the KUZ-RAM model has been found to give remarkably consistent and reliable results in both small- and large-scale blasting operations. It is a useful tool for predicting maximum fragment size and also for comparing implications of various blasting layouts. Richard *et al*, 1976 have measured fragmentation from blasting in limestone using 22m blastholes. The correlation was good by any standard and agreement was over the whole size range (Figure 4.5). For an overburden blast at an Australian coal mine, Lean and Paine (1981) showed that the KUZ-RAM model gives extremely good correlation in the coarser section of the curve but rather indicates more fines than given by photographic analysis (Figure 4.6). However accuracy is generally more important for the coarse fraction than for the fines.



(Source: Richard et al, USBM Report on Inv. 7704)

Figure 4.5 Measured Values and KUZ-RAM Prediction of Mean Fragment Size

To use the KUZ-RAM model for the analysis of fragmentation at the Syferfontein operation, a spreadsheet was developed by the author using the input parameters illustrated in Table 4.4. The inputs are categorised under blast design parameters, rock factors and explosive properties as these are the major factors that affect fragmentation. As this project report is based on the studies confined to interburden blasting on

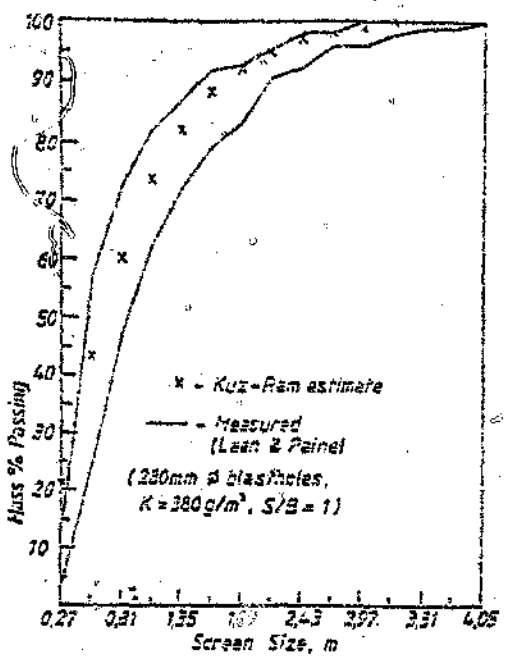


Figure 4.6 Measured values and KUZ-RAM prediction for fragmentation in limestone.

the West bank, strips 5 and 6 where fragmentation problems were experienced, each of the twelve blast (blocks) has been analyzed separately. The outputs from the computer are the uniformity index or exponent and the mean fragment size. As the principal argument is to compare the results of the various blasts in terms of fragmentation, the uniformity index and the mean fragment size of each blast are shown on the same Table 4.4. It can be seen that the uniformity index or exponent of all the blasts is greater than 1 and the mean size ranges from 27.55 to 32.54cm which apparently portrays averagely satisfactory results. Nevertheless for optimum fragmentation and a sufficiently loose muckpile affording easy penetration of the dragline bucket teeth, the uniformity index value is expected to fall within 1.15 - 1.25, a situation scarcely attained from Table 4.4 (Cunningham, 1992b).

Figures 4.7a and 4.7b illustrate the Rosin-Rammler fragmentation curves based on the input parameters given

Table 4.4 - INPUT PARAMETERS FOR FRAGMENT SIZE ANALYSIS

- KUZRAM MODEL FOR DIFFERENT BLASTS -

BLOCK No.	3205109	320587	320576	320575	320543	320531	320501	3203108	320678	320676	320665	320643
Burden (B)	10.2	10.2	10.2	10.2	10.2	10.2	10.5	10.5	10.5	10.5	10.5	10.5
Spacing (S)	10	10	10	10	11	11	11	11	11	11	11	10.5
Hole Diameter (d)	250	250	250	250	250	250	250	250	250	250	250	250
Std. Dev (w)	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3
Hole Length (H)	27	28	27.5	27.5	[illegible]	27.5	27	27.5	27	28.5	28	26.6
Charge length (L)	18	20	19.5	17.5	[illegible]	17.5	19	17.5	17	18.5	23	21.03
Young Modulus	15	15	15	15	15	15	15	15	15	15	15	15
Density	2.57	2.57	2.57	2.57	2.57	2.57	2.57	2.57	2.57	2.57	2.57	2.57
Rk Mass Desc.	50	50	50	50	50	50	50	50	50	50	50	50
Density influ.	14.25	14.25	14.25	14.25	14.25	14.25	14.25	14.25	14.25	14.25	14.25	14.25
Hardness Factor	5	5	5	5	5	5	5	5	5	5	5	5
Rk Hardness Index	4.16	4.16	4.16	4.16	4.16	4.16	4.16	4.16	4.16	4.16	4.16	4.16
Powder factor	0.48	0.44	0.49	0.44	0.39	0.48	0.43	0.45	0.48	0.4	0.49	0.48
Kg Explosive/Hole	1171	1273.78	1310	1200.7	1116.7	1234.3	1239.5	1265.5	1449.4	1288.6	2004.7	1306.5
RWS of Explosive	83.3	83.3	83	83.3	83.3	83.3	83.3	83.3	82.5	82.5	82.5	82.5
RWS of TNT	131	131	131	131	131	131	131	131	131	131	131	131
* Uniformity Index	1.06	1.13	1.12	1.01	1.03	1.03	1.14	1.02	1.01	1.04	1.31	1.28
* Mean size	27.57	29.93	27.56	29.66	[illegible]33	27.78	30.53	29.36	28.6	32.54	29.48	28.18

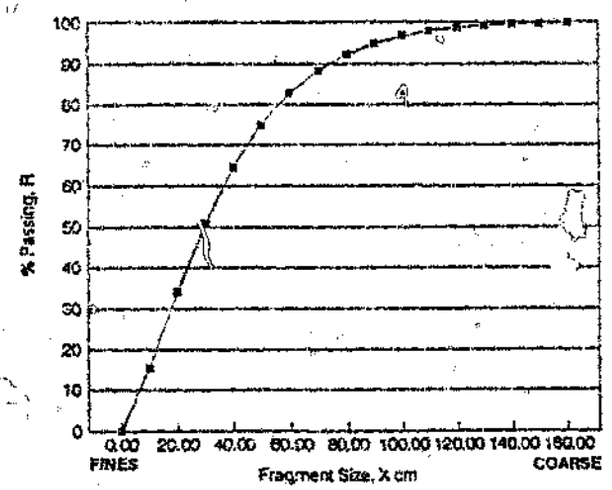


Fig 4.7a: Rosin Rammler fragmentation curve
- high uniformity index, n = 1.31 -

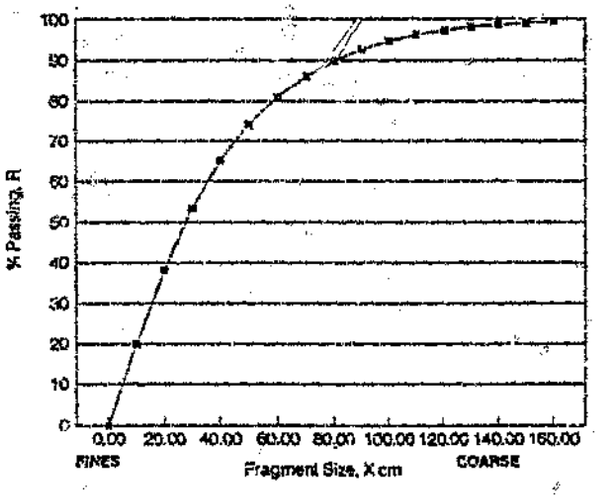


Fig 4.7b Rosin-Rammler Fragmentation Curve
- Low uniformity index, n = 1.01 -

in Table 4.4. Similar curves for the other blasts are shown in Appendix B . In principle the curve shows the percentage passing at the various size fractions equivalent to different screen sizes. Furthermore the curve shows the distributions of fines, coarse and mid-range sizes of each blast. The S-shaped curve gives the best distribution. A typical example is exhibited by Figure 4.7b.

Fines in this context have been defined as any size smaller than 5cm as the dragline bucket finds it well-nigh impossible to pick up rock of this size. Oversize with respect to the dragline bucket is defined here as sizes larger than 140cm i.e. one-third the width of the dragline bucket (Cunningham, 1993). According to Cunningham, any blasting operation continually resulting in sizes greater than 80cm should be reviewed. Against this background, sizes in excess of 80cm have also been analyzed from the Rosin-Rammler curve. A summary of the percentage fines (-5cm) and oversize (fragment in excess of 80cm as well as 140cm) for the various blasts, as deduced from the Rosin-Rammler curve are shown in Table 4.5. It can be deduced from Table 4.5 that the percentage fines range from 7 to 11 with a mean value of 9.31% and oversize (>80cm) range from 8 to 18.8 with a mean value of 13.36%, and boulders greater than 140cm range from 0.2 to 4.5% with a mean value of 2.07%. The results of the fragmentation analysis are discussed below.

4.2.2 Discussion of results

A study of Table 4.5 reveals uniformity index 'n' values ranging from 1.01 to 1.28. Cunningham (1991) has shown through a series of studies that the normal range of 'n' is between 0.75 and 1.5 with an average around 1.

Table 4.5 Percent Fines and Oversize Deduced from Rosin-Rammler Curve

BLAST NUMBER	MEAN SIZE	UNIFORMITY INDEX	FINES (% < 5cm)	OVERSIZE (% > 140cm)	BOULDERS (% > 80cm)
3205109	27.57	1.06	10	1.2	12
320587	29.93	1.13	8	2	12.5
320576	27.58	1.12	10.5	1	10
320575	29.66	1.01	11	3	13
320543	32.33	1.03	9	5	18
320531	27.78	1.03	11	1.5	11.5
320501	30.63	1.14	8.5	4.5	18.8
3206109	29.36	1.02	10.2	2.3	14.5
320678	28.6	1.01	10	2	14
320676	32.54	1.04	9	4	17.5
320665	29.48	1.31	7	0.2	8
320643	38.14	1.28	7.5	0.3	8.5
MEAN	29.45		9.31	2.07	13.36
RANGE	27.56-32.54		7 - 11	0.2 - 4.5	8 - 18.8

Therefore the range of 1.01 to 1.31 observed in this study appears satisfactory. It is noteworthy that values of 'n' below 0.75 represent a situation of 'dust and boulders' which if it occurs on a wide scale in practice, indicates that the rock conditions are not conducive to control of fragmentation through changes in blasting. Variations in the uniformity index are more critical to oversize and fines for values below 1. For n = 1.5 and higher, muckpile texture does not change much and errors in judgement are less punitive. As seen in the previous section optimum fragmentation and sufficiently loose muckpile affording easy penetration of the dragline bucket teeth is achievable if the uniformity index falls within 1.15 and 1.25.

As mentioned earlier, percentage fines range from 7 to 11

with a mean value of 9.31%. A large percentage of fines indicate excessive use of explosive. This same situation may lead to low fill factors or bucket factors of the dragline. However a mean value of 9.31% is not sufficient to warrant much attention. Furthermore handling of fines as such does not normally pose a problem for the mine.

The proportion of boulders in excess of 140cm ranges from 0.2 to 4.5%. This indicates that there is virtually no oversize per se with respect to the dragline bucket ($75m^3$). Nevertheless boulders in excess of 80cm range from 8 to 18.8% with a mean of 13.36% and calls for concern. With on average more than 10% oversize i.e. greater than 80cm uncovered by the model it is worth reviewing the blasting operation. The 'Table Top effect' or completely unfragmented section which was encountered on two occasions confirms this argument. These occasions obviously called for secondary breakage. The reasons for the occurrence of the 'Table Top effect' are explained in the next chapter.

The mean fragment size ranges from 27.55 to 32.54cm. This measurement, although giving a rough idea of the entire fragmentation in quantitative terms it does not have any practical significance as such. One of the important attributes of the KUZ-RAM model is its use in analyzing the effect of some major variables that affect fragmentation. This is the subject of the next section.

4.2.3 Sensitivity analysis

There are a host of factors, ranging from rock factors through design to explosive factors, that affect the fragmentation of rock. Since the burden spacing ratio and powder factor are the major factors which can be easily varied on site, sensitivity analysis using the

model have been conducted on the [illegible]. The results of varying the spacing burden ratio [illegible] powder factor are given in Tables 4.6 and 4.7.

Table 4.6 SENSITIVITY ANALYSIS

- BURDEN : SPACING RATIO -

SPACING / BURDEN	11.00	10.50	10.00	9.50	9.00	8.50	8.00
10.50	*1.14* 10.5x11	*1.13* 10.5x10.5	-	-	-	-	-
10.00	*1.17* 10x11	*1.16* 10x10.5	*1.15* 10X10	-	-	-	-
9.50	*1.21* 9.5x11	*1.19* 9.5x10.5	*1.18* 9.5X10	*1.16* 9.5x9.5	-	-	-
9.00	*1.24* 9x11	*1.23* 9x10.5	*1.21* 9X10	*1.20* 9x9.5	*1.18* 9x9	-	-
8.50	*1.28* 8.5x11	*1.26* 8.5x10.5	*1.25* 8.5X10	*1.23* 8.5x9.5	*1.21* 8.5x9	*1.20* 8.5x8.5	-
8.00	*1.32* 8x11	*1.30* 8x10.5	*1.29* 8X10	*1.27* 8x9.5	*1.25* 8x9	*1.23* 8x8.5	*1.23* 8x8
7.50	*1.36* 7.5x11	*1.34* 7.5x10.5	*1.33* 7.5X10	*1.27* 7.5x9.5	*1.29* 7.5x9	*1.27* 7.5x8.5	*1.27* 7.5x8
7.00	*1.41* 7x11	*1.39* 7x10.5	*1.37* 7X10	*1.35* 7x9.5	*1.33* 7x9	*1.31* 7x8.5	*1.31* 7x8

* Top Right hand corner figures refer to uniformity index (n values)

Table 4.7 SENSITIVITY ANALYSIS
- POWDER FACTOR -

POWDER FACTOR (Kg/m3)	MEAN SIZE (Xm) (cm)	OVERSIZE % > 140cm	OVERSIZE % >80cm
0.49	27.45	0.08	4.09
0.48	27.91	0.10	5.34
0.47	28.38	0.11	5.71
0.46	28.87	0.13	6.19
0.45	29.38	0.16	6.52
0.44	29.92	0.18	6.97
0.43	30.47	0.21	7.44
0.42	31.05	0.25	7.95
0.41	31.66	0.29	8.50
0.40	32.29	0.34	9.07
0.39	32.95	0.40	9.69
0.38	33.64	0.47	10.34

The uniformity exponent is the output of such an analysis or index of measurement. The subjects of burden and spacing have been discussed earlier in Sections 4.2.1 and 4.2.2 respectively when reviewing the existing practice on the mine and are considered here again .

It can be seen from Table 4.6 that the various combinations of burden and spacing give uniformity indices ranging from 1.13 to 1.41. However, as mentioned earlier, to optimise fragmentation and muckpile displacement (the two basic necessities for efficient dragline operation), the uniformity index should not fall below 1 and practically be restricted to 1.15 to 1.25. Therefore, to balance the competing priorities of the economics of drilling and blasting on one hand and good fragmentation affording easy bucket penetration on the other, a burden and spacing of 8.5 x 9.5m giving a uniformity index of 1.23 is recommendable. This supports the pattern arrived at using the empirical relation given by equation (3) and (5) above. A critical examination of Table 4.6 reveals that the existing pattern on the mine, viz 10.5 x 10.5m, 10.5 x 11m and 10 x 10.2m, gives corresponding indices of 1.13, 1.14 and 1.14 respectively all of which fall outside the practical optimum. This reinforces the conclusion that with this burden spacing ratio, the problem of fragmentation can be expected and in fact does exist.

The percentage oversize (>80cm) and mean sizes at various powder factors are shown in Table 4.7. A powder factor of 0.49kg/m^3 is the best from the results shown. Nevertheless from economic considerations a powder factor of 0.45kg/m^3 which represents the average for the blasts considered under this project can give equally satisfactory results. Such powder factors (Table 4.4) employed by the mine should not give fragmentation problems. Contrary to expectation few blasting problems

such as the table top effect necessitating secondary breakage with its associated unbudgeted costs as well as low dragline digging efficiencies, have been encountered in certain blasts. Apparently the gains to be expected from higher powder factors are negated by using larger drill patterns. As a consequence certain blasts were monitored to assess the cause of fragmentation problems and this is the issue in the next chapter.

4.4 Summary of Chapter

The existing practice (design) on the mine has been critically reviewed and empirical relations used where appropriate. Fragmentation has been assessed through a model and sensitivity analysis has been conducted on key variables that affect fragmentation. The following results are quite evident:

- The burden and spacing used on the mine are incompatible with the hole diameter (a major determinant of burden and spacing ratio) and appears to be too much. A burden-spacing ratio of 8.5 x 9.5m has been recommended which, to all intents and purposes, will minimise the damaging effects of using an excessive pattern and yet achieve, at lowest cost, the required breakage, displacement and muckpile looseness for an efficient digging operation of the dragline.

- The 42ms and 67ms delay interval used between rows do not practically conform to the existing drill pattern. However with the recommended new drill pattern (8.5 x 9.5m), an 84ms delay is appropriate and the status quo must be maintained if the new drill pattern is to be used.

- The uniformity index or exponent of the Rosin-Rammler model for all blasts investigated is greater than 1 and hence falls within the normal range of 0.75 to 1.5. Nevertheless optimum fragmentation and muckpile displacement is achievable if the index falls within 1.15 to 1.25 and this condition is satisfied by the drill pattern recommended above.

- With the recommended drill pattern a powder factor of 0.45kg/m^3 is appropriate and will not jeopardise fragmentation. This will yield corresponding percentage oversize in excess of 80cm or 6.52%, a situation which can be handled by the dragline without much problems.

CHAPTER 5

BLAST MONITORING

The cost of drilling and blasting forms a significant part of the overall mining costs and it is therefore important that the most efficient use be made of explosives for maximising profitability. Often when poor blasting results occur the tendency is to increase drilling and explosives use because the real cause of the problem is not known. This is usually done subconsciously where high powder factors compensate for problems related to the performance of the explosives and initiation system or poor blast design. As mentioned in the previous chapter, very large boulders and portions of completely unfragmented rock (table top effect) were encountered during the interburden blast at Syferfontein Colliery. This usually hinders the dragline productivity, an inimical situation the mine cannot afford to accommodate. It was therefore imperative to monitor certain blasts in order to address performance related problems, and as a bonus, to practically and cost effectively optimise blast results. Velocity of detonation (VOD) measurement forms a significant part of this monitoring exercise. High speed photography measurement was also done to complement the results of VOD measurement. This monitoring forms part of the optimisation work on blasting launched by the mine as a continuous programme to maximise profitability and efficiency in the drill and blast operation.

The objectives of this monitoring programme as covered in this chapter are therefore:

- To assess the in-situ performance of explosives as related to the explosive quality.
- To verify the performance of the initiation system in terms of timing accuracy and reliability and its effect on overall blasting performance such as fragmentation.
- To assess the in-hole as well as surface delay performances.
- To check the sequence of holes firing in a row, i.e. effects of out-of-sequence firing of holes connected to a surface trunkline.
- To determine the effects of decking and decking material on fragmentation.
- To assess the performance of full column charge as opposed to deck charging.

In this chapter, the studies have been categorised under three different scenarios and are presented as abbreviated cases. It covers such topics as deck performance assessment, sequential firing analyses through high speed photography and impact of full column charge on fragmentation. It must be reiterated that VOD measurements form the central core of this monitoring exercise. Although a model has been used in the previous chapter to assess fragmentation, the real causes of blasting-related problems are uncovered through physically monitoring the blast, a subject given much attention in this chapter.

5.1 Blast Monitoring : Deck Performance Trials

Two separate blasts were monitored on 10/3/93 and 18/3/93 to assess the explosive performance and also to determine the effects of decking and decking material on fragmentation. The explosive used was Doped Emulsion of relative density 1.31. Decking was of concern to the

mine because of a soft shale layer, (5 meters thick) sandwiched between two hard sandstone layers. Less energy is therefore needed in this layer which was hitherto accomplished by reducing the explosive load per hole. A decking material was placed in the centre of the shale, leaving the harder sandstone layers at the top and the bottom of the hole in contact with the explosive. Monitoring was done by measuring the VODs of the explosive of both the top and bottom charges.

5.1.1 Instrumentation

The VODs of the explosive in the blasts were measured 'in situ' using a VODR-1 VOD recorder. Coaxial sensor cables were used, looping the cable from one hole to the next (Figure 5.1). Fourteen holes were monitored (nine holes for the blast on 10/3/93 and five holes for blast on 18/3/93) using RG-58 coaxial cable. Different stemming materials were used, consisting of crushed stone, coal ash and air bags. The position of the deck in these six holes was kept within the shale layer as shown in Figure 5.2.

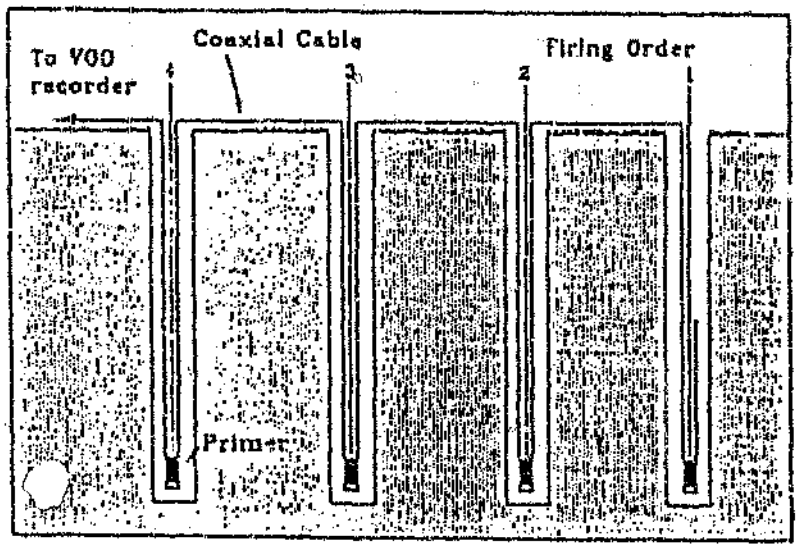


Figure 5.1 Schematic Showing VOD Monitoring and Cable Installation

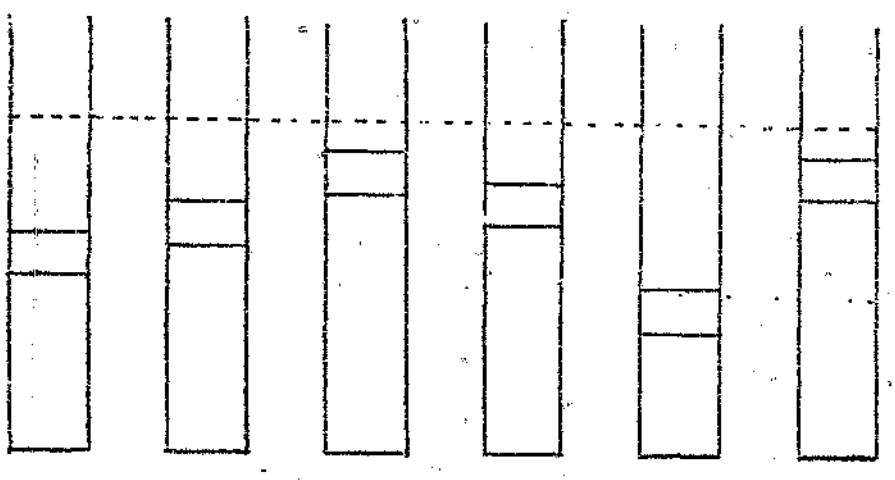


Fig 5.2 A diagram showing the position of deck load in holes monitored for VOD.

5.1.2 Results

The results of the VOD measurements are summarised in Table 5.1. Also shown in the Table is the precise firing time of both the top and bottom deck deduced from the VOD traces which allows the assessment of the in-hole delay. For interpretation of VOD trace, refer to figure 3.1. A quick glance at the table reveals that all the holes except hole 8 on block 320575 saw VODs below characteristic values. This is a clear manifestation of the unsatisfactory performnace resulting from poor judgement or failure to sensitise the product.

Concern is warranted when the VODs fall below the characteristic range, indicating that some of the ingredients are not reacting at all. This could be attributable to a host of factors including borehole water, poor mixing or loading of

Table 5.1 VOD Measurements: Decked holes

BLAST DATE (1)	BLOCK (2)	EXPLOSIVE DENSITY (3)	HOLE (4)	MEASURED VOD (m/s) TOP (5)	MEASURED VOD (m/s) BOTTOM (6)	CHAR. VOD (m/s) (7)	FIRING TIME (ms) TOP EXPECTED	FIRING TIME (ms) TOP ACTUAL	FIRING TIME (ms) BOTTOM EXPECTED	FIRING TIME (ms) BOTTOM ACTUAL
		1.31	1	–	894	4500 – 5000	0	2.3	0	2.3
		1.31	2	957	850	4500 – 5000	85	77.18	60	60.42
		1.31	3	619	953	4500 – 5000	135	139	110	117.98
		1.31	4	713	1813	4500 – 5000	175	173.47	150	151.68
10/3/93	320675	1.31	5	700	1516	4500 – 5000	225	210.93	200	198.45
		1.31	6	1323	–	4500 – 5000	95	94.25	70	77.93
		1.31	7	–	1444	4500 – 5000	145	120.92	120	135.61
		1.31	8	4565	74	4500 – 5000	175	176.06	150	176.02
		1.31	9	–	3786	4500 – 5000	225	227.06	200	221.5
		1.31	1	756		4500 – 5000	25	–	0	–
		1.31	2	4691		4500 – 5000	50	49.36	25	–
18/3/93	320543	1.31	3	1194		4500 – 5000	80	56.34	35	–
		1.31	4	1339		4500 – 5000	87	45.74	42	–
		1.31	5	2706		4500 – 5000				
		1.31	1	–	737	4500 – 5000	25	45.4	0	0.96
26/3/93	320531	1.31	2	4505	912	4500 – 5000	50	50	42	46.34
		1.31	3	–	778	4500 – 5000	120	80.44	100	126.78
		1.31	4	–	889	4500 – 5000	175	44.52	130	171.3

explosive, density, inadequate priming or desensitisation due to shock from earlier firing holes (Ladds *et al.*, 1992). A more adequate explanation could probably lie in the quality of the explosive because the necessary precautions with regard to loading and priming were taken. Nevertheless further experimentation (more monitoring) was warranted. More blasts were therefore monitored the results of which will be seen in the subsequent sections (see Section 5.3 and 5.4). With VODs below the characteristic values, the explosive reaction proceeds abnormally, detonation becomes unstable and the secondary reaction b nd the detonation front may not reach completion and as a consequence energy output drops drastically. It is quite logical to expect that fragmentation would be jeopardised with such low VODs. Contrary to expectation fragmentation was apparently good by the visual assessment. Many different plausible arguments could be advanced to explain these findings. This issue is therefore given further consideration in the next chapter where dragline diggability is critically assessed and related to fragmentation as a surface view of the muckpile can be misleading. Also seen from the table are initiation related problems. This is well illustrated in Figure 5.3. For instance sympathetic detonation of the top deck with the bottom deck occurred in hole 1 as can be seen from the VOD trace in Figure 5.4a. In holes 7 and 9 interference occurred with the cable with the result that no VODs were recorded.

For the blast monitored on 18/3/93 (Block 320543), the top charge fired before the bottom for all of the five holes monitored. Furthermore only hole 2 fired at characteristic VOD. Figure 5.5 is a graphical presentation of the results. Figure 5.4a and b are the VOD traces illustrating the top firing before the bottom although timed to initiate the reverse. The most plausible explanation for such an occurrence might be

either frequent failure of the bottom units or timing problems where the top 525 ms units are firing before the bottom 500 ms units. The boulders and difficult

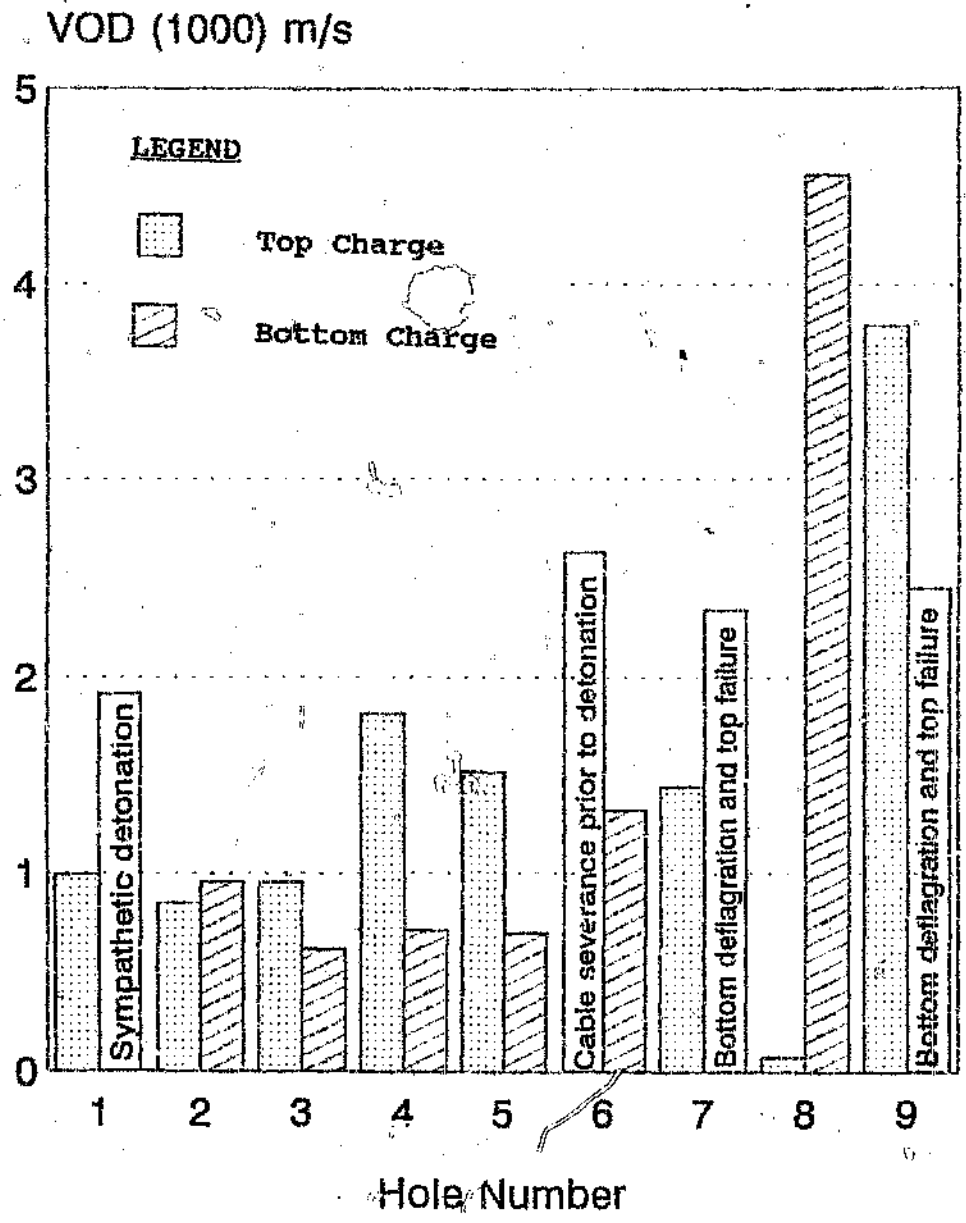


Figure 5.3 Bar Chart showing VOD measurements and Initiation related problems.

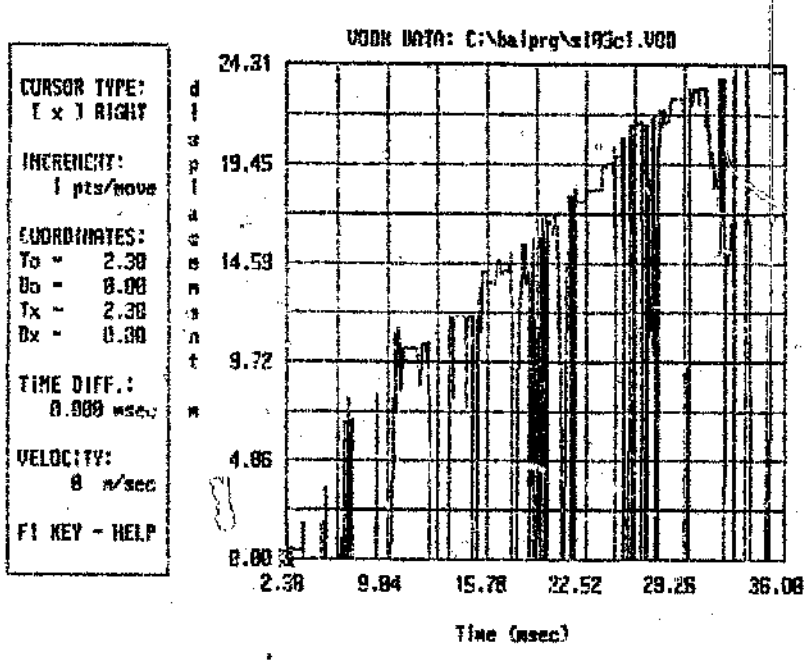


Figure 5.4 VOD trace illustrating instantaneous sympathetic detonation of top charge with bottom Charge

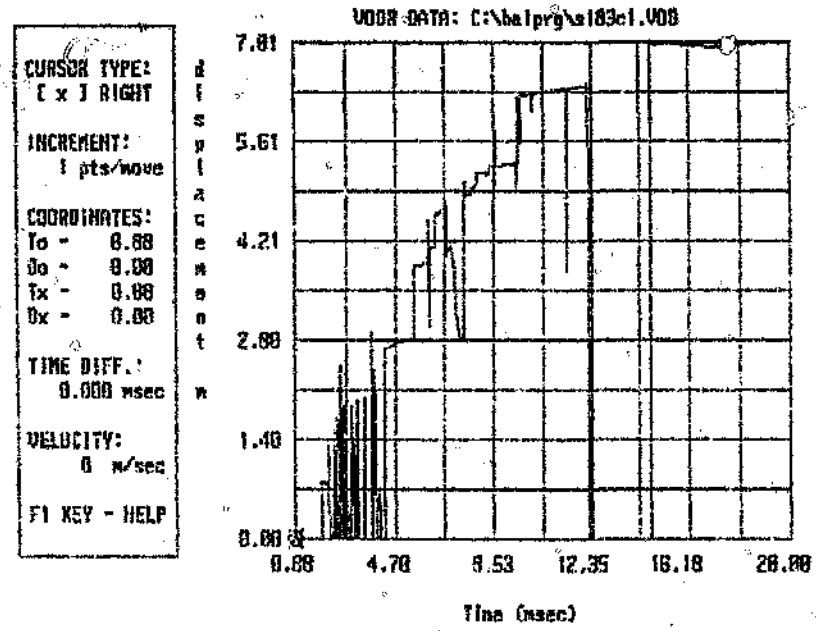


Figure 5.4 (a) VOD trace illustrating bottom firing and top charge failure.

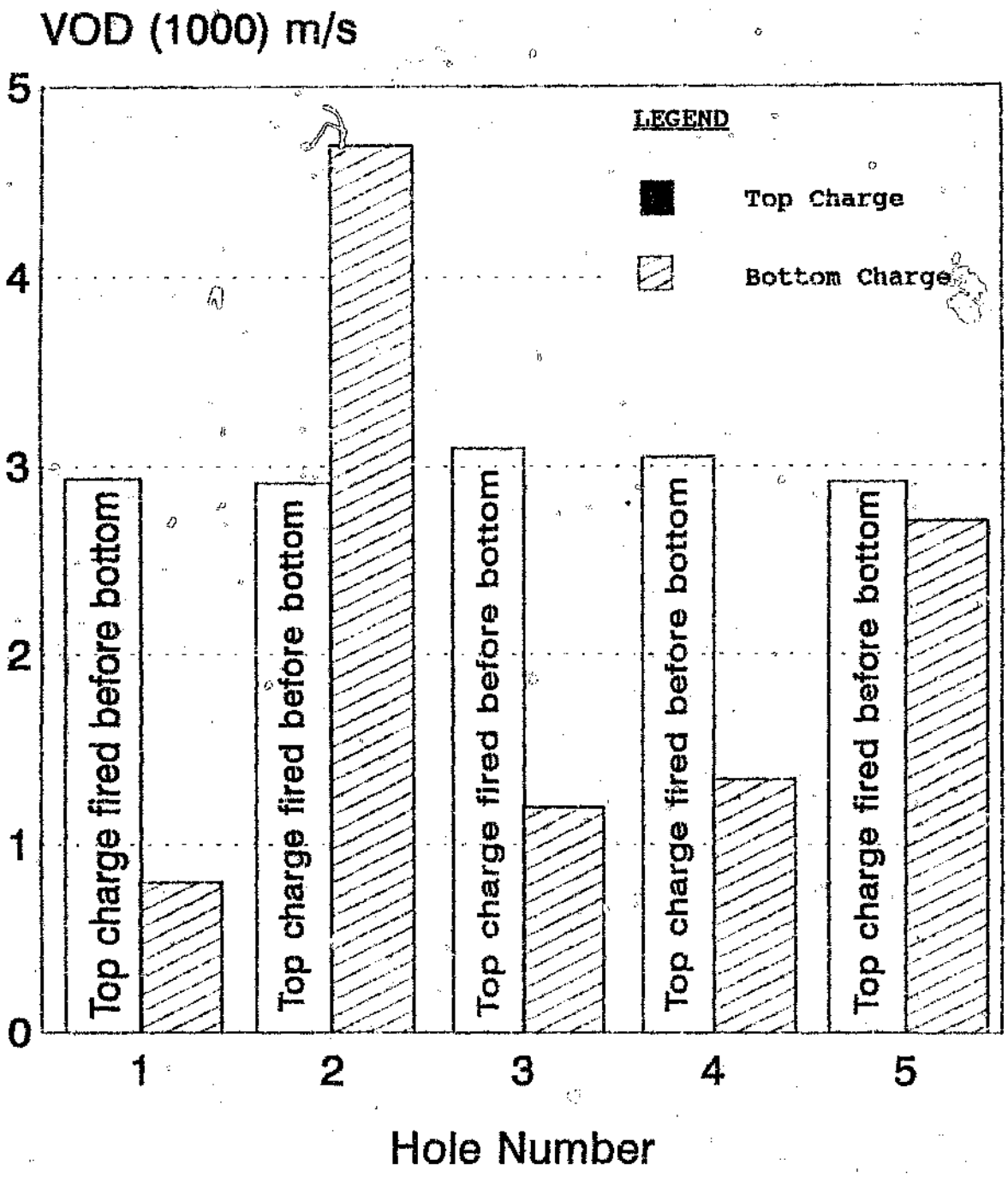


Fig 5.5 Bar chart showing average VOD measurements where top charge fired before bottom charge.

digging observed in this area were not unexpected. This prompted an investigation into the initiation system performance in terms of timing accuracy. This is dealt with in Section 5.4. No meaningful conclusions could be drawn regarding the performance of the various decking materials as virtually all the charges fired at VODs considerably lower than the characteristic range. This situation could drastically affect fragmentation results. Top charge firing before bottom charge for the blast monitored on 18/3/93 exacerbates the problem and would make any conclusions suspect.

5.2 Blast Monitoring : Holes Firing Sequence

A blast was monitored at Syferfontein on 26 March 1993 to determine the sequence of holes firing in a row. The aim was to test, through high speed photography, the reliability of the 42 ms delay between rows and the effects of out-of-sequence firing of holes connected to a surface trunkline. Some areas which were not monitored by the VOD recorder were also assessed using the high speed film. The use of high-speed photography in blast evaluation, blast design and field control has been well established; Blair (1960), Potel (1961), Chunk *et al.* (1975) and Chiapetta *et al.* (1983). As a diagnostic field tool, the data obtained from high speed photography for use in blast evaluation, blast design, and improving productivity are useful for determining (Chiapetta *et al.* 1983):

- cause of misfires
- poor loading practices and poor firing practices
- effects of redrills, water, gas venting and buffers
- proper delay interval between holes and rows of holes to obtain adequate forward burden relief

- optimum initiation system
- massive ground movement
- source of oversize and explosive efficiency
- optimum cast of fragmented material and muckpile profile
- optimum explosive - rock burden combination.

For the monitoring exercise in question, E-cord flares were used on each hole to visually show when the explosives in the top of the hole detonated. These flares were approximately one meter into the explosive charge in the top deck of every blast hole and were standing 1.5 meters high above surface. VOD was also monitored for 4 holes to assess the performance of the explosive. The blast was charged with ungassed E-6000 in two decks in each borehole. Decking of 4 meters was done using gas bags approximately 10 meters from the bottom of the hole. A typical cross-section through such a blasthole is as shown in figure 4.3.

5.2.1 Instrumentation

The blast was filmed using a LOCAM II high speed camera set at 500 frames per second, giving a time resolution of 2ms and thus providing a detailed record of the performance of the surface initiation system. The camera was located 300m from the blast. The VODs were measured using a VODR-1 VOD recorder with FSJI-50A coaxial cable looped between the holes. The tie-up of the blast is shown in Figure 5.6. Unfortunately, the camera was not running at full frame speed when the blast was initiated, but reached full speed after the initiation of row 12 on the blast. Delay times of the first 12 rows could therefore not be determined accurately.

5.2.2. Discussion of Results

The delay times measured for rows 13 to 32 are given in Table 5.2. The spread of delay times of the 42ms relays is shown in Figure 5.7. Such a wide variation (scatter range of 10 ms) places a question mark of the accuracy of the the 42 ms surface delay. Flashes of the flares could only be seen in eight of the rows in the southern portion of the blast. The sequence in which the flares ignited is shown in Table 5.3. The results of the VOD measurements are also presented in Table 5.1.

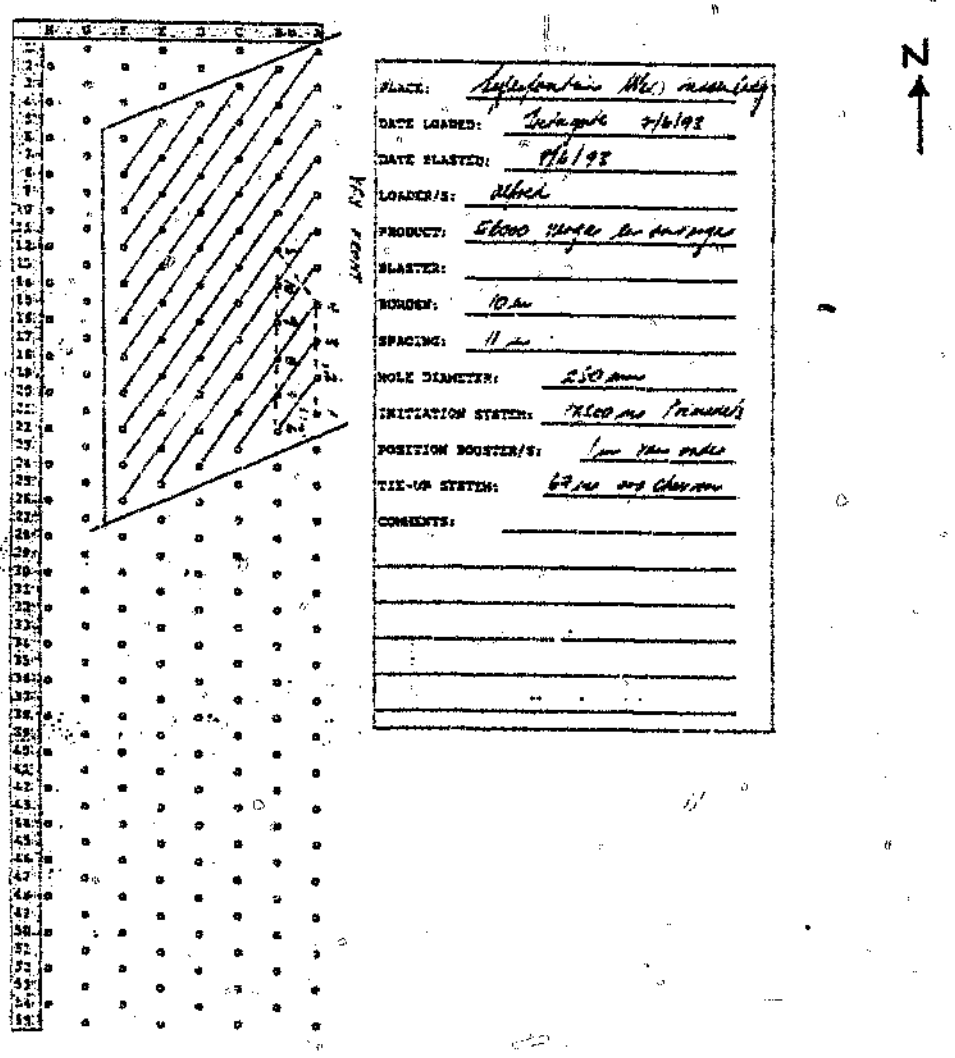


Figure 5.6 A Blast Tie-up in which VOD and High Speed Photography were monitored.

Table 5.2 Firing Times of 42 ms NTD Relays

ROW	DELAY (ms)	ROW	DELAY (ms)
13	34	*23	
14	34	24	40
15	38	25	36
16	38	26	42
17	44	27	40
18	40	28	42
19	38	29	36
20	40	30	44
21	36	31	40
22	36	32	40
RANGE		= 10	
MEAN DELAY		= 38.9	
STD		= 2.9	

* Row 23 was skipped and the firing time between Rows 22 and 24 was 40 ms.

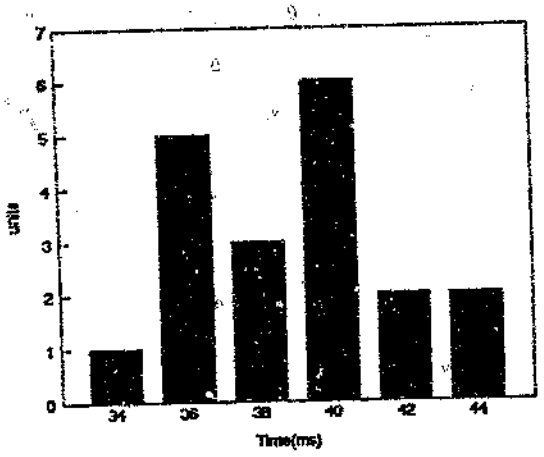


Fig 5.7 Delay range of 42ms NTD

Figure 5.8 illustrates graphically the VOD measured in both top and bottom charges and the performance related problems.

The delay times determined with the high speed camera showed no trunklines detonating out of sequence. The only problem experienced was the failure of the trunkline in row 23. The delay time between rows 22 and 24 was measured at 40 ms, indicating the possibility that the trunkline in row 23 was not tied in at all.

The sequence in which the flares detonated is given in Table 5.3. Unfortunately, the flashes of the flares of only eight of the rows could be seen clearly, due to the amount of dust caused by the ignited trunklines and stemming ejection on the blast. The flares in this portion of the blast were tied 1.5 m high, making them visible above the dust.

Table 5.3 Sequence of Holes Firing in a Row

ROW	1	2	3	4
18	1	4		
19	1	2	4	
20	1	2; 3		
21	1; 2	3	4	
22	2; 3	1	4	
24	2	1	3	
25	2	3	1	
26	1	2	3	5

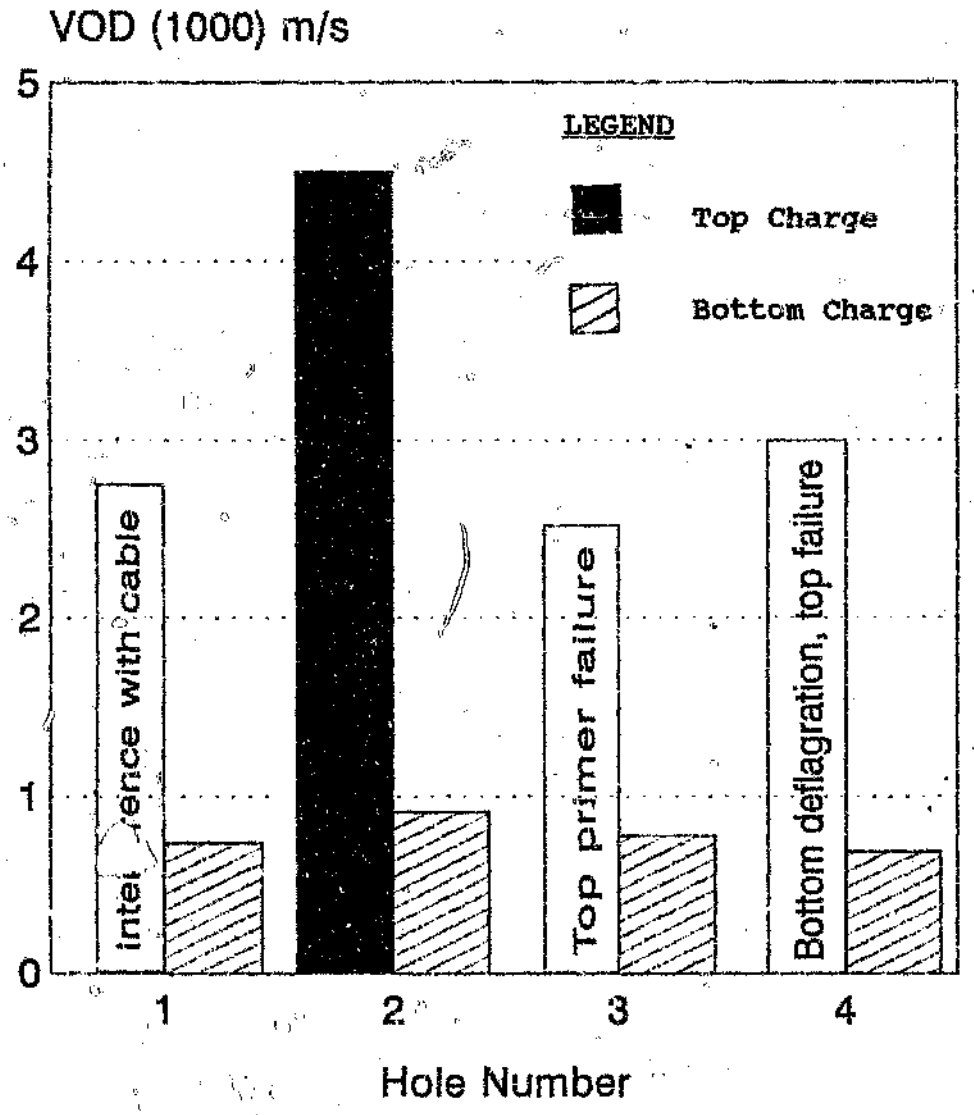


Fig 5.8 A schematic diagram showing VOD measurements in decked holes and performance related problems.

In three out of the eight rows (38%) the flares did not ignite in sequence. The time difference between the flares igniting was 10 ms in the worst case. Such a scenario (out-of-sequence firing) can easily result in poor blast performance. The difference in timing also indicates a questionable performance in terms of the accuracy of the EBSA MS-delay units down the hole.

Another interesting feature of the blast was that no flashes of flares nor stemming ejection could be seen in the area of row 23 and in the back portion of the bench adjacent to this row. It was in this area that a table top effect was seen after the blast. In five of the holes in this area, the un-ignited E-cord flares were seen after the blast, indicating that the top of the explosive column was not in contact with the E-cord. This could have been caused by no detonation of the explosives or no explosives in the deck or slumping of the explosives. The latter could have been caused by slipping of the gas bags used for decking and is considered to be the most likely explanation.

Of concern also was the instantaneous detonation of the explosives in hole 2 in row 9 when the trunkline was initiated. This could have been caused by the detonator failing or by the trunkline initiating the E-cord flare which detonated the explosives. The latter is the more likely scenario. In this section of the blast the E-cord flares were initially attached to the booster in the top deck of the hole. This was changed and the possibility exists that this one was overlooked, since E-6000 is not known to be cord sensitive.

The detonation of row 5 hole 5 in the back of the bench on the northern side of the blast had most of its energy lost in the air because of a blow out. This was caused by the overburdening of this hole because of cracks in

that northern side of the bench. No other blow out holes were observed in this portion of the blast. The effect of effective stemming was observed in this section of the blast, where no stemming ejection but good swelling was observed.

Because of the overburdening of holes from row 24 onwards, caused by the failure in row 23, this portion of the blast was choked and produced a number of blow out holes. Stemming ejection on two holes of the blast occurred 1620 ms after the stemming ejection of the last hole on the blast. The late firing of the last two holes could be attributed to the late firing of the detonator. These holes were both in the back row of the bench. The specific location of these holes could not be established, nor could this occurrence be adequately explained although excessive confinement at the back row is thought of as being a possible cause.

In reference to Figure 5.8, it can be seen that the VODs are very low (less than 1000 m/s) whereas the characteristic VOD is within the range of 4500 to 5000 m/s. Such low VODs encountered during the third instance of blast monitoring justified critical investigation into the quality of the explosive (Emulsion E-6000, RD of 1.31) being used. The other key point is that no VODs were recorded for the top charges of holes 1, 3 and 4 owing to cable severance and/or misfires in the hole. Figure 5.9 is a VOD trace of hole 1.

What is of interest is the event in hole 2 in which the bottom primer initiated the explosives, but low order detonation occurred until the top primer fired after which proper detonation followed. This is evident from

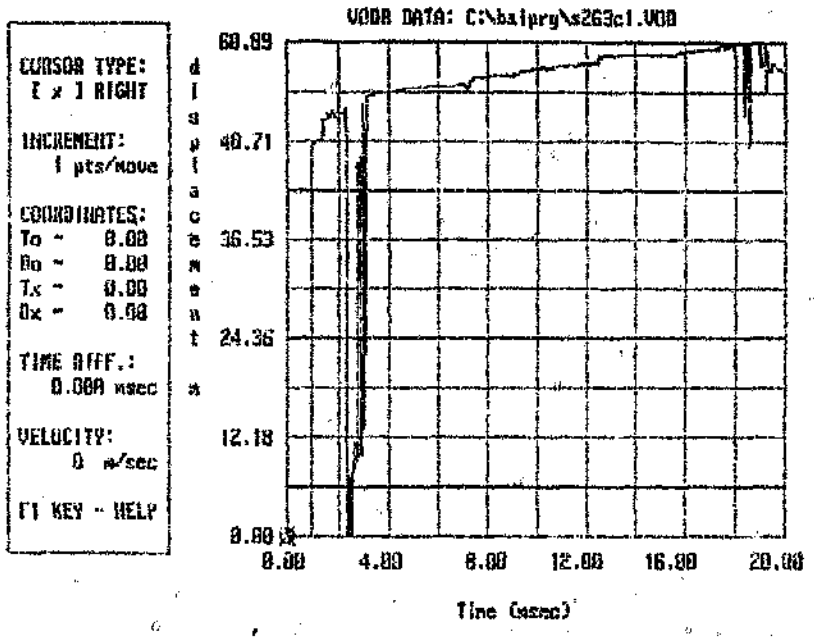


Figure 5.9: Signal Trace from VODR-1 showing bottom Deflagration and Top failure due to cable severance. No VODs were recorded for the top charges of holes 1, 3 and 4 owing to cable severance and/or misfires.

the VOD trace in Figure 5.10. Two possible reasons for this are given (Ladds *et al.*, 1991); the first is that the explosives in the primer region may have been above or close to critical density resulting in reduced explosive sensitivity. Alternatively, scatter in timing of the in-hole delays may have resulted in holes on either side of the hole in question firing slightly earlier and disrupting part of the explosive column thus inhibiting proper detonation. This is thought to be a likely scenario as the low order VOD appeared to remain constant for a long distance without notable increase as would be expected if densities were decreasing up the hole. The fact that 21 out of 24 explosive charges (87.5%) fired below the characteristic VOD and also that erratic detonation results were observed gave a good base

from which to analyze the results. Furthermore, as no meaningful conclusions could be drawn from the decking performance analysis and for the fact that other performance related problems had been encountered for the 3 blasts separately monitored, it was decided to monitor full column blasts (without decking) with the explosive gassed to a density of 1.15 or less. The next section discusses this monitoring programme.

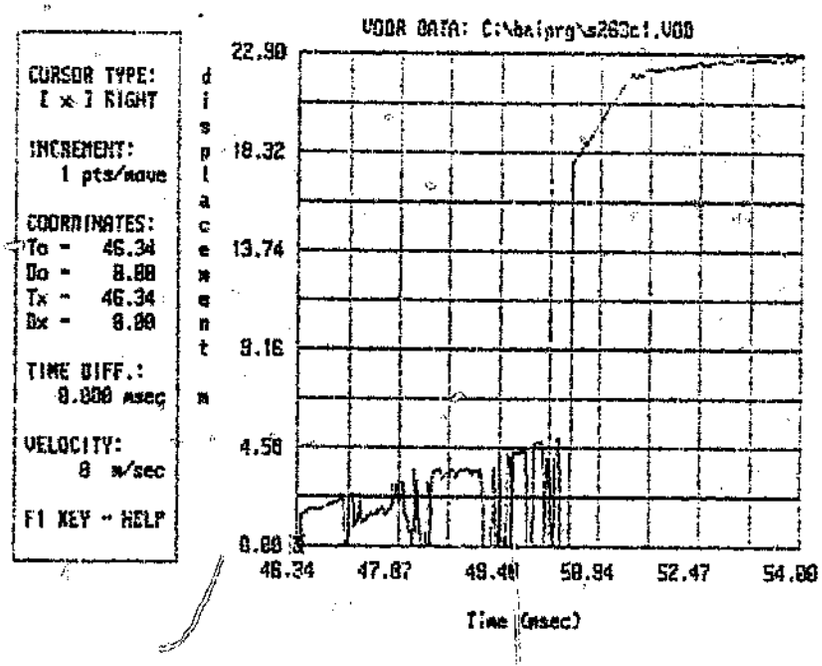


Figure 5.10 VOD Trace showing Bottom Charge Deflagration and Top Detonation

5.3 Blast Monitoring: Full Column Charge

The previous sections (5.1 and 5.2) detail the report of the VOD measurements of the Syferfontein Mine in decking trials. Table 5.1 on VODs certainly dramatise the fact that the explosive (Emulsion E-6000, density of 1.31) was

detonating below the characteristic VOD, a situation which in most instances gives rise to fragmentation problems. It was therefore decided to (a) gas the explosive to a density of 1.15 or less and (b) conduct a full column charge (without decking) and to monitor the performance of such a blast. These tests were done to characterise the products in the Syferfontein environment and to fine tune the blasting operation on the mine. Three blasts were monitored with full column E-6000 product. This section details the VOD results measured on these blasts.

5.3.1 Instrumentation

The VODs were measured using the VODR1 VOD recorder with FSJI-50A coaxial cable looped between the holes for the first blast monitored on 8/6/93 (Block 320678). The product that was used was in two different formulations; Emulsion E-6000 of density 1.31 in 6 holes and gassed emulsion E-6000 of density of 1.05 or less in 4 holes. Figure 5.11 shows the blast tie-in for the other two blasts monitored on 18/6/93 and 8/7/93 in which case the densities of the explosives were 1.12 and 1.15 respectively.

5.3.2 Discussion of Results

Table 5.4 shows the results of the VOD measurements in question. As is to be expected low VODs (below characteristic values) were recorded in all the holes charged with E-6000 density 1.31 for the blast monitored on 8/6/93 (Figure 5.12). On the contrary VODs of all holes charged with E-6000 (density 1.05) were fired within the characteristic range. Figures 5.13a and b give the raw data trace for two of these holes; one charged with E-6000 of density 1.31 and the other two with E-6000 of density 1.05 or less. This reinforces the

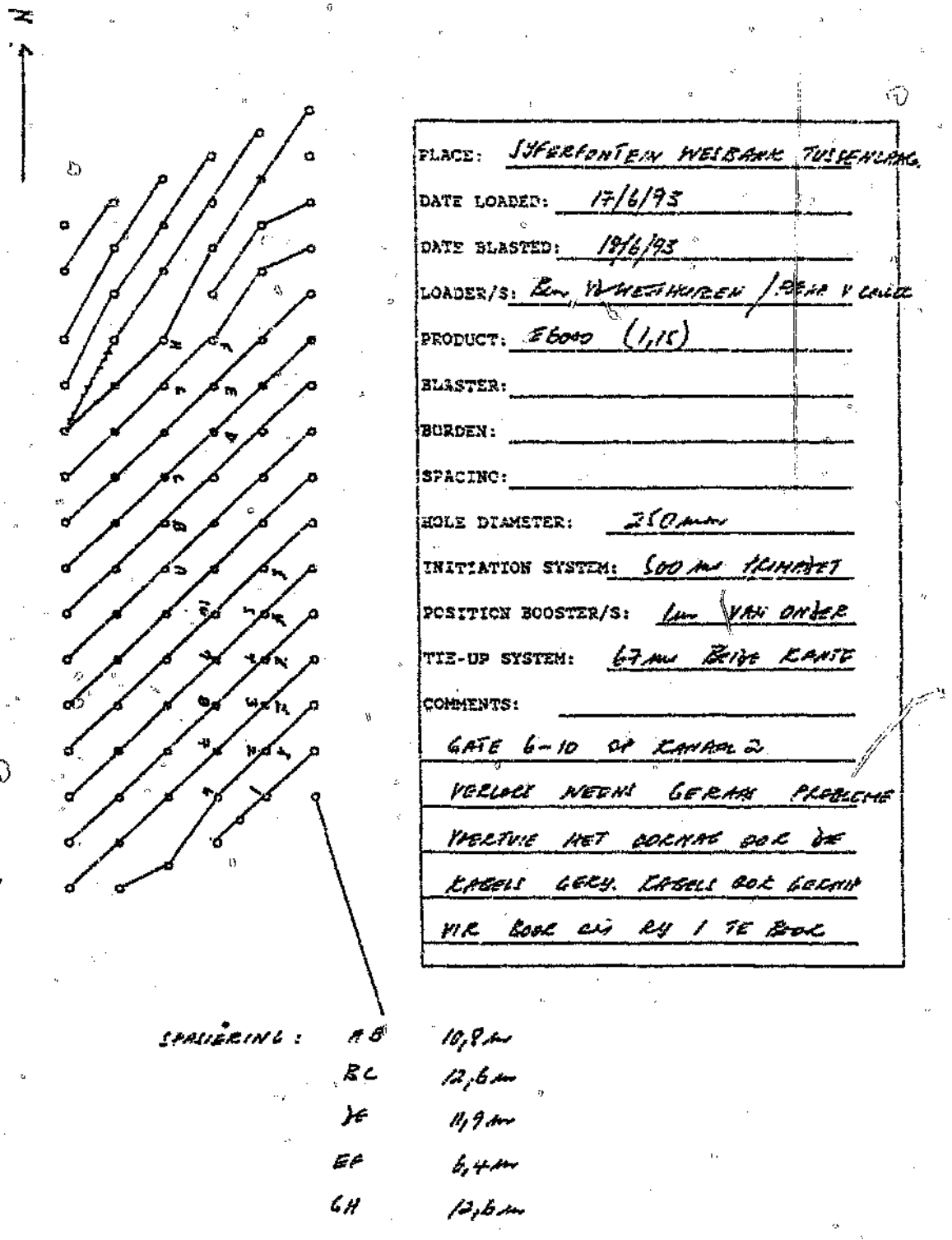


Figure 5.11 Blast Tie-in for VOD Measurement for Full column Charge (Explosive Density 1.15 or less)

BLAST DATE (1)	BLOCK NUMBER (2)	EXPLOSIVE DENSITY (3)	HOLE NUMBER (4)	MEASURED VOD (m/s) (5)	CHAR. VOD (m/s) (6)
8/6/93	320678	1.31	1	715	4500 - 5000
		1.31	3	451	4500 - 5000
		1.31	4	488	4500 - 5000
		1.31	6	923	4500 - 5000
		1.31	8	540	4500 - 5000
		1.31	9	560	4500 - 5000
		1.05	2	5351	4700 - 6000
		1.03	5	5116	4700 - 6000
		1.04	7	5359	4700 - 6000
		1.05	10	5061	4700 - 6000
18/6/93	320665	1.12	1	5404	4700 - 6000
		1.12	2	5304	4700 - 6000
		1.12	3	5737	4700 - 6000
		1.12	5	704	4700 - 6000
18/6/93	320621	1.15	1	-	4700 - 6000
		1.31	2	4825	4700 - 6000
		1.15	3	5212	4700 - 6000
		1.15	4	4851	4700 - 6000
		1.11	5	5642	4700 - 6000
		1.1	6	5550	4700 - 6000
		1.15	7	5305	4700 - 6000
		1.15	8	5575	4700 - 6000

Table 5.4 VOD Measurement: Full Column Charge

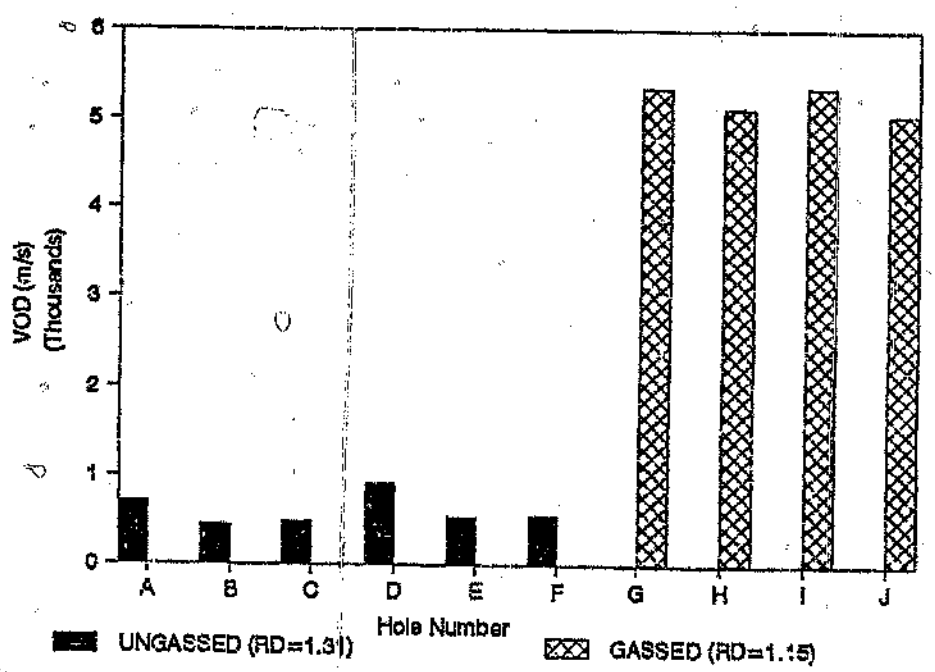


Figure 5.12 Bar Chart Showing VODs of Emulsion E-6000 - Ungassed (RD=1.31) & Gassed (RD=1.15) -

conclusion that the use of E-6000 explosive of density 1.31 is inappropriate and should be discontinued. Emulsion E-6000 of densities 1.12 to 1.15 were also experimented in two separate blasts (Block 320665 and 320621) and in all except two cases, detonated at characteristic VOD (see Figure 13). At characteristic VOD, detonation is at its most stable. Thus the density dependency factor of emulsion E-6000 is evident. As mentioned earlier E-6000 is a water resistant product consisting of 60% Matrix or Emulsion and 40% ANFO. Earlier investigation by Brinkman (1990) reveals that the large particle sizes of the constituents in ANFO lead to a majority of the energy being released in the secondary reaction zone which takes place behind the primary detonation front. On the other hand, emulsions contain mainly finely divided constituents. The greater surface area of the reactants leads to rapid combustion with most of the energy being released as the detonation front moves through the explosive charge and less energy being released in the secondary reaction zone behind the detonation front. This accounts for the high VOD of Emulsion. The combination of Emulsion and ANFO (heavy ANFO) such as E-6000 giving some degree of heave and shock energy has been found to perform well in the Syferfontein environment as opposed to bulk Emulsion (100% Emulsion). Delay performance is evaluated in the next section.

5.4 Delay Assessment

The performance of the initiation system in terms of timing accuracy and reliability can affect overall blast results. Tables 5.5a and 5.5b are extracts from Table 5.1 and 5.4 respectively which give the VOD measurements and firing times discussed earlier. In particular the firing times are deduced from the VOD traces (see Figures 5.4, 5.9 and 5.10). Shown in Table 5.5a is an assessment of

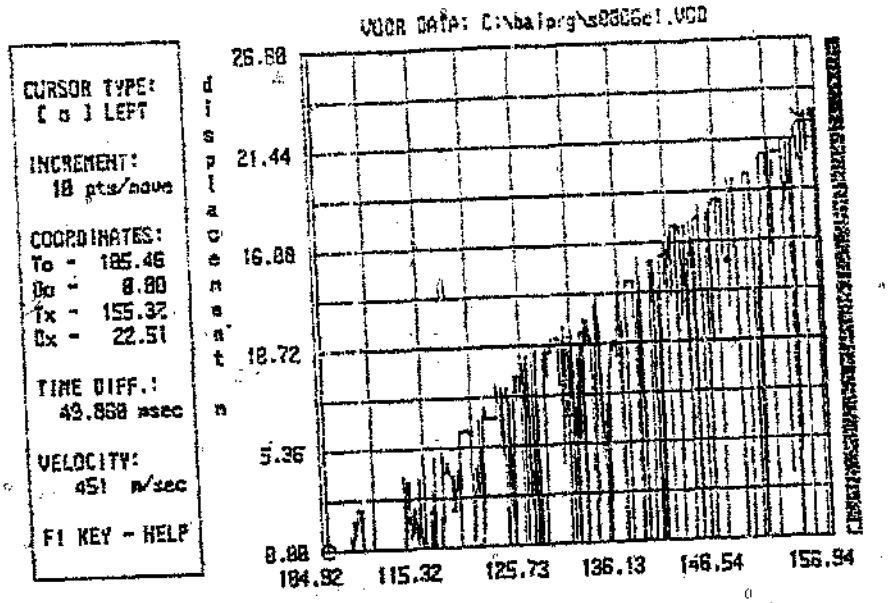


Figure 5.13(a) VOD Trace For E-6000 of Density 1.31 (VOD Below Characteristic Range)

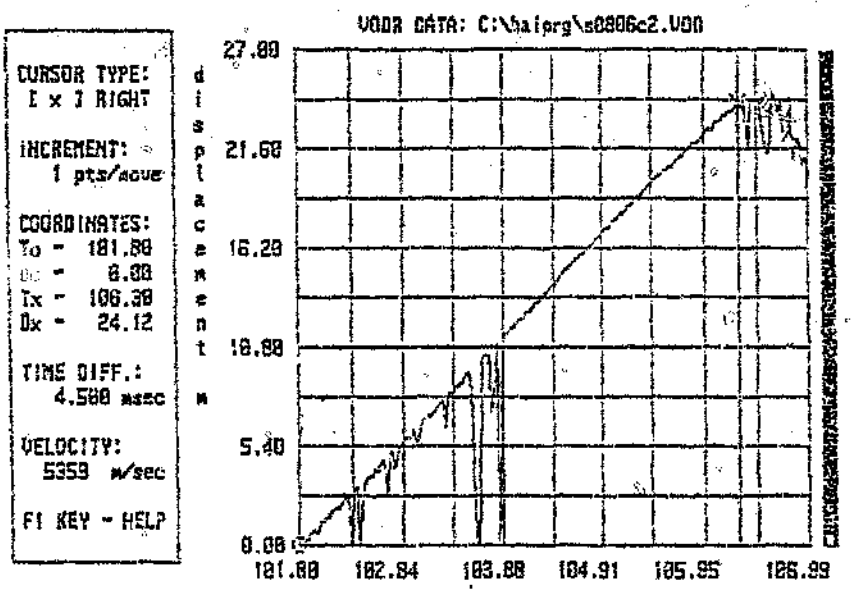


Figure 5.13b VOD Trace of E-600 assed to density 1.04 - (VOD within Characteristic Range) -

the in-hole delay interval between top (500 ms) and bottom (525 ms) charges. The expected delay interval is 25 ms with an allowable scatter of 4 ms according to the manufacturers specification. Discounting the effect of this allowance the last column gives the deviation from the expected value of 25 ms. A close look at the table reveals that 7 out of 9 holes (77.78%) recorded a distinct deviation from the expected value.

Statistically the probability density function derived from the normal distribution curve can be used to determine the probability of a charge firing outside allowable scatter time. The Probability Density function, Z, is such that

$$Z = (X_A - M)/S \qquad (12)$$

where

X_A = Allowable deviation

M = Mean deviation

S = Standard deviation

From Table 5.5a, $X_A = 4$, $M = 9.36$, $S = 9.71$ giving

$$Z = (4 - 9.36)/9.71 = 0.6153$$

From the Normal distribution table, the area under the curve (Figure 5.14) between Z = 0 and Z = 0.6153 is given as 0.2324.

Therefore the probability of the allowable deviation (X_A) exceeding 4 is represented by the area to the right of Z = 0.6153, i.e.

$$P(x) > 4 = 0.2324 + 0.5 = 0.7324 \text{ or } 73.24\%.$$

This indicates that the 25ms delay interval stands a significant risk of deviation, a situation which usually culminates in desensitisation of the top charge by the bottom charge, or desensitisation of the explosives

Table 5.5a In-hole delay assessment

Blast Date : 10/03/973 Explosive density : 1:31
Block Number : 320575 Delay Time : 500ms Top & 525ms Bottom

HOLE NUMBER	FIRING TIME (ms)		DELAY INTERVAL		DEVIATION	DISCOUNTED DEVIATION
	BOTTOM CHARGE	TOP CHARGE	ACTUAL	EXPECTED		
1	2.3	2.3	0	25	25	21
2	60.42	77.18	16.76	25	8.24	4.24
3	117.98	139	21.02	25	3.98	-
4	151.68	173.47	21.79	25	3.21	-
5	198.45	210.93	12.48	25	12.52	8.52
6	77.93	94.25	16.32	25	8.68	4.68
7	135.6	130.92	*	25	*	*
8	176.02	176.06	0.02	25	24.98	20.98
9	221.5	227.06	5.56	25	19.44	15.44

* Top charge, contrary to expectation, fired earlier than bottom charge

Table 5.5b: Surface Delay Assessment

BLOCK NUMBER	DELAY POSITION	DELAY (ms)		DEVIATION (ms)	ALLOWABLE SCATTER	DISCOUNTED DEVIATION
		ACTUAL	EXPECTED			
320678	Row 1 & 2	62.05	67	4.95	5	-
	Row 2 & 3	41.42	67	25.58	5	20.58
	Row 3 & 4	135.4	67	69.4	5	65.4
	Row 4 & 5	11.14	67	55.86	5	50.86
	Row 5 & 6	87.49	67	20.49	5	15.49
	Holes 2 & 6	12.21	0	12.21	0	5
	Holes 3 & 7	3.12	0	3.12	0	-
	Holes 4 & 8	67.73	67	0.73	5	-
320665	Holes 1 & 2	60.61	67	6.39	5	1.39
	Holes 2 & 3	67.09	67	0.09	5	-
	Holes 3 & 5	123.68	120	3.68	10	-
320621	Row 2 & 3	93.52	84	9.52	8	1.52
	Row 3 & 4	92.44	84	8.44	8	0.44
	Row 4 & 5	119.34	84	35.34	8	27.34
	Row 5 & 6	42.12	42	0.12	4	-
	Holes 5 & 7	3.12	0	3.12	0	-
	Holes 6 & 8	3.98	0	3.98	0	-

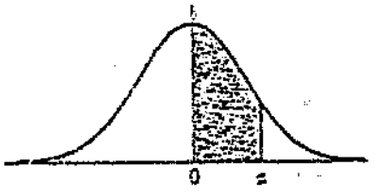


Fig 5.14 Normal distribution curve

column in the adjacent holes. The top charge firing before the bottom charge for all the 5 holes monitored for VOD on 18/3/93 (Table 5.1a) could be attributable to either timing problems where the top 525ms units were firing before the bottom 500ms units or frequent failure of the bottom units.

Surface delays between rows and holes are assessed in Table 5.5b. Delay intervals of 42ms, 67ms and 84ms have been used in different cases with 67ms being the most frequent. For the 67ms delay, 4 out of 8 cases (50%) showed a distinct deviation from the expected value whereas 2 out of 3 cases indicated successful performance for the 84ms delay. Chiapetta (1983), through a series of high speed photographic studies, has shown that the criterion of allowing 1 to 2ms per foot of burden in most multiple blasting operations is insufficient and usually results in a tight muckpile. He rather recommends 3ms per foot (9.8ms per meter) of burden. Hence with the burden of 10.5m used on the mine, a more appropriate surface delay would be 100ms. However

with the suggested new drill pattern (8.5 x 9.5m), a delay of 8.5 x 9.84 or 84ms would be appropriate. This new criterion will reduce the tight muckpile characteristic of causing hard digging, low productivity of the dragline, increased wear and tear of dragline teeth and backbreak.

5.5 Summary of results

The use of velocity of detonation monitoring and high speed photography in a rapid and cost effectively identifying blast performance related problems are thus illustrated. The advantages of backing up VOD measurements with high speed photography are also explained. Three blasts on decked holes and three others on full column charge have been monitored and the results indicated the following:

(i) Unsatisfactory performance of explosive (E-6000) of relative density 1.31 as in 39 out of 42 cases it detonated below the characteristic VOD. This situation unequivocally led to fragmentation problems in certain blasts. Explosive was gassed to density of 1.15 or less and three blasts (16 holes) were monitored. All the holes except two fired within the characteristic range.

(ii) VOD measurements uncovered some performance related problems such as variable detonation results, top primer firing before bottom primer although timed to initiate the reverse, misfires etc. These scenarios have been summarised in Table 5.6. The low VODs and apparent bottom charge failure

accounted for the boulders and difficult digging observed in certain areas of the blast.

(iii) No meaningful conclusions could be drawn regarding the performance of the various decking materials as virtually all the charges fired at VODs considerably lower than the characteristic values.

(iv) The results from high speed photography also indicated that

- out-of-sequence firing occurred in three rows out of eight. Also Row 23 did not fire. The effect was overburdening of Row 24 onwards, the blast was choked and more blowouts observed. This accounted for a completely unbroken section or the 'Table top effect' on that blast.

- The performance of the 42 ms Nonel type detonators (NTDs) should be investigated further as a delay range of 10 ms places a question mark on its accuracy.

- The best way of utilising the air bags for decking should be investigated further, since indications are that slipping of the bags does occur.

(v) Assessment of in-hole delay has revealed unsatisfactory performance of the 25ms delay interval i.e 525ms and 500ms delays for top and bottom charges respectively as 7 out of 9 holes (77.78%) recorded distinct deviation from the allowable or expected value. This assures 73.24%

confidence of the deviation exceeding the allowable scatter. Excessive scatter usually culminates in desensitisation of the top charge by the bottom charge or desensitisation of explosive column in the adjacent holes and is bound to have negative impact on blasting performance.

Table 5.6 Summary of Results of Blasts Monitored for VODs

DATE	BLAST	NUMBER OF HOLES	EXPLOSIVE DENSITY	ACTUAL V O D MEASURED (m/s)	MEAN VOD MEASURED	CHAR. VOD(m/s)	REMARK
10/3/93	320575	9	1.31	T = 619 – 4565 B = 850 – 3788	1480 1429	4500 – 5000	(a) Objective: Deck performance verification (b) Low and erratic VODs recorded. (c) No VODs reCorded for some holes (d) Only one charge detonated at characteristic VOD.
18/3/93	320543	5	1.31	798 – 2708	1753	4500 – 5000	(a) Low VODs but good dragline performance (b) No VODS recorded for some holes (c) Performance related problem includes – Sympathetic detonation of top charge with bottom charge – Cable severance prior to detonation – Bottom deflagration and top failure.
26/3/93	320531	4	1.31	T = 4505 B = 689 – 912	4505 779	4500 – 5000	(a) Only one hole detonated at the top and low VODs at the bottom (b) Erratic detonation results and initiation related problems – Inteferance with cable – Frequent top charge failure – Bottom deflagration and top detonation.
8/6/93	320678	6 4	1.31 1.04	461 – 923 5061 – 5351	621 5221	4500 – 5000 4700 – 5500	(a) Two explosive types (RD of 1.31 & 1.15) tested. (b) Low VOD below char. value for 1.31 RD explosive (c) Acceptable VOD within char. range for 1.04 RD explosive (d) Poor blast results (poor fragmentation) partly attributable to 1.31 RD explosive.
18/6/93	320665	4	1.12	5404 – 5737	5404 – 5737	4700 – 5500	(a) All holes (except one fired at 704) detonated within the char. range (b) Good looking fragmentation. (c) Full column charge
8/7/93	320621	8	1.15	4851 – 5575	5308	4700 – 5500	(a) Full column blasting for test blastholes (b) All test holes fired within characteristic VOD (c) The need to review density of explosive was justified.

CHAPTER 6

DRAGLINE PRODUCTIVITY

The productivity of a dragline can be assessed through several indices. For the purpose of this study, fill factor (bucket factor), fill time or digging time and bank cubic meter per production hour (BCM/Pr.Hr) are the main diagnostic tools used in this chapter for evaluating dragline performance. This is because these are the real indices that reflect blasting performance which to all intents and purposes form the core or basis of this project. Crosby (1981) has produced much evidence to support this view. This chapter therefore evaluates dragline performance on the basis of these indices and relates dragline productivity to blasting performance. The influence of certain key blast design parameters such as drill pattern, operators performance etc on dragline productivity is also assessed. Dragline scheduling is briefly discussed. Table 6.1 lists the values of these indices for the various blasts under study. The blast giving the best results in terms of dragline performance from the combined effects of all the various indices will be illustrated later in this section.

6.1 Statistical Analysis of Dragline Productivity

In the subsequent sections, dragline performance (measured through the various indices) is assessed based on the results of 12 blasts. For samples of size N>30 called **large samples** the sampling distribution is

Table 6.1 DRAGLINE OPERATIONAL DATA

BLAST DATE (1)	BLOCK NUMBER. (2)	DIGGING PERIOD (3)	MECH. AVAIL. (%) (4)	% UTIL. (5)	VOLUME INSITU (BCM) (6)	PROD. HOURS (7)	BCM/Pr.Hr. (6)/(7)=8	TIMING CYCLE TIME (s) (9)	TIMING FILL TIME (s) (10)	FILL FACTOR (%) (11)	COST/BCM (R/BCM) (12)
23/3/93	3205109	23/3/93 - 13/3/93	82	78	396130	270.76	1463.03	65	20.11	74.05	0.826
26/2/93	320587	13/3/93 - 23/3/93	86	85	406620	188.06	2162.18	60.6	17	74.05	0.59
03/3/93	320576	23/3/93 - 30/3/93	89	84	257580	140.82	1929.14	62.5	18.2	74.05	0.661
10/3/93	320575	30/3/93 - 08/4/93	80	82	308226	154.46	1995.51	60.8	17.6	72.19	0.606
18/3/93	320543	08/4/93 - 15/4/93	80	82	241205	122.39	1970.79	58	17	75.3	0.613
25/3/93	320531	15/4/93 - 03/5/93	86	83	508754	314.37	1618.32	62.6	18.93	66.41	0.747
30/3/93	320501	03/5/93 - 08/5/93	89	76	185002	69.67	2655.4	65.4	19.74	72.11	0.444
01/6/93	3206108	04/6/93 - 22/6/93	86	80	542700	301.29	1801.25	68	20	71.23	0.671
08/6/93	320678	22/6/93 - 29/6/93	87	72	194400	110.77	1754.98	64.6	18.71	63.76	0.688
11/6/93	320676	29/6/93 - 08/7/93	87	76	364230	152.88	2382.46	63.4	19.46	65.72	0.507
18/6/93	320665	08/7/93 - 19/7/93	90	75	276750	175.61	1575.94	61	17.86	63.94	0.767
29/6/93	320643	19/7/93 - 02/8/93	87	83	453486	237.42	1910.05	60.8	18	62.56	0.633
AVG			85.75	79.67	344590.25	186.54	1926.59	62.73	18.55	69.61	0.65
STD			3.22	3.99	114438.75	74.77	327.74	2.60	1.06	4.55	0.10

approximately normal, the approximation becoming better with increasing N (Fisher, 1938). For samples of size N<30, called **small samples**, such as 12 blasts in question, this approximation is poor and becomes worse with decreasing N. As a result the 'small sampling theory' is used in the analysis that follows.

Using the student's t distribution (Figure 6.1), if $-t_{0,975}$ and $+t_{0,975}$ are the values of t for which 2.5% of the area lies in each tail of the t-distribution the 95% confidence interval for t is

$$- t_{0,975} < (x - u)/s \sqrt{N - 1} < t_{0,975} \qquad (13)$$

where

X	=	sample mean
μ	=	population mean
S	=	sample standard deviation
N	=	number of samples
V_t	=	N-1 (v_t = degrees of freedom)

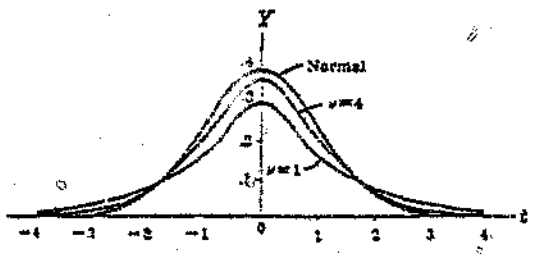


Fig 6.1 Students t-distribution for various values of v

From the above equation, the population mean (u) can be estimated to lie in the interval

$$X - t_{0.975}S(N-1) < u < X + t_{0.975}S(N-1) \qquad (14)$$

With reference to Table 6.2 the following information can be deduced for the fill factor:

Mean (X) = 69.25%

Standard deviation = 4.55%

Number of samples000 (N) = 12

Number of degrees of freedom = v - N

= 12-1 = 11

$t_{0.975}$ = 2.2 (from tables)

Table 6.2: Mean estimates for the various performance indices

Parameter	Mean	Standard Deviation	Interval (μ)
Fill Factor (%)	69.62	4.55	$68.34 < \mu < 70.16$
Fill time (S)	18.73	1.14	$18.50 < \mu < 18.96$
BCM/Pr.Hr.	1926.24	372.22	$1848.80 < \mu < 1.12$
Uniformity index	1.10	0.1	$1.08 < \mu < 1.12$

Applying equation (14), the population mean (μ) lies in the interval

$$69.25 - 2.2 \times \frac{4.55}{12-1} < \mu < 69.25 + 2.2 \times \frac{4.55}{12-1}$$

$$68.34 < \mu < 70.16$$

Although 12 samples are used, the above calculation assures 95% confidence that the population mean (fill factor values for larger samples, N>30) will lie between the intervals 68.34% and 70.16%. By

similar analysis, the tabulation below indicates the possible mean interval for the other productivity indices based on 95% confidence limits (i.e. probability of 0.95).

6.2 Dragline Productivity Evaluation

This section evaluates the dragline performance on the basis of the productivity indices mentioned above, viz, fill factor, fill time and bank cubic meters per production hour (BCM/Pr.Hr.).

6.2.1 Fill factor

Fill factor is the ratio of loose cubic meters versus bucket rating in cubic meters. This parameter is dependent on the material heaping qualities, the angle of repose and the consistency with which the operator fills the bucket. Operators' performance is evaluated in section 6.6. Generally speaking the fill factor value usually varies from 0.4 to 1.1 and depends on the ease with which materials flow into the bucket. It is the most important productivity index that reflects blasting performance and as a consequence much emphasis has been placed on it not only in comparing the various blasts (Table 6.3) but also as a key target of improving dragline productivity. The fill factor, as can be deduced from Table 6.3, varies from 74.05% to a minimum of 62.56% with an average of 69.25%. Chiapetta (1993) points out that fill factor should practically be between 85% to 95% for efficient dragline operation. Bucyrus-Erie, 1976 throws more light on this argument by stating that although it is theoretically possible to have a bucket fill factor greater than 100% the usual values are in the 80% to 95% range. In the light of the above, the average value of 69.25% as deduced from Table 6.3 is apparently low and implies either difficulty in digging or excessive fines production.

Table 6.3 Summary of Results: Dragline Productivity and Fragmentation Assessment

BLAST	PRODUCTIVITY INDICES			FRAGMENTATION		COST	REMARKS FROM VOD MEASUREMENT
	FILL TIME (s)	FILL FACTOR (%)	BCM / H	UNIFORM INDEX(n)	OVERSIZE (%)	TOTAL (R/BCM)	
3205109	20.11	74.05	1463.03	1.06	12	1.315	
320587	17.2	74.05	2162.06	1.13	12	1.011	– Good fragmentation
320576	16.2	74.05	1829.14	1.12	10	1.162	– Relatively easy digging
320575	17.6	72.19	1995.51	1.01	15	1.064	– VOD below characteristic values
320543	17	75.3	1970.79	1.03	17	1.026	
320531	18.93	66.41	1618.32	1.03	12	1.159	– Unfragmented sections
320501	19.74	72.11	2655.4	1.14	17	0.876	– Difficult digging
3206108	20	71.23	1801.25	1.02	14	1.096	– VOD below characteristic values
320678	18.71	83.76	1754.98	1.01	13	1.156	
320676	19.46	85.72	2382.46	1.04	17	0.906	– Good fragmentation
320685	17.86	83.97	1575.94		8	1.232	– Difficult digging
320643	18	82.56	1910.05	.28	8	1.167	– But VOD at characteristic range
SUM	222.81	835.40	23118.95	13.16	155.00	13.171	
AVG	18.57	69.62	1926.58	1.10	12.92	1.098	
STD	1.04	4.55	327.74	0.10	3.09	0.123	

Figure 6.2 is a diagrammatic illustration of the chronological pattern of the fill factor for the study period (23/3/93 - 29/6/93). A decreasing trend of fill

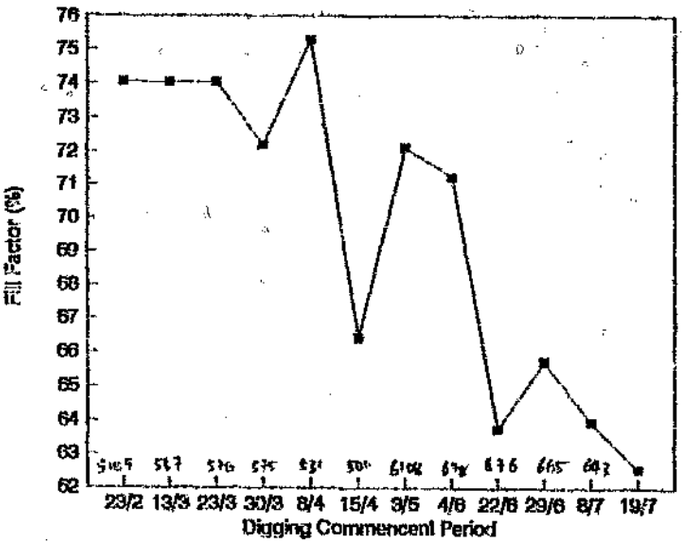


Figure 6.2 Diagrammatic Illustration of a Declining Trend of Fill Factor (23/2/93 - 2/8/93)

factor with time is again visible. Given the high production cost of the dragline operation, the mine simply can not afford to tolerate such an inimical situation, i.e. low productivity of the dragline. The need to immediately review the blasting operation to assist the dragline diggability can therefore not be overemphasised.

6.2.2 Fill time

The dragline cycle time refers to the length of time taken between commencing to dump one bucket load to commencing to dump subsequent load. In other words the time for one complete operation consists of digging, swinging to dump, dumping time and swing back to load. The time should also include all the unrecorded delays inherent in the operation including all the inefficiencies in motors, operators, occasional poor digging etc.

The digging time (fill time) is the aspect of the cycle time which is mainly related to fragmentation and which is used as a productivity index in this study. Table 6.4 is an extract from Table 6.1 and it shows the cycle time, fill time and the fill time values expressed as a percentage of cycle time. From the table, fill time values range from 17s to 20.7s with a mean value of 18.72s. This corresponds to 28.05% to 30.94% of the cycle time with a mean value of 30%. Crosby *et al.* (1981) found that fill time for a good digging operation is typically 25% of the cycle time although this depends to some extent on the swing angle. An average value of 30% appears unsatisfactory and is a manifestation of tight digging conditions of the muckpile which could result from a number of design factors including the excessive spacing and burden discussed in Chapter four. Two dragline bucket lengths should fill the bucket if

Table 6.4 Fill Time Expressed as Percentage of the Cycle Time

BLAST DATE	BLOCK NUMBER	TIMING		FILL TIME AS % OF CYCLE TIME
		CYCLE TIME (s)	FILL TIME (s)	
23/3/93	3205109	65	20.11	30.94
26/2/93	320587	60.6	17	28.05
03/3/93	320576	62.5	18.2	29.12
10/3/93	320575	60.8	17.6	28.95
18/3/93	320543	58	17	29.31
25/3/93	320531	62.6	18.93	30.24
30/3/93	320501	65.4	19.74	30.18
01/6/93	3206108	68	20	29.41
08/6/93	320673	64.6	18.71	28.96
11/6/93	320676	63.4	19.46	30.69
18/6/93	320665	61	17.86	29.61
29/6/93	320643	60.8	18	
	AVG	62.73	18.55	29.59
	STD	3.60	1.06	0.82

fragmentation is good and muckpile is sufficiently loose (Chiapetta, 1993). Ideally, the muckpile profile and displacement should compliment the digging equipment. To achieve this, there should be a trade-off between higher productivity and closer drilling patterns which necessitate higher energy input (more explosive consumption).

6.2.3 Bank cubic meter per production hour (BCM/Pr.Hr)

Bank cubic meter refers to the in situ volume of the block or bench. Production hour represents the time when the dragline is physically digging. Essentially it excludes null element times not directly tied to fragment size such as time for preparation, walking and delays. Furthermore the effect of mechanical availability and utilisation is discounted. Used as a productivity index, bank cubic meter per production hour gives a volumetric measure of the digging capability of the dragline shovel.

The measured values for the blasts in question are again shown in Table 6.3. The values range from 1755 to 2655 BCM/Pr.Hr with a mean value of 1926. Theoretically the maximum value for the dragline in question is 3208 BCM/Pr.Hr. It is therefore evident from the available data that the dragline is performing far below expectation and there is room for improvement if related drilling and blasting problems are corrected. The results of the dragline performance evaluation is summarised in Table 6.4a.

6.3 Dragline Performance Related to Blasting

The previous chapters have dug into a critical assessment of the existing practice (design consideration) on the mine as well as the causes of fragmentation problems through VOD measurements and high speed photography. The salient points are recapitulated in this section and for the argument of brevity and clarity are presented in tabular form (Table 6.4a). On the question of fragmentation 69% of the fragments were in excess of 80cm as surfaced by the model, this justifying a review of the blasting operation. virtually all the blasts analysed showed a uniformity exponent of the Rosin-Rammler model greater than 1 which is the optimum requirement for optimum fragmentation (Cunningham, 1993). The foregoing analysis reveals that the dragline productivity assessed through the various indices, viz, fill time, fill factor and BCM/Pr.Hr, is far below expectation. Furthermore the decreasing trend of fill factor during the period under investigation (Figure 6.1) leaves much to be desired. These issues will be addressed in due course.

The VOD measurements (Chapter 5) have placed in better perspective some initiation related problems such as poor in-hole delay performance, top primer initiation before bottom primer (although timed to initiate the reverse)

Table 6.4c Summary of Results: Performance Evaluation

PAPAMETER	INDEX	VALUE	REMARK
PRODUCTIVITY	1. Fill Time	Range: 17–20.7s Average: 18.72s 30% of cycle time	(a) Fill time for a good digging operation is typically 25% of total cycle time. Average value of 30% is quite unsatisfactory. (b) Difficult digging condition is manifested. (c) Two dragline lengths should fill the bucket if fragmentation is good and muckpile is sufficiently loose (Chiappeta, 1993).
	2. Fill factor (Bucket factor)	Range: 52.6–74% Average: 63%	(a) Most important index that improves dragline productivity. (b) Fill factor values for satisfactory digging conditions are: Theoritically: 40 – 110% Practically: 85 – 95% (Chiappeta) (c) Average value of 63% is quite low. Again difficult digging condition or tight muckpile is manifested.
	3. BCM / Pr. Hr	Range: 1755–2655 Average: 1926	(a) There is room for improvement if related drilling and blasting problems are corrected and blast displacement induced.
FRAGMENTATION	4. Uniformity	Range: 1.01 – 1.31	(a) Uniformity index of 1 or more portrays good fragmentation.
	5. % Oversize	> 150cm: 5% > 80cm: 69%	(b) For dragline consideration, no oversize per se. (c) However from design perpective, blast continously resulting in boulders greater than 80cm should be reviewed (Cunningham ,1993). * Generally fragmentation appear not to pose problem on the property. However the 'Table Top Effect' calls for alarm.
COST		% OUT OF TOTAL COST	
	6. Drilling	Range: 5.08–6.91% Average: 6%	(a) Drilling cost is insignificant.
	7. Blasting	Range: 30.3–42.4% Average: 36%	(b) Any change that may result in increased drilling and blasting cost but which can possibly enhance dragline productivity will drastically reduce overall cost.
	8. Dragline	Range: 50.7–62.83% Average: 58%	

and cable severance all of which have drastic influence on blasting performance (refer to Table 6.5). Explosives firing below the characteristic values and explosive incompatibility with the blasting environment are also evident. The high speed photography is used as a valuable diagnostic field tool to complement VOD measurements. Assessment of the existing practice from the design perspective (Chapter 4) has indicated lack of field control in many areas including spacing burden ratio, delay selection and loading.

In relating dragline performance to blasting, the major design factors influencing dragline productivity are discussed; reasons for poor fragmentation and hence low dragline digging efficiencies as uncovered through velocity of detonation measurement and high speed photography are explained in the subsequent sections. Other external factors impacting on dragline productivity such as operators' performance are also evaluated to eliminate the experience factor associated with different operators.

6.3.1 Design factors related to fragmentation and productivity

There are a number of factors affecting fragmentation of rock which the blaster must consider when deciding on a specific blast design. These factors are categorised under uncontrollable (such as geological factors) and controllable (explosive and design) factors. It is the latter case which is within the control of the blaster (Paul, 1993) and must be adjusted to achieve optimum fragmentation and improve dragline productivity. In Table 6.6, the dragline productivity using the various indices, viz. fill time, fill factor and BCM/Pr.Hr has been studied under various conditions of decking, spacing-burden ratio, delay and powder factor. These

Table 6.5 RESULTS OF VOD MEASUREMENTS: Emulsion E–6000 (RD=1.31)

No of Holes monitored = 25, Range of VOD measured = 4491m/s

No of Holes	% of Total	Event	Remarks (causes)
1	5.5	Sympathetic detonation of top deck with bottom deck	–
6	33.3	Inteference with the cable	–
5	27.8	Top primer initiated before the bottom primer although timed to initiate the reverse.	(a) Frequent failure of the bottom units or (b) Timing problem where the top 525ms units are firing before the bottom 500ms units.
2	11.11	Deflagration of the bottom primer followed by detonation of the top primer.	(a) Explosive at the bottom primer region above or close to the critical density resulting in reduced explosive sensitivity. (b) Scatter in timing of the primer region results in earlier firing, thus disrupting part of the explosive column and inhibit proper detonation.
5	11.11	Deflagration of bottom charge without recovery into detonation i.e top failure.	(a) Possible desensitisation of top charge by bottom charge or adjacent charge due to excessive time (delay) scatter or (b) Performance problem of the explosive.
16	88.8	Firing below the characteristic VOD. In most cases, run up did not occur and detonation remained linear.	(a) Indicates performance problem or explosive quality problem. (b) Explosive incompatibility with the blasting environment manifested. * The need to investigate into the explosive quality was thus justified. * Explosive gassed to 1.15 and 3 blasts (16 holes) monitored. RESULTS: All except one hole detonated at characteristic VOD. CONCLUSION: Confirms problem not with initiation but rather with the explosive (Emulsion E–6000, RD = 1.31).

Table 6.6 Design Factors Related to Dragline Productivity and Fragmentation

PARAMETER	BLOCK	MATERIAL (VALUE)	PRODUCTIVITY (AVERAGE)			FRAGMENTATION	REMARKS
			FILL TIME(s)	FILL FACTOR(%)	BCM/Hr	UNIFORMITY INDEX(n)	
1 Decking	5109-543 531-676 665-622	ash Air Full col.	18.4 19.4 18.4	73.93 67.85 63.79	1884.1 2042.5 1697.8	1.07 1.05 1.29	(a) Ash performs relatively better. (b) With air deck gas move quickly within 1-2ms from bottom to top deck, desensitising the product thus causing sympathetic detonation. Explosive deflagrates, poor breakage results. (c) Ash deck slows down such gas movement to 5-15ms. (d) Full column charge has not yielded the expected productivity. Its probable that more explosives than necessary has been used to compensate for some blasting inadequacies.
2. S X B	5109-575 543-501 6108-665 643-622	10 x 10.2 11 x 10.2 11 x 10.5 10.5x10.5	18.3 18.4 19 19.3	73.58 71.03 66.17 63.71	1862.4 2081.5 1878.7 1758.8	1.08 1.07 1.1 1.3	(a) S X B = 10 x 10.2m gives relatively the best results. (b) Gives an indication of the possibility of getting better results by further reducing the drill pattern.
3 In-hole delay	5109-543 531-622	525&500ms 500&500ms	18.4 19.17	73.93 66.34	1884.1 1913.2	1.07 1.14	Further investigation using VOD trace necessary Refer to section 5.5.
4 Powder Factor (kg/m3)	5109-543 531-676 665-622	0.448 0.456 0.483	18.4 19.4 18.4	73.93 67.85 63.79	1884.1 2042.5 1697.8	1.07 1.05 1.29	(a) Easy digging with 0.448 (b) Medium digging with 0.456 (c) Difficult digging with 0.483 Increased powder factor did not necessarily improve dragline performance. The effect of increased powder factor was negated by using larger drill pattern .i.e S x B ratio of 11 x 10.5m. Crater due to one hole may not overlap the next hole thus resulting in toes.

design factors have already been covered in Chapter 4 and are reproduced here in summarised form on account of their relevance to the present subject. The influence of the various factors on dragline productivity is discussed below. It must nonetheless be emphasised that these factors only offer partial explanation to the dragline performance. other factors come into play although their influence are not quantified or isolated in this report.

Effect of Decking on dragline productivity

From Chapter 5, it was evident that the results of the deck performance trials are inconclusive as a substantial number of explosive charges fired below characteristic VOD ranges. The investigation that follows therefore arises from the fallacies or inadequacies of the earlier studies. Table 6.6 shows dragline productivities for the different decking materials, viz, air, ash and full column change. These indices expressed as percentages are depicted on Figure 6.3.

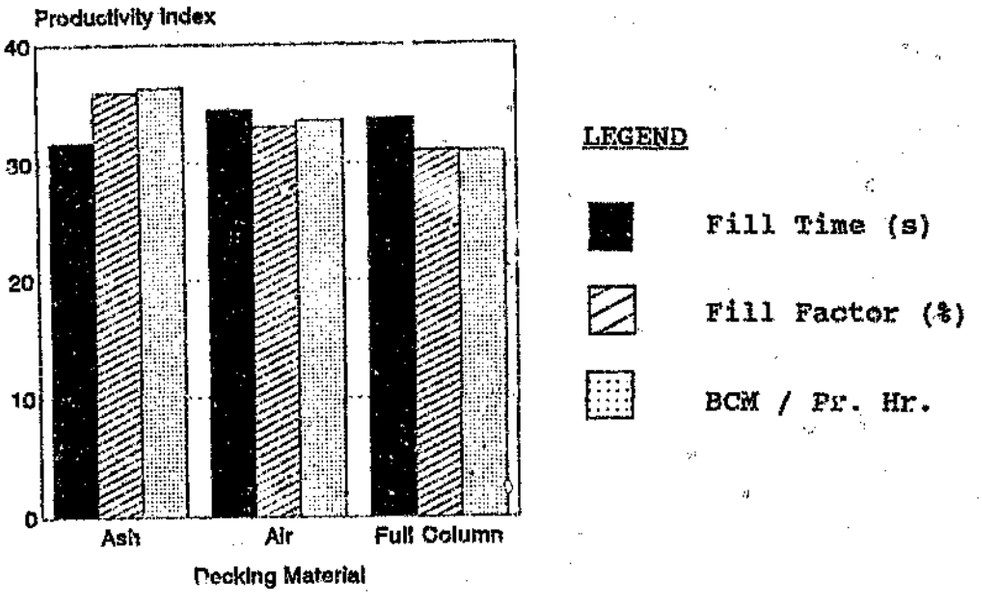


Figure 6.3 Effect of Decking on Dragline productivity

It is easily remarkable from the combined effect of the three productivity indices that ash used as decking material performs relatively better than air. The good performance in the ash deck is reflected in the lower fill times and higher fill factors as well as bank cubic meters per hour. With air decking (contrary for ash decking) gas moves quickly, within 1 to 2ms, from bottom to top charge, possibly desensitising the product and causing sympathetic detonation (Chiapetta, 1993). The explosive deflagrates and poor breakage results. This situation can be aggravated by high scatter in delay intervals as seen in the previous chapter. Ash decking is capable of slowing down such gas movement to 5 to 15ms. A full column charge did not yield the expected higher productivity. It is therefore probable that more explosives than necessary have been used to compensate for some blasting inadequacies. Earlier investigation by the mine operators also revealed equally acceptable fragmentation results when crushed stone is used as a decking material.

Effect of Drill Pattern on dragline productivity

The subject of spacing and burden has been covered by many authors: Brinkmann (1990), Hagan (1982), Vutukuri (1973), Cunningham (1991). In Table 6.6 various combinations of spacing and burden have been used: 10 x 10.2m, 11 x 10.2m, 11 x 10.5m and 10.5 x 10.5m. The dragline productivities for each pattern (expressed as percentages) are graphically illustrated in Figure 6.4. It is seen that the lowest drill pattern (10 x 10.2) gives the best results. This is manifest in higher fill factors as well as BCM/H values and lower fill times. This gives an indication of the possibility of getting better results by further

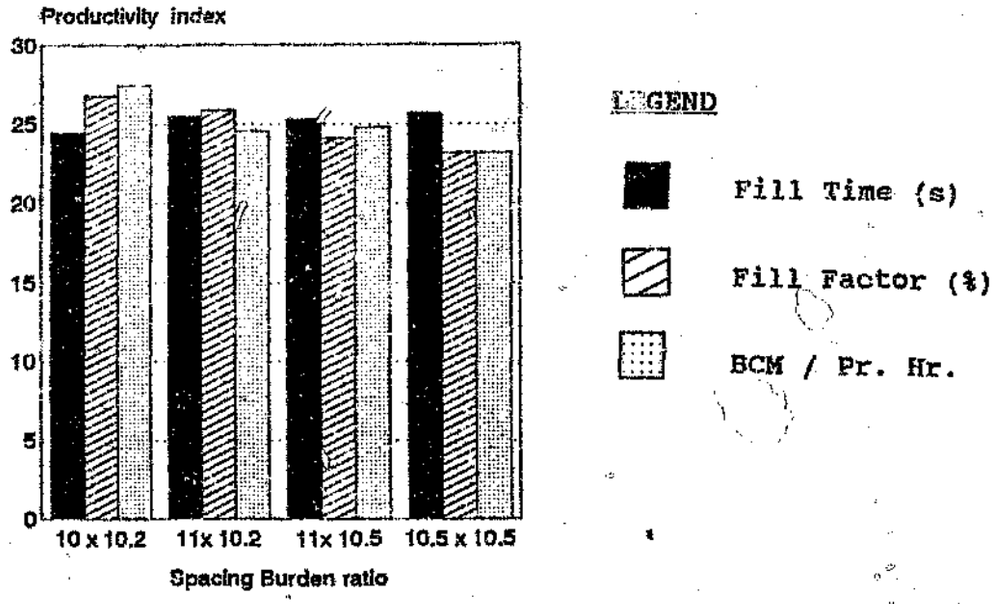


Figure 6.4 Effect of Spacing-Burden ratio on Dragline productivity

reducing the drill pattern. A spacing burden ratio of 9.5 x 8.5m has already been recommended following the sensitivity analysis conducted in Chapter 4 using the KUZ-RAM model.

Effect of powder factor on dragline productivity

Average powder factor values of 0.448kg/m^3, 0.456kg/m^3 and 0.483kg/m^3 have been used in different instances (Table 6.6). Figure 6.5 is a drawing of dragline productivities corresponding to the various powder factors. The figure presents an interesting picture in that higher productivity values are ironically associated with the lowest powder factor value of 0.448kg/m^3. In a similar vein, lower digging efficiency is portrayed with

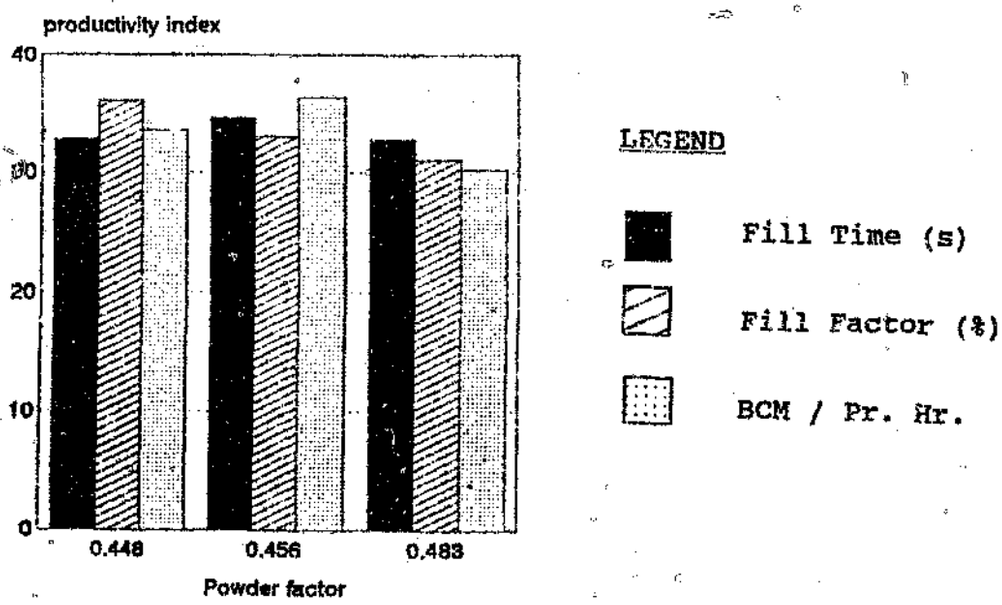


Figure 6.5 Effect of Powder factor on dragline Productivity.

the higher powder factor (0.483 Kg/m^3). This means that the avenues to exploit to improve fragmentation is not only increasing energy input but also other factors such as drill pattern. The increased powder factor did not necessarily improve dragline performance. The most plausible explanation for this observation is that the effect of increased powder factor (full column charge) is negated by using larger dimension drill pattern. i.e. a of 10.5 x 11cm. With this larger pattern the crater due to one hole may not overlap the next hole. Explosive energy appears inadequate and unevenly distributed resulting in problems such as misfires, poor fragmentation, tight muckpiles, excessive cratering, backspills and toe problems (Baur et al., 1981).

It is postulated here that irrespective of the energy input (powder factors), if other engineering field control measures such as drill pattern are not properly

and adequately checked, the desired fragmentation can hardly be achieved. In consequence dragline product vity is bound to be low.

6.4 Blast Monitoring Versus Dragline Productivity

Table 6.3 presents the dragline productivities and fragmentation results (assessed through the uniformity index) of the various blocks. These productivities were discussed in sections 6.2.1, 6.2.2 and 6.2.3 and these sections showed the importance of considering even what may at first seem unimportant details. The results of VOD measurements were presented in Table 5.1 and 5.4 and summarised in Table 6.5. This section relates dragline productivity to fragmentation and explains the reason for low productivities or otherwise of the dragline as uncovered by blast monitoring. It will throw light on such questions as:

- is the dragline production rate satisfactory or is the production at optimum rate?
- can the production rate be increased using better blasting? In other words is the drilling and blasting operation good enough?
- what is the effect of VOD of explosive and delay performance on dragline productivity?

It is very difficult to obtain accurate meaningful figures directly showing the actual influence of the drilling and blasting practice on dragline production as there are many variables that affect the blasting operation which are difficult to isolate. Nevertheless through blast monitoring (VOD measurements and high speed photography) and dragline productivity evaluation (using fill time, fill factor and BCM/Pr.Hr), the following scenarios have been encountered (refer to Table 6.3):

- Block 3205109-320543: Good fragmentation, relatively

easy digging but VOD below characteristic values.

- Block 320531-320676: Unfragmented sections of blast, difficult digging and VOD below characteristic values.
- Block 320665-320653: Apparently good fragmentation, uniform VOD but poor dragline performance.

The reasons for the occurrence of these specific scenarios are discussed below and presented as abbreviated case histories.

CASE 1: Good fragmentation, relatively easy digging but VOD below characteristic values:

Although most of the charges fired below characteristic VOD's, fragmentation appeared good and digging was relatively easier as manifested by the dragline productivity values in Table 6.3. The following reasons accounted for this situation:

- Geological information confirmed that rock in that area which is of sedimentary nature is soft, jointed and pre-fractured by nature. There is therefore in-built fragmentation which enhanced the breaking effect, regardless of explosive performance.
- Secondly, according to the propagation mechanism, if a charge is detonated instantaneous reaction at the secondary reaction zone behind the shock wave gives little shock and gas energy sufficient to fracture the weak rock in spite of low VOD (Brinkmann, 1993).
- The use of a smaller drilling pattern and ash decking obviously enhanced the breaking characteristics of the explosive.

It is therefore evident from the foregoing that fracturing in these blast areas is more a function of the natural prefractured state of the rock rather than the drilling and blasting technique. Hard rocks have higher

compressive and tensile strength than soft rocks, and the ability to transmit strain with much less attenuation (Atkinson *et al*, 1964). To blast hard rock effectively and to create good fragmentation, an explosive is needed with enough strain energy to overcome the tensile strength of the rock to initiate radial fractures and to cause spalling at the free face. Soft rock, on the other hand, requires comparatively little strain to start the cracks, but attenuate strain quickly. If a large strain is applied, the result will be much near-hole crushing without better breakage, at least in the vicinity of the hole. A low VOD explosive provides the strain required to initiate the cracks, but does not excessively crush the material. If the gas pressure is sufficient the material will be crushed in the same manner as with the high VOD explosive, but without the near hole crushing. For this reason, low VOD explosive is recommendable for the weak rock in the Syferfontein environment. Fracturing of the rock therefore does not necessarily require that much high VOD of the explosive, ie 4500-5000m/s. Provided the reaction is complete, a low VOD does not indicate reduced energy as long as the VOD is consistent (Ladds et al, 1990). In the circumstances, it even resulted in enhanced breaking effect in the weak rock.

CASE II: Unfragmented sections, difficult digging and VOD below characteristic values:

In certain portions of the area under study there were completely unfragmented sections where secondary blasting was effected, low dragline digging efficiencies (Table 6.3) recorded and VOD found to be below characteristic values. Reblasting an area is costly as well as difficult. The initial blasting operation tends to open up cracks and fissures in the bank that provide ready exit routes for the gaseous products formed by later explosive charges. The result was that an even higher

powder factor was used. The cost of having a dragline idle for want of suitably prepared material was, of course, prohibitive. Back tracking the results to blast monitoring, the poor fragmentation culminating in low dragline performance has been found to be attributable to the following:

- With such poor explosive performance (mean VOD of 2146 m/s and Range 3893 m/s) as observed in block 320543 in Table 5.1, explosive reaction proceeds abnormally and detonation becomes unstable. Secondary reaction behind the detonation front may not reach completion. The result is a drastic fall in energy output, and rock breakage is bound to suffer if the rock is not weak or prefractured.

- Furthermore VOD and explosion pressure (peak pressure applied to the borehole walls) are related (Brinkman, 1990)

$$P_e = 0.13dV^2 \qquad (15)$$

where

Pe = the explosion pressure (in Pascals). The explosion pressure is the peak pressure that a blasthole would experience if there was no expansion of wall during detonation.

d = density of the explosive in kg/m³

V = Velocity of detonation in m/s

The equation shows that pressure decreases with the square of VOD. The partial detonation of explosive in several instances as evidenced by low VODs (Table 5.1) could therefore result in significant drop-off in the peak pressure applied to the blasthole walls.

- High speed photography measurement for blast 320531

confirmed out-of-sequence firing in three rows. Also row 23 did not fire. The effect was overburdening for rows 24 onwards, and the blast was choked as more blowouts were observed.

- Timing inaccuracy partly contributed to the fragmentation problems. This is because the scatter in timing of the in-hole detonators (Table 5.5a) and the 67m/s surface delay (Table 5.5b, Block 320678) could possibly desensitise explosive in the adjacent holes and/or to part of the explosive column within the same hole.

The foregoing therefore reinforces the conclusion that the emulsion E-6000 has not been efficient enough to achieve the necessary rock breakage as portrayed by the erratic detonation results. Explosive incompatibility with the blasting environment is manifested with a concomitant poor dragline diggability and that re-evaluation of the explosive is necessary. The table top effect or completely unfragmented section encountered in block 320531 and 320576 is the result of the foregoing problems.

CASE III: Good fragmentation, uniform VOD but poor dragline performance:

In certain blasts dragline performance has been found to be unsatisfactory in spite of apparently good fragmentation and uniform VOD of emulsion E-6000 which was gassed to a density of 1.15 or less (See Table 6.6). It is established that:

- 'shooting in place', results in tight muckpile resisting bucket penetration in spite of apparently good fragmentation.
- Other external factors such as operators'

performance could contribute to such an anomaly. The operators' performance have been assessed in the next section.

loose muckpile which can assist the dragline to achieve high digging efficiency is warranted. On that basis a third factor (probably more heave) is needed which can be achieved through the selection of the right surface delay. An 84ms delay interval is appropriate for the recommended drill pattern of 8.5 x 9.5m. A re-evaluation of blast design and powder factors is again justified.

6.5 Evaluation of specific blasts

The above discussions focused in broader perspective or in general terms the influence of fragmentation on dragline productivity as revealed through blast monitoring. Thus, for comparative purposes it is essential to assess the blast yielding the poorest dragline productivity as well as the best blast among the 12 blasts studied. These would provide useful guidelines for future designs.

Using the combined effect of the dragline productivity and fragmentation, blast 320531 (refer to Table 6.6) is seen as yielding the lowest dragline efficiency. Several reasons already indicated in section 6.4 above are identified as the causes of this unacceptable performance. These include:

- Larger variation in VOD of explosive used. Fragmentation is adversely affected with such inconsistent detonation results.
- High speed photography revealed failure in Row 23 and consequently overburdening of Row 24 onwards.

Blast was choked and more blowouts observed, a situation which accounted for the "Table Top Effect" discussed earlier.

- Use of Emulsion E-6000 (density of 1.31) which has been found to be incompatible with the blasting environment.
- Drill pattern i.e spacing burden ratio of 11 x 10.2m is excessive and toe problems are not unexpected.
- Inefficient in-hole delay performance, i.e scatter in timing could result in desensitisation of top charge by bottom charge or of explosive charge in the adjacent holes.
- Use of air decking due to the reasons given in Section 6.2.3.

Appendix D presents the blasting parameters of Blast 320531 in question.

By the same token, of paramount interest is the Blast 320587 (Table 6.6) rated as the best blast as it yielded the highest dragline productivity. The design parameters of this blast together with the recommended changes are presented in Fig 6.6. This was the blast used as the base case for the sensitivity analysis in Chapter 4. This same blast will again be used as the base case in the next chapter when evaluating the influence of increased productivity on production cost.

6.6 Other Factors Influencing Dragline Productivity

The previous sections have evaluated dragline productivity on the basis of the various productivity indices. A lot of weight has been placed on fragmentation as a key parameter affecting dragline productivity. It should nonetheless be remembered that other external factors such as operator's performance and machine scheduling can have substantial impact on-dragline productivity and hence merit attention. This section

PARAMETERS	QUANTITY
A. Design Parameters	
1. Drill Pattern (SxB)	10.2 X 10
2. Hole Diameter (d)	250mm
3. Standard Deviation (w)	0.3
4. Hole Length (H)	28
5. Charge Length (L)	20
Rock Properties	
6. Young Modulus (Y)	15 MPa
7. Rock density	2.57
8. Poisson's ratio	0.25
9. Uniaxial Compressive Strength	80
10. Rock Mass Description (RMD)	50
11. Density Influence (RDI)	14.25
12. Hardness Factor	5
12. Rock Hardness index	4.16
Explosive Properties	
14. Powder Factor (Kg/m3)	0.49
15. Kg Explosive/hole	1273.8
16. RWS of Explosive	82.5 @ 1.15 RD
17. RWS of TNT	131
18. Bulk Density	1.323 @ 1.15 RD
B. Other Properties	
19. Explosive type	E-6000 (RD of 1.15)
20. Decking material	3 m Ash
21. Number of Holes/blast	-
22. Delays (a) Between Rows (b) In-hole	 84 ms T = 525, S = 450 ms
23. Boosters	360 X 454 g
24. Stemming	5 m drill chippings
25. Crest Burden	6
26. Toe Burden	8.5
27. Pres-plit	3m
28. Tie-in	V1 Configuration
29. Detonator	1 x 6D

Figure 6.6 Blast Parameters of the Best Blast - BLOCK 320587 -

therefore evaluates dragline productivity in these lights.

6.6.1 Assessment of Operators' Performance

One of the key factors greatly influencing dragline productivity but often neglected in performance analysis is the operators' performance. This is because of the subjective judgement entailed in such an evaluation. The low productivity of the dragline can hardly be justified solely on the basis of fragmentation consideration. Against this background this section assesses the influence, if any, of the operators performance on the dragline productivity to alleviate the experience factor associated with different operators. The operators performance will be separately evaluated for each of the twelve blasts (blocks) in question within the study period and will eventually be correlated to the dragline productivity.

As mentioned earlier the mine operates a 3 shift system with each shift of 8 hours duration. Each operator is identified by one of the symbols A, B, C etc. (See Appendix C). This has been done for all the blocks during the periods for which the dragline was operational. In Table 6.7 the operator's frequency during the digging period of each block is shown. As the mine keeps record of the operator's productivity by means of the total cubic material moved per operating hour (TCM/Hr), the performance index of each operator can be easily computed. This is effected by defining a parameter "Performance Index" as the product of the operator's productivity (P), percentage availability (A), and percentage utilisation (U). In particular, the "Performance Index" is given by the relation:

$$PI = P \times A \times U \qquad (16)$$

Table 6.7 Illustration of Operators' Frequency
- (23/2/93 - 29/6/93) -

BLOCK NUMBER	BLAST DATE	DIGGING PERIOD	OPERATORS FREQUE											
			A	B	C	D	E	F	G	H	I	J	K	L
3205109	23/2/93	23/2/93-13/3/93	10	12	13	5	4	6	-	-	-	-	-	-
320587	26/2/93	13/3/93-23/3/93	9	8	7	-	2	7	-	-	-	-	-	-
320576	03/3/93	23/3/93-30/3/93	5	1	6	-	4	6	1	1	-	-	-	-
320575	10/3/93	30/3/93-8/4/93	1	-	6	6	-	8	5	1	-	-	-	-
320543	18/3/93	8/4/93-15/4/93	2	-	4	4	2	4	7	1	-	-	-	-
320531	25/3/93	15/4/93-3/5/93	12	7	12	1	8	11	4	2	-	-	-	-
320501	30/3/93	3/5/93-8/5/93	3	4	4	2	3	1	-	-	1	-	-	-
3206108	01/6/93	4/6/93-22/6/93	14	12	-	-	3	16	1	-	-	-	10	1
320678	08/6/93	22/6/93-29/6/93	4	5	-	-	1	7	1	-	-	1	6	-
320676	11/6/93	29/6/93-8/7/93	4	7	-	-	-	9	2	-	-	-	8	-
320665	18/6/93	8/7/93-19/7/93	8	9	-	-	5	11	-	-	-	-	3	-
320643	29/6/93	19/7/93-2/8/93	4	11	1	-	1	14	-	-	1	-	11	2

For instance for Operator A with productivity (P) of 2247 Tcm/Hr, availability (A) of 86% and utilisation (U) of 78%, the Performance Index is calculated thus:

$$PI = 2247 \times 0.86 \times 0.7 = 1507$$

By similar calculation, each operator's "Performance Index" is presented in Table 6.8. On the basis of this "Performance Index" the operators are rated as 1, 2, 3 etc. with 1 being the best. (See the last column of Table 6.8). This rating associates an "efficiency factor" (F) to each operator.

To put these figures into perspective the average operators performance (AOP) of each block is computed. This is simply the weighted average obtained by taking the algebraic sum for the various operators (product of efficiency factor and operators frequency) and dividing by the sum of the operators frequencies. Thus the Average Operator's Performance (AOP) is such that:

$$AOP = \frac{E1F1 + E2F2 + E3F3+...+EnFn}{F1 + F2 + F3+...+Fn} \qquad (17)$$

In raference to Table 6.7 which shows the operator frequency (F) and Table 6.8 giving the efficiency factor (E), the "Average Operator's Performance" for block 3205109 can be calculated as follows:

$$AOP = \frac{10x9+12x2+13x3+5x10+1x1+6x11}{10+12+13+5+1+6} = \frac{270}{47} = 5.74$$

Presented in Table 6.9 is the summary of the Average Operator's Performance values for the various blocks. Rating has been effected on these values for each block to obtain the final "Operator's Productivity Rating"

Table 6.8 Evaluation of Operators' Performance

OPERATOR	PRODUCTIVITY (TCM/H)	AVAILABILITY (%)	UTILISATION (%)	PERFORMANCE INDEX (PI)+	EFFICIENCY FACTOR (E)
A	2247	86	78	1507	9
B	2654	90	83	1983	2
C	2430	88	87	1860	3
D	2320	83	76	1463	10
E	2130	81	78	1402	12
F	2755	78	97	2084	1
G	2213	85	75	1411	11
H	2330	90	77	1615	7
I	2498	88	84	1847	4
J	2275	87	80	1583	8
K	2254	89	84	1685	6
L	2417	87	82	1724	5

+ PI = Productivity x %Availability x % Utilisation

Table 6.9 Average Operators' Performance Versus Dragline Productivity.

BLOCK	AVERAGE OPERATORS PERFORMANCE	DRAGLINE PRODUCTIVITY (BCM/Pr.Hr)	OPERATOR PRODUCTIVITY RATING (OPR)	DRAGLINE PRODUCTIVITY RATING (DPR)	REMARKS
3205709	5.76	1463.05	3	12	Uncorrelated
320567	5.909	2162.06	4	3	Well correlated. Good performance.
320576	5.917	1829.14	5	7	Well correlated. Average performance.
320575	7.926	1995.51	12	4	Uncorrelated.
320545	7.791	1970.79	11	5	Uncorrelated.
320531	5.702	1618.32	1	10	Uncorrelated.
320501	5.730	2655.40	2	1	Well correlated. Good performance.
3206108	5.41	1801.25	10	8	Well correlated. Poor performance.
320678	6.167	1754.98	6	9	Uncorrelated.
320676	7.033	2382.46	8	2	Uncorrelated.
320665	7.387	1575.94	9	11	Well correlated. Poor performance.
320643	6.795	1910.05	7	6	Well correlated. Average performance.

(OPR). Also shown in the column 5 of Table 6.9 is the "Dragline Productivity Rating" (DPR) which gives an indication of the order of the dragline performance in the various blocks on the basis of the productivity index (BCM/Pr.Hr.) (Refer to Table 6.3). Regression analysis has been conducted to verify the relationship between the Operator's Performance and Dragline Performance (Figure 6.7).

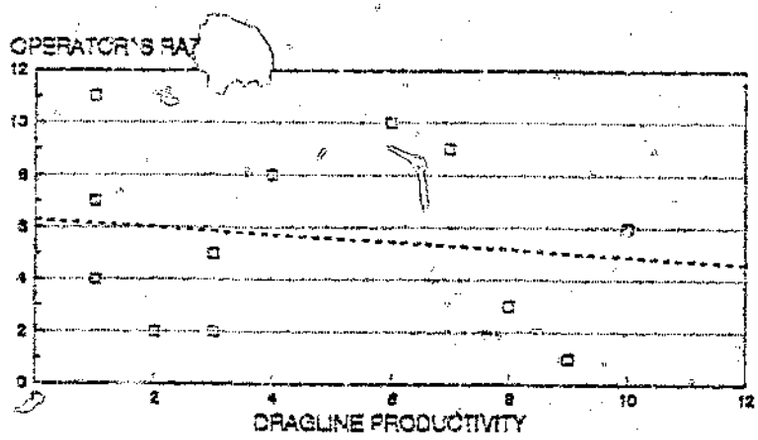


Figure 6.7 Regression Analysis: Operators' Rating Versus Dragline Productivity.

6.6.2 Discussion of Results

Viewed from a broader perspective there is no general linear correlation between the operators performance and dragline productivity *per se* as evidenced by a correlation coefficient of 0.42. This notwithstanding, by analyzing the results in Table 6.9 on a block by block basis the following scenario are evident:

- The relatively satisfactory dragline productivities in blocks 320587 and 320501 are partly attributable to the good performance of the operators.
- In a similar vein the poor performance of some operators is reflected in the low dragline digging efficiencies of blocks 3206108 and 320665.
- Average operators' performance is also exhibited in block 320643 where the operators' performance is seen to be well correlated with dragline productivity.
- Block 320531 and 3205109 present somewhat of a paradox. Whilst the operators' productivities are rated among the best in these blocks, the dragline productivities are considerably low. It is interesting to note that it is in this block

(320531) that the 'Table Top Effect' was encountered. This confirms the earlier conclusion that low dragline productivities in these blocks should be blamed on blasting related problems and not on operators, ability.

6.6.3 Dragline Scheduling

The dragline scheduled operating time makes allowance for periods of time when the dragline will not be operated. A scheduled operating time for draglines of 90% or better is typical (Bucyrus-Erie, 1976). The dragline availability is the part of the scheduled hours, expressed as percentage, that the machine is electrically and mechanically ready to work. Thus this is the total time the operator has available in which to strip coal. According to Bucyrus-Erie (1976) a 90% mechanical availability can be attained although a more normal figure would be 80%. Table 6.1 summarises the mechanical availability figures for the various blasts on the property being investigated. Mechanical availability values ranging form 80% to 89% with a mean value of 85.75% appear quite satisfactory. From a utilisation point of view the dragline utility is defined as the part of the machine available time, expressed as a percentage, that the dragline is actually stripping. The types of delay the utility allows for are those other than mechanical and electrical such as lunch, loss of electric power and waiting for blasting. This factor, according to Bucyrus-Erie (1976) varies from mine to mine but values in the low 90%'s can regularly be attained at well-organised properties (refer to Table 6.1). The values range from 72% to 85% with a mean value of 79.67% which certainly appears low. It is therefore evident that although availability on the dragline is satisfactory the utilisation is quite disappointing. The dragline scheduling calls for concern in order to reap the

benefits of efficient dragline operations.

6.7 Summary of results and Discussions

Dragline productivities are reviewed in the light of the productivity indices of fill factor, fill time and BCM/Pr.Hr. and have essentially been related to fragmentation. The study has also elucidated understanding of the operators' performances in relation to dragline productivity. The following are brought to light:

1. Fill factor values ranging from 52.6 to 74% with a mean of 69.62% are apparently low and implicity portray difficulty in digging. Furthermore a decreasing trend of fill factors within the study period leaves much to be desired.

2. There are no significant differences in fill time (digging time) for the various blasts considered. Nevertheless a fill time representing 30% of average cycle time is a cause for alarm as it portrays tight digging conditions resisting dragline bucket penetration.

3. The maximum productivity ever achieved among the blasts considered is 2655 BCM/Pr.Hr whilst the theoritical maximum is 3082 BCM/Pr. Hr. This substantiates the argument that there is room for improvement if related drilling and blasting problems are corrected.

4. The choice between ash and crushed stone as a decking material should be further investigated even though the greatest dragline productivities were recorded with ash decking. Crushed stone was not used for this test programme. In spite of that if

the same delay is used for both top and bottom charges then air decking is found to be the most appropriate.

5. Low productivities were found to be associated with higher powder factors simply because the effect of increased powder factor(through full column charge) is negated by using an excessive drill pattern , i.e (10.5 x 11m). It is envisaged that with the recommended drill pattern of 9.5 x 8.5m and powder factor of 0.49 kg/m^3, fragmentation will be improved and as a consequence higher dragline productivity can be expected.

6. In relating dragline productivity to fragmentation, three scenarios are revealed and explained:

- Good fragmentation, relatively easy digging but VOD are found to be below characteristic values. Rocks in this area are weak, and the use of smaller drill pattern as well as ash decking account for this occurrence. This justifies the use of a low VOD product for the Syferfontein operation.
- Unfragmented sections, difficult digging and VOD below characteristic values. This is blamed on very low VODs resulting in a significantly low explosion pressure; out-of-sequence firing and blowouts and poor in-hole delay performance are also contributory factors.
- Good fragmentation, uniform VOD but poor dragline performance. Shooting in place results in tight muckpile resisting bucket penetration in spite of apparently good fragmentation. Operators' performance is also a contributing factor.

7. Evaluation of the operators performance revealed in vivid terms that the low dragline productivities or otherwise in some specific blasts are attributable not only to blasting performance but also to operators ability. Thus the influence of the experience factor associated with different operators on dragline productivity is illustrated.

8. From availability standpoint, dragline availability of 86% is very satisfactory but a bleak picture of utilisation is portrayed as evidenced by the average utilisation of 79.67%. This calls for a review of the machine scheduling exercise.

CHAPTER 7

COST ANALYSIS

Chapter 7 will deal in general terms with the method of evaluating the drilling, blasting and dragline costs with a view to minimising the overall cost of production. It will touch base with such topics as contribution of the various cost centres to the total cost, regression analysis and stripping and mining cost relationship to production. Basically it will answer the question " what is the effect on costs when production is increased by small amounts on the mine? ". There is a growing tendency on the mine to lower drilling and explosive cost. However, it will be seen later in this chapter that the minimum cost may not always be in the best interest to the overall mining system. Any reported saving might possibly have been directly or indirectly passed onto the dragline operations. Often little or no attention is given to the dragline operating cost. In avoiding these pitfalls, this chapter attempts to investigate the total operation as a whole, rather than isolate any one segment. The cost figures are not actual values but they are realistic estimates from the authors own judgement based on the available data. Labour cost is excluded in the analysis.

7.1 Computation of drilling, blasting and dragline cost

In the analysis that follows drilling, blasting and dragline costs are the main cost centres each warranting separate consideration. The drilling and blasting costs are dependent on design parameters such as burden, spacing size of block (i.e BCM) as well as the quantity and price of explosives.

7.1.1 Drilling Cost

Consider a block of size 406620 bank cubic meters with design parameters as follows:

BCM	= 406620
Burden (B)	= 8.5m
Spacing (S)	= 9.5m
Depth of hole (h)	= 28m

The number of holes (N) to be drilled in this block is calculated from the relation:

$$N = BCM / (B \times S \times h) \qquad (18)$$

i.e Number of holes (N) = 406620 / (8.5 x 9.5 x 28)
= 180 holes

The drilling cost per BCM can therefore be estimated from the equation:

$$C_{dr} = N \times h \times C_h / (P \times BCM) \qquad (19)$$

Where:

N = Number of holes - 180
h = Depth of hole - 28m
Ch = Cost/Hr - R260.75/hr
P = Production rate - 39m/hr

The drilling cost is calculated from eqn (19) as

$$C_{dr} = (180 \times 28 \times 260.75) / (39 \times 406620)$$
$$= R\ 0.0829 / BCM$$

The drilling cost for all the 12 blasts (blocks) under investigation (Refer to Table 6.3) are calculated using the above methods and the results summarized in Table 7.1.

7.1.2 Blasting Cost

Before considering the blasting cost it is advantageous at this point to review a loading configuration of a typical blasthole. Assume that the blast hole is charged thus:

Stemming height (S) = 5m
Top Charge of Explosive (Et) = 5m
Decking Length (D) = 5m
Bottom Charge of explosive (Eb) = 12m

Length of explosive column (L) = Et + Eb (21)
= 5 + 12 = 17m

Assume further that the Explosive Mass per column rise i.e

Explosive mass/Column rise[2] (Kg/m) = 67, and
Cost of Explosive/Charge[2] (R/kg) = 1.016

Then Total Charge/hole = mass/column rise x charge length
= 67 x 17 = 1139 Kg/hole
.. Explosive Cost/hole = Kg/hole x Cost/kg (22)
= 1139 x 1.016
= R 1157.22/hole

[2] Mine standard values are used

If Cost of Accessories

i.e (boosters, Nonel, Delay)^2 = R41.76/hole

Then

Blasting Cost/hole = cost of explosives + Accessories

= 1157.22 + 41.76

= R1198.98/hole

Now number of holes = 180

The Total Blasting Cost is computed from the relation,

$$B_T = B_H \times N_H \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (23)$$

where

B_T = Total Blasting Cost - R
B_H = Blasting Cost / Hole - R/Hole
N_H = Number of Holes

∴ Total Blasting cost = R 1198.98 / Hole x 180

= R 215 816.40

The Blasting Cost / BCM can be computed from equation

$$\therefore \text{Blasting Cost / BCM} = \frac{\text{Total Blasting Cost}}{\text{Total BCM}} \quad (24)$$

$$= \frac{215816.4}{406620}$$

= R 0.531 / BCM

i.e Blasting Cost / BCM = R 0.531 / BCM

Presented in Table 7.1 is the blasting cost for the various blocks under investigation which were computed in a manner similar to the above.

7.1.3 Dragline Cost

The dragline cost is calculated simply from the cost per production hour (inclusive of ownership cost) and the number of production hours used for digging any particular block.

For instance for block 320587 rated as the best blast

Number of production hours = 188.06
Cost per production hour = 1208.71
Bank Cubic meters (BCM) = 406620

$$\text{Dragline Cost/BCM} = \frac{\text{Cost/hr} \times \text{Hours of production}}{\text{Bank Cubic Meters}} \quad (25)$$

$$= \frac{1208.71 \times 188.06}{406620}$$

$$= R0.5590/BCM$$

The total cost is the sum of the drilling, blasting and dragline cost. Thus

$$\text{Total Cost} = \text{Drilling} + \text{Blasting} + \text{Dragline cost} \quad (26)$$
$$= 0.0828 + 0.531 + 0.5590$$
$$= \text{R } 1.1728 \text{ / BCM}$$

The dragline cost as well as the total cost for all the blocks are also shown in Table 7.1. To facilitate the computation of the costs a spreadsheet was developed by the author based on the above analysis and a typical output is shown in Table 7.2. It is from this spreadsheet that Table 7.1 is built. The next section discusses the relationship between cost and dragline production and concludes with a regression analysis as well as the economic implications of the changes in design suggested earlier.

7.2 Stripping and Mining Cost Relationship to Production

The figures in Table 7.1 represent the drilling, blasting and dragline costs for the various blocks. Also shown on the table are these costs expressed as percentages of the total cost. To illustrate these graphically Figure 7.1 presents the percent distribution of these cost centres. It is remarkable that the drilling cost is insignificant as it contributes only 5.44% to 6.91% with a mean of 6% to the total cost. The blasting cost ranges from 33.33% to 45.26% with a mean contribution of 38% to the total cost. What is of paramount concern is the dragline cost which contributes 47.7% to 62.0% or an average of 55.94% to the total cost. To reduce the overall mining cost the best place to start is the operation with the highest individual cost, in the circumstances, the dragline cost. This is because reductions in this cost can produce the most significant reduction in overall cost (Bucyrus-Erie Company, 1976). This means that any change that may

TABLE 7.1 COST ESTIMATES FOR DRILLING, BLASTING AND DRAGLINE OPERATION

BLAST DATE (1)	BLOCK NUMBER (2)	VOLUME INSITU (BCM) (3)	NUMBER OF HOLES (4)	DRILLING COST		BLASTING COST		DRAGLINE COST		TOTAL COST (R/BCM) (11)
				COST (R/BCM) (5)	% OF TOTAL COST (6)	COST (R/BCM) (7)	% OF TOTAL COST (8)	COST (R/BCM) (9)	% OF TOTAL COST (10)	
23/2/93	3205109	396130	164	0.078	5.54	0.496	35.46	0.826	59.00	1.400
26/2/93	320587	406620	180	0.083	7.07	0.531	45.26	0.559	47.67	1.173
03/3/93	320576	257580	97	0.070	5.96	0.452	38.17	0.661	55.87	1.183
10/3/93	320575	308226	114	0.069	6.19	0.443	39.64	0.606	54.17	1.119
18/3/93	320543	241205	85	0.066	5.99	0.423	38.36	0.613	55.65	1.101
25/3/93	320531	508754	168	0.062	5.13	0.396	32.86	0.747	62.00	1.205
30/3/93	320501	185002	62	0.063	6.91	0.402	44.23	0.444	48.87	0.909
01/6/93	3206108	542700	195	0.067	5.75	0.431	36.85	0.671	57.40	1.169
08/6/93	320678	194400	65	0.063	5.44	0.401	34.82	0.688	59.75	1.151
11/6/93	320676	364230	114	0.059	6.23	0.375	39.89	0.507	53.89	0.941
18/6/93	320665	276750	96	0.065	5.20	0.416	33.33	0.767	61.47	1.248
29/6/93	320643	453466	166	0.069	6.01	0.439	38.49	0.633	55.51	1.140
AVG		344590.25		0.068	5.951	0.434	38.11	0.644	55.937	1.145
STG		114438.75		0.007	0.577	0.042	3.698	0.104	4.275	0.123

Table 7.2 Typical Output of a Spreadsheet for Cost Computation

A] Calculation of number of holes		
BCM	=	406620
Burden	=	8.5
Spacing	=	9.5
Depth of hole	=	27
* Number of holes	=	180
B] Calculation of drilling cost		
No of Holes	=	180
Depth of Hole	=	27
Cost/Hr	=	260.75
Production rate (m/hr)	=	39
** Drilling cost /BCM*	=	*0.080*
C] Calculation of blasting cost		
Stemming	=	5
Top Charge	=	5
Deck	=	5
Bottom charge	=	12
Explosive mass/column rise	=	67
* Total charge/hole	=	1139
Cost/charge (R/kg)	=	1.016
* Explosive cost/hole	=	1157.22
Accessories (boosters, Nonel,Delay)	=	41.76
Total BCM	=	406620
Number of Holes	=	180
* Blasting cost / Hole (R/Hole)	=	1198.98
* Total Blasting cost (R)		215817.12
** Blasting cost / BCM (R/BCM)*	=	*0.53*
D] Calculation of dragline Cost		
Cost/Pr.hr	=	1208.71
Number of production hours	=	188.06
** Dragline Cost / BCM*	=	*0.559*
TOTAL COST	=	1.170

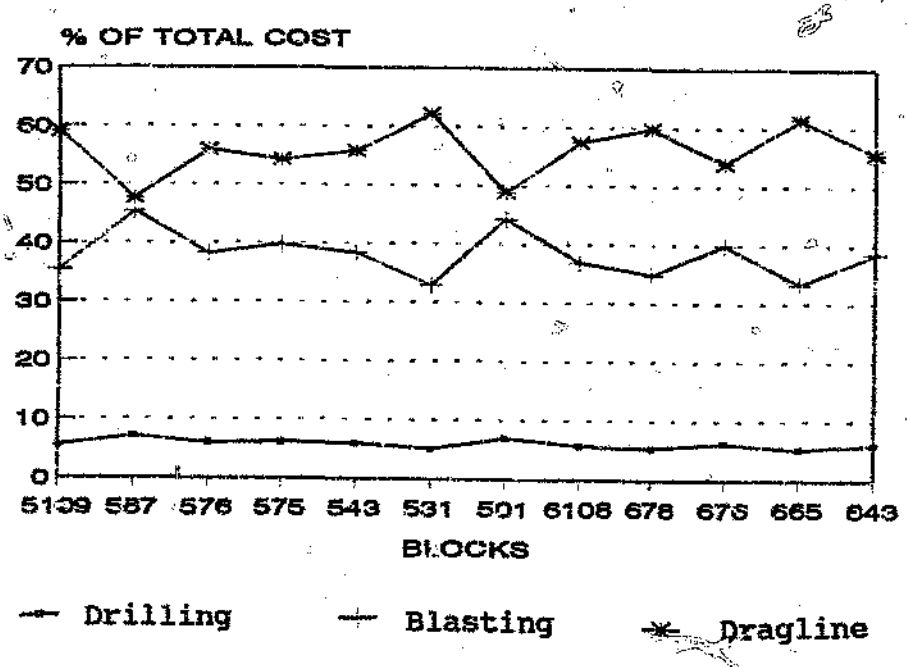


Figure 7.1 Contribution of the Various Cost Centres to the Total Cost

result in increased drilling and blasting cost but which can possibly enhance dragline productivity will drastically reduce overall cost.

7.2.1 Regression analysis on Cost

Figure 7.2 illustrates graphically the cost relationship to productivity (BCM/Pr.Hr) for the blasts (blocks) under investigation, the raw data of which is presented in Table 7.1. The following points are visible from the graph:

- The drilling and blasting cost is relatively constant and smaller than the dragline cost.
- The dragline cost is the most determinant factor, accounting for 60% of total cost. A decreasing trend of dragline cost with increasing productivity is evident.
- The total cost as for the dragline cost declines

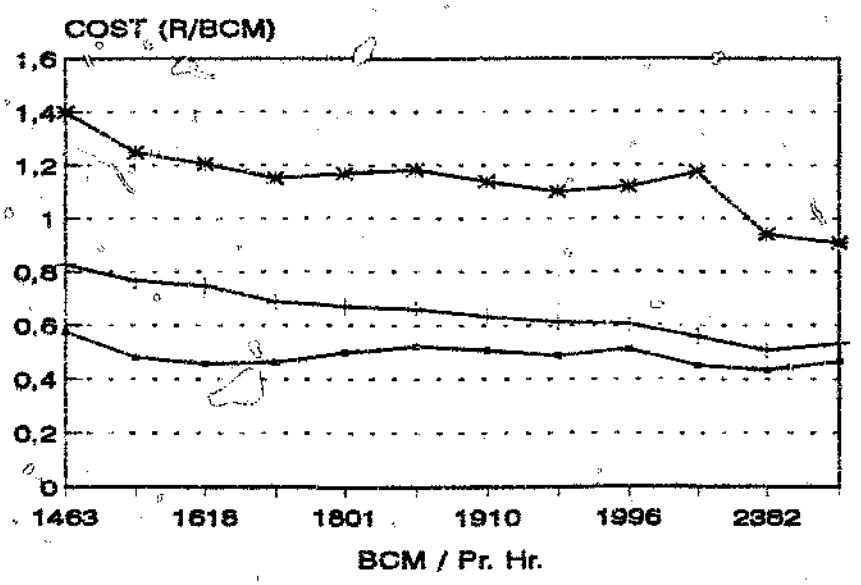


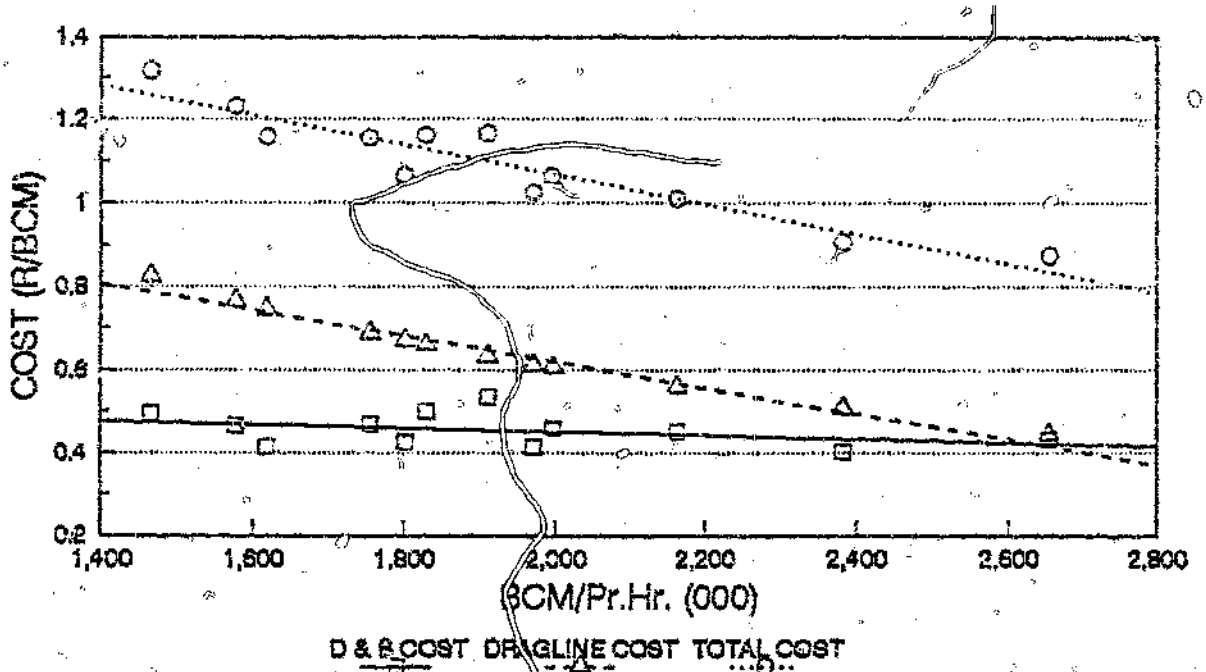
Figure 7.2 Relationship Between cost and productivity
- Existing Practice at Syferfontein -

with an increase in productivity.

- For productivity in excess of 2655 BCM/Pr.Hr (the maximum ever recorded during the study period), dragline cost is likely to be lower than drill and blast cost. The company's target should therefore be productivity in excess of 2655 BCM/Pr.Hr, in which case cost saving of a sort can be realised. It is important to recall here that the theoretical maximum productivity attainable by the Marion 8200m dragline is 3209 BCM/Pr.Hr. There is therefore no gainsaying the fact that the drilling and blasting practices on the mine must be improved to assist dragline diggability.

Regression analysis has been conducted to verify the relationship between the cost and dragline productivity

(Figure 7.3).



Figur 7.3 Regression Analysis of Cost Versus Productivity

This has led to the following deductions:

- the dragline cost is linearly related to dragline productivity by the regression equation

$$Y_D = 1.2473 - 0.00031X \qquad (27)$$

with a correlation coefficient of 0.97.

Where: Y = dragline Cost - R/BCM
X = Dragline Productivity - BCM/Pr. Hr.

- the total cost is also linearly correlated to the dragline productivity in accordance with equation:

$$Y_T = 1.07768 - 0.000354X \quad \ldots\ldots\ldots \quad (28)$$

the correlation coefficient being 0.88

- there is no linear correlation between the drilling and blasting cost and dragline productivity since the correlation coefficient is 0.14.

The dragline cost is also found to exhibit perfect linear correlation with the total cost as depicted by Fig 7.4.

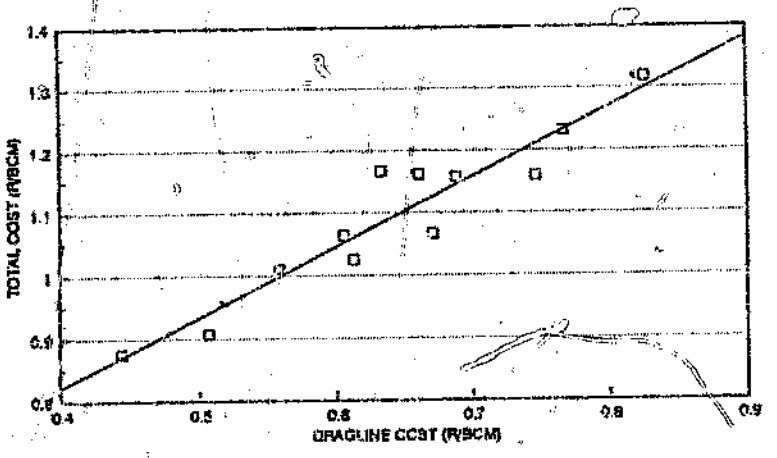


Figure 7.4 Regression Analysis: Dragline Cost versus Total Cost

The correlation equation is as follows:

$$C_T = 0.37348 + 1.12174C_D$$

Where:

C_T = Total Cost

C_D = Dragline Cost

The correlation coefficient is 0.90. The earlier claim that the dragline cost is the one to be targeted to achieve meaningful cost reduction is thus confirmed. Ironically this is the area least exploited by the mine when embarking on any cost reduction exercise. The doomsday attitude on cutting back on explosive cost as a cost reduction strategy should be reviewed. In chapter four which dealt with fragmentation, the existing practice on the mine was reviewed using empirical relations as well as a model and some changes were suggested. As a result the next section attempts to quantify the cost implications or benefits to be derived from increased productivity if the proposed changes are implemented.

7.2.2 Economic Implications of The Proposed Changes In Design

In chapter 4, some changes in design were suggested. This section analyses the cost implications of the proposed changes and uses the design parameters of Block 320587 (r..ed as the blast yielding highest dragline productivity) as the base case. The changes are summarised in Table 7.3 and graphically presented in Figure 7.5. In reference to Table 7.3 the original drill pattern is 10 x 10.2m at a current dragline productivity of 0162 BCM/Pr.Hr.

A change over to a closer pattern of 8.5 x 9.5m will necessitate drilling 38 additional holes, The drilling and blasting cost then increases from R 0.4451/BCM (Point A in Figure 7.5) to R 0.5247 (Point B) which represents a 17.88% cost increase. The dragline cost (D on Figure 7.5) apparently remains unchanged at R 0.559/BCM as it is independent of the number of holes drilled or quantity of explosives used. Point F on Figure 7.5 represents the

Table 7.3 Cost Implications of the Proposed Changes

PARAMETER	EXISTING PRACTICE	PROPOSED CHANGE	IMPLICATIONS OF PROPOSED CHANGE	EXPECTED RESULT AT (3082 BCM/H)	EXPECTED SAVINGS (R/BCM)	OVERALL REMARKS
Drill pattern	10 x 10.2	8.5 x 9.5	38 additional holes			(a) The proposed drill pattern (8.5 x 9.5) and powder factor (0.5 kg/m3) requires an economic breakeven point of 2256 BCM/H. This represents 17% and 4.3% improvement above company's average production of 1927 and current production of 2162 BCM/H resp.
Powder factor	0.4-0.49	0.5	Represents Company's average			
Drill & Blast cost (R/BCM)	0.4451	0.5247	- 10% above company's average - 17.88% cost increase			
Dragline cost (R/BCM)	0.559	0.559		0.292	0.262 (48% less)	(b) At the optimistic end, 3082 BCM/H production level or 60% improvement above average, estimated cost saving of R0.328/BCM is envisaged. This actually represents 16% above the maximum savings ever made at 2655 BCM/H production.
Total cost (R/BCM)	1.01414	1.08373	6.86% cost increase	0.6858	0.328 (32% less)	
Productivity (a) average (b) current	1927 2162		Requires 2256 BCM/H to breakeven i.e 17.15% & 4.3% above company's average and current level of production respectively.	3082 BCM/H (60% increase)		(c) The dragline cost and total cost are related to productivity as follows: Dragline cost: Y = 1.2473 - 0.00031X Total cost : Y = 107768 - 0.000354X where X = BCM/Pr.Hr.

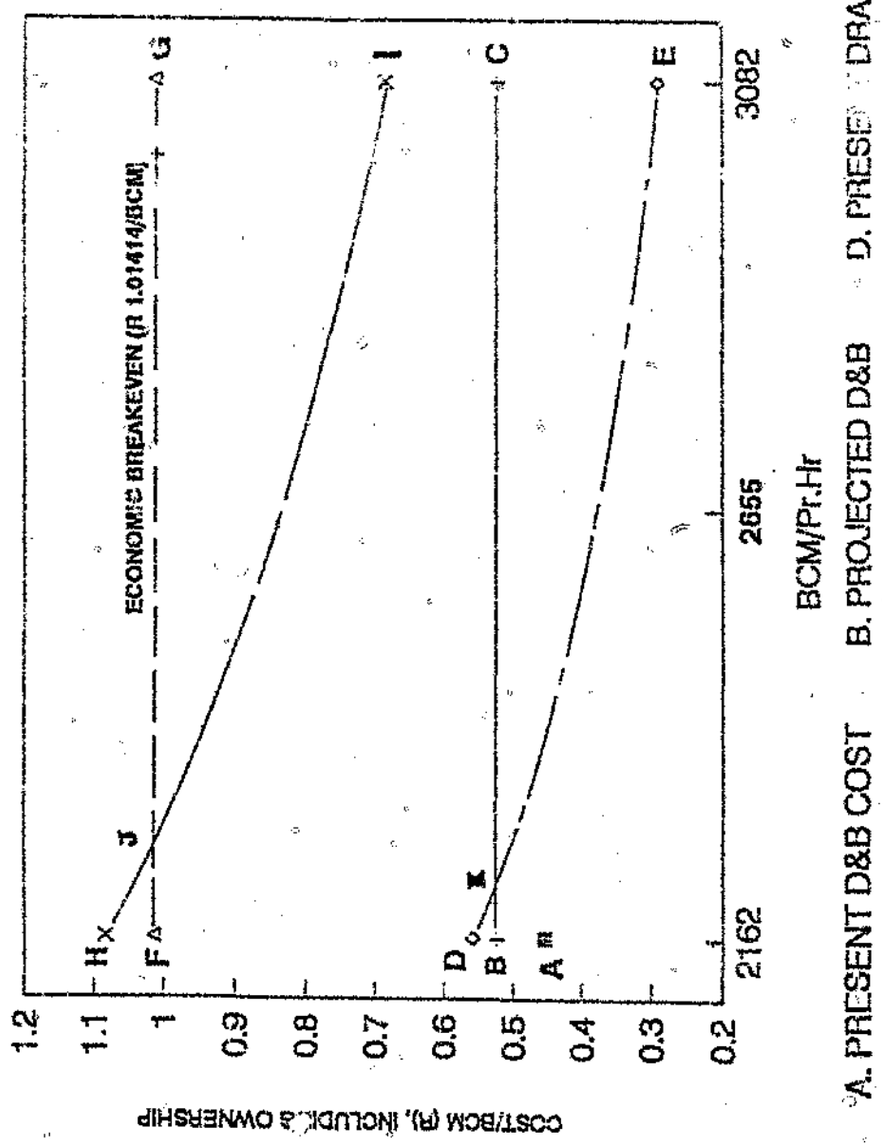


Figure 7.5 Cost Implications of The Proposed Changes

present total cost of R1.01414/BCM and with the changes increases to R 1.0837/BCM (Point H on Figure 7.5). This in effect represents 6.86% increase.

The horizontal line FG shows the total cost assuming there is no increase in productivity of the dragline with the changes and thus depicts the economic breakeven scenario. As observed through the regression analysis (section 7.2.1) the drilling and blasting cost showed apparently no variation with dragline productivity, hence the line BC. It is envisaged that with the recommended changes in design a dragline productivity value of 3082 BCM/Pr.Hr. can be attained. (The theoretical maximum attainable is 3209 BCM/Pr.Hr). The dragline cost as well as the total cost can be computed for this level of productivity from the regression equations (27) and (28) in section 7.2.1. Points E and I therefore represent the dragline and total cost respectively at the optimistic end of the scale (3082 BCM/Pr.Hr.). DE and HI therefore indicates the cost curves for the dragline and total cost respectively at the various levels of dragline productivities. It is worth mentioning that the company's average and current productivities are 1927 BCM/Pr.Hr. and 2162 BCM/Pr.Hr. respectively.

The following are remarkable from the graph:

- Beyond point J, the breakeven productivity of 2256 BCM/Pr.Hr., the projected total cost (JI) is bound to be lower than the existing total cost (JG). This implies that the proposed changes require dragline productivity of 2256 BCM/Pr.Hr. to breakeven. This represents 17.15% and 4.3% above the company's average and current level of 1927 BCM/Pr.Hr. of production respectively .

- For productivity in excess of 2190 BCM/Pr.Hr. (Point

K), the dragline cost would be lower than drilling and blasting costs in the projected scenario and significant savings in dragline cost can be expected as seen in section 7.2.1.

- The difference between FG and HI at any level of productivity beyond Point J manifests as cost saving. Figure 7.6 therefore shows the cost savings expressed as percentage of the present total cost to be expected at the various levels of productivity.

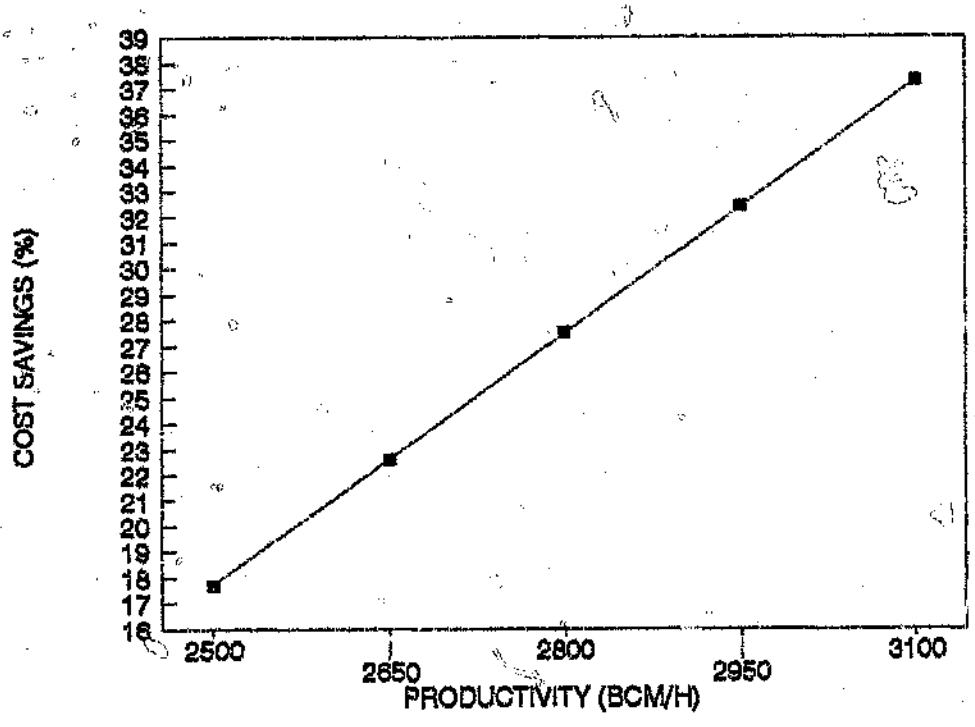


Figure 7.6 Percent Cost Savings Versus Productivity

At the optimistic end of the scale, a 3082 BCM/Pr.Hr production level or 60% improvement above average, an estimated cost saving of R0.328/BCM is envisaged which, to say the least, is a very significant saving. This actually represents 16% above the maximum saving ever

made at 2655 BCM/Pr.Hr. level.

To put this figure in perspective consider the block under investigation which has a size of 406620 BCM and was dug within a two week period by the dragline. The savings in Rand terms is

R 0.328/BCM x 406620 BCM = R133,371.36

In all certainty, these are large chunks of money realisable within two weeks at the said level of productivity. The need to improve the blasting technique, rather than economising on explosives, to assist dragline diggability can therefore not be over-stressed.

7.3 Summary of Chapter

1. The above cost analysis has revealed a contribution of 39% by the drilling and blasting operation to the total cost. With the greatest contribution by the dragline (61%) all attention must be focused here as this can produce the most tangible advantage in any cost reduction effort. It stands to reason that any change increasing the drilling and blasting costs with concomitant reduction in the dragline cost is bound to reduce the overall cost drastically

2. A declining trend of dragline cost and hence total cost with increase in production is established. The company's minimum target should be 2655 BCM/Pr.Hr to achieve cost savings. At the optimistic end of the scale, ie production level of 3082 BCM/Pr.Hr., a cost saving of R0.328/BCM is envisaged. This figure is not unrealistic if related drilling and blasting problems are critically addressed.

3. The company's dragline cost and total cost are related to productivity (X) as follows:

Dragline Cost: Y = 1.2473 - 0.000310X
Total Cost: Y = 1.7768 - 0.000354X

where X = BCM/Pr.Hr

Furthermore the dragline Cost (C_d) is related to the total Cost (C_t) by the regression equation.

$$C_t = 0.37348 + 1.12174\ C_d$$

with correlation coefficient of 0.90.

CHAPTER 8

CONCLUSION AND RECOMMENDATION

8.1 Conclusion

The existing practice (design) on the mine has been critically reviewed and empirical relations used where appropriate. Fragmentation has been assessed through a model and sensitivity analysis has been conducted on key variables that affect fragmentation. The following results are quite evident:

- The burden and spacing used on the mine are incompatible with the blast hole diameter (a major determinant of the burden-spacing ratio) and appears to be too much. A burden and spacing of 8.5 and 9.5metres have been recommended which, to all intents and purposes, will minimise the damaging effects of using excessive pattern and yet achieve, at the lowest possible cost, the required breakage, displacement and muckpile looseness for an efficient digging of the dragline.

- The 42ms and 67ms surface delay intervals between rows do not practically conform with the existing drill pattern. An 84ms delay interval is appropriate for the recommended drill pattern and will reduce the tight muckpile characteristic of causing hard

digging, low productivity of the dragline and increased wear and tear of the dragline teeth.

- The powder factors used are not expected to jeopardise fragmentation. Contrary to expectation fragmentation problems do exist and were found to be partly attributable to the excessive drill pattern used in certain blasts. Against the background of fragmentation problems in spite of apparently satisfactory powder factors, it was imperative to monitor certain blasts.

The usefulness of monitoring blasts through velocity of detonation measurement and high speed photography in rapidly and cost effectively identifying blast performance related problems are thus illustrated. The advantages of backing up VOD measurements with high speed photography are also realised. Three blasts on decked holes and three others on full column charge (no decking) have been monitored and the results indicated the following:

- Unsatisfactory performance of the explosive emulsion E-6000 of relative density 1.31 as in 39 out of 42 cases it detonated below characteristic VOD. This situation unequivocally led to fragmentation problems in certain blasts. Explosive was gassed to density 1.15 or less and three blasts were monitored. All the holes except two fired within the characteristic range.

- Some performance-related problems such as variable detonation results, top primer firing before bottom primer although timed to initiate the reverse, misfires and very low VODs have been uncovered through VOD measurements. It has been more than

amply demonstrated that the foregoing are partly responsible for fragmentation problems and hence low dragline digging efficiencies. A number of factors may have caused inconsistent explosive performances. The scope of the monitoring did not, however, extend to identifying the causes of the variable performance of explosives.

- The results from high speed photography also indicated that out-of-sequence firing occurred in three rows out of eight. Also failure of one row led to overburdening of the subsequent rows, blast becoming choked and more blowouts observed. This accounted for a completely unbroken section or "Table Top Effect" on that blast.

- It has been conclusively demonstrated that the 25ms in-hole delay interval, i.e top 525ms and bottom 500ms, is simply inadequate as 7 out of 9 cases (77.78%) a distinct deviation from expected scatter was observed. Statistically this assures 73.24% confidence of deviation exceeding the allowable scatter, a situation which could result in bottom charge desensitising the top charge or explosive column in adjacent holes.

- The best way of utilising the air bags for decking should be investigated further, since indications are that slipping of the bags does occur. This might have been the reason why some of the flares did not ignite.

Dragline productivity has been reviewed in the light of the productivity indices, viz, Bank Cubic meter per production hour (BCM/Pr.Hr.), fill factor (bucket factor) and fill time, and has essentially been related to fragmentation. It has been established that dragline

productivity falls short of expectation. The BCM/Pr/Hr. was found to be the most satisfactory productivity measurement criterion and as a consequence received preference in assessing the dragline performance. This was found to show noticeable fluctuations for the different blasts investigated. A study of the fill factors for the various blast places a question mark on dragline productivity as it has gradually, and yet noticeably decreased over the study period. Generally speaking, from availability standpoint, dragline availability appears satisfactory. Nevertheless a bleak picture of utilisation is portrayed as evidenced by average utilisation of 76.67%, a situation necessitating a review of the machine scheduling operation.

In relating dragline diggability to fragmentation the following scenario are encountered:
scenarios

- Good fragmentation and relatively easy digging although VODs were found to be below characteristic values. Rock in this area was found to be weak and the use of smaller drill patterns accounted for this occurrence. Fracturing of the rock does not necessarily require that much high VOD of emulsion ie 4700 - 5500 m/s. This justifies the use of low VOD products for the Syferfontein operation.

- Unfragmented section, difficult digging and VOD below characteristic values. This is blamed on very low VODs recorded in some blasts resulting in significantly very low explosive pressure, out-of-sequence firing and blowouts as well as poor in-hole delay performance.

- Good fragmentation, uniform VOD but poor dragline performance. Shooting in place results in tight muckpile resisting bucket penetration in spite of

apparently good fragmentation. Operators' performance was also found to be a contributory factor. The importance of evaluating the operator's performance as a key parameter influencing dragline productivity is also illustrated.

cost analysis revealed a contribution of the drilling and blasting cost of 39% to the total cost (Table 7.1). With the greatest contribution by the dragline cost (61%) any change increasing the drilling and blasting cost with a concomitant reduction in the dragline cost is bound to reduce the overall cost drastically. Attention must therefore be refocussed on the dragline cost as it will produce the most tangible advantages in any cost reduction effort.

- A change in drill pattern has been proposed with the new drill pattern of 9.5 x 8.5m and the cost implications analyzed. Any dragline productivity in excess of 2655 BCM/Pr.Hr. would result in cost savings to some extent. It is envisaged that at the optimistic end of the scale, i.e a production level of 3082 BCM/Pr.Hr (the theoretical maximum being 3209 BCM/Pr.Hr), a cost saving of R0.328/BCM can be realized.

- It has been found through regression analysis that the company's dragline cost and total cost are related to productivity as follows:

  Dragline cost: $Y_d = 1.2473 - 0.000310X$
  Total cost: $Y_t = 1.7768 - 0.000354X$
  Where: X = BCM/Pr.Hr.

  Furthermore the dragline cost (C_D) is related to the total cost (C_T) by the regression equation

$$C_T = 0.37348 = 1.12174\ C_D$$

8.2 RECOMMENDATIONS

The foregoing has evaluated the existing design on the mine and uncovered the reasons for low productivity of the dragline through blast monitoring and fragmentation assessment. In the light of the findings the following recommendations are worth considering:

1. Emulsion E-6000 of relative density 1.15 (com for density 1.31) is found to exhibit a satisfa level of performance an cs use must be encouraged. It is recommended that the VOD of this explosive be reduced to further enhance its breaking capability in the weak rock.

2. Decking is recommended if the shale layer is interbedded between sandstone layers. However if the shale layer disappears then a full column charge must be used. The choice between ash and crushed stone as a decking material should be further investigated.

3. The in-hole delay of 25ms (Top 525ms and Bottom 500ms) gives unacceptable scatter and its use should be discouraged. The in-hole delay interval could be increased to 50ms , i.e top charge 500ms and bottom charge 450ms. Alternatively undelayed blast could yield equally acceptable results in which case top charge firing time as for bottom charge should be 350ms.

4. For surface delay consideration an 84ms delay interval would produce the necessary loose muckpile and enhance the digging capability of the dragline.

5. If the mine wants to reap the benefits of increased productivity of the dragline and reduce cost, the

drill pattern of 9.5 x 8.5m and powder factor of 0.49 Kg/m³ can be used.

For such a change the typical loading configuration for a 27m hole will be as follows:

Stemming to surface = 5m
Top charge = 5m
Stemming deck = 5m
Bottom charge = 12m

Total explosive weight/hole = 1139Kg
BCM/hole = 2713.2
Overall Powder factor = 0.49Kg/m³

Cost implication of the proposed charges are as follows:

Drilling Cost (R/BCM) = 0.0827
Blasting Cost (R/BCM) = 0.4419
Dragline Cost (R/BCM) = 0.559
Total Cost = 1.08373

6. The mine can use the calculation similar to that in section 7.1 for the estimation of drilling, blasting and dragline cost. From the production viewpoint the following regression equations can be used to estimate the dragline as well as the total cost:

Dragline Cost: $Y = 1.2473 - 0.000310X$
Total Cost: $Y = 1.7768 - 0.00035X$

Where X = BCM/Pr.Hr.

These equations would, to all intents and purposes, serve as a useful guide in quick assessment of cost at any production level.

7. Few changes such as drill pattern and powder factor have been suggested. For the other design considerations, maintain the status quo. A typical example is given in Appendix E.

8. In order to effectively, strategically and economically monitor the dragline productivity, it is suggested that the dragline budget should be based not only on volume of waste moved per unit time i.e BCM/Pr.Hr but also the cost per bank cubic meter (R/BCM).

9. The operators' performance needs to be assessed periodically to keep them on their toes. One should also not loose sight of the fact that for effective dragline digging, the bucket should be dragged from the floor up so that gravity can assist in quickly filling the bucket. The frequency of dragging the bucket on the surface must be minimized as it prolongs the fill time which in turn manifest as production cost increases.

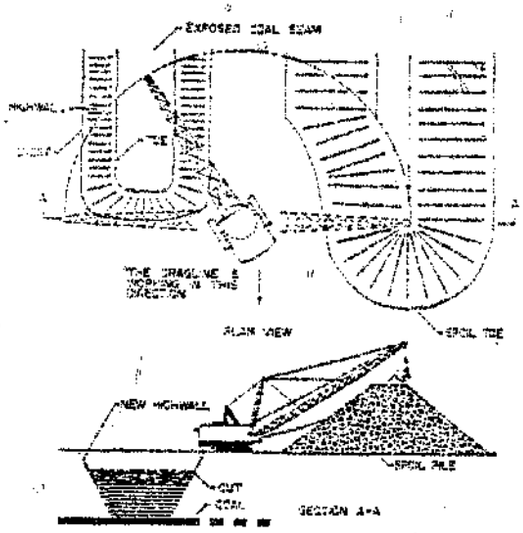
10. Given that fragmentation related problems are addressed, it is further suggested that the dragline scheduling be critically reviewed in order to more efficiently improve its utilisation. The high capital intensity of the dragline operation justifies this argument.

With the closer drilling pattern, improved powder distribution, improvement in operators' performance, well organised dragline scheduling and increased delay interval to eliminate tight muckpile, savings could results from the following:

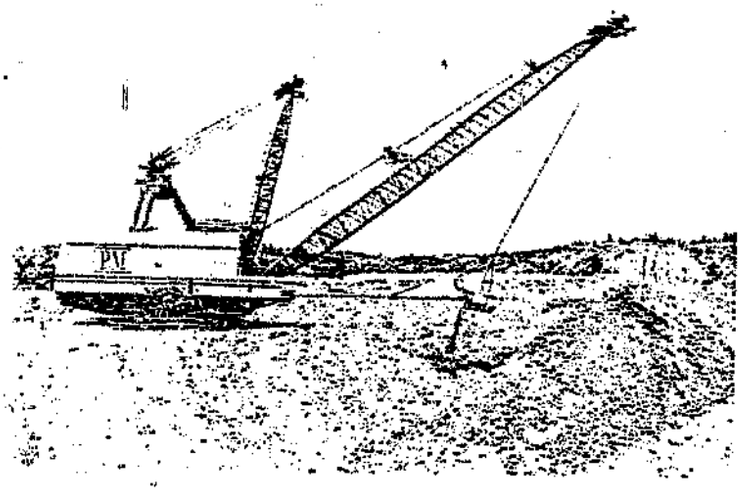
* Increased loading rate and hence increased dragline productivity and reduced fill time.

* Reduced bill for wear and tear.
* Lower power consumption.
* Reduction of vibrational levels.

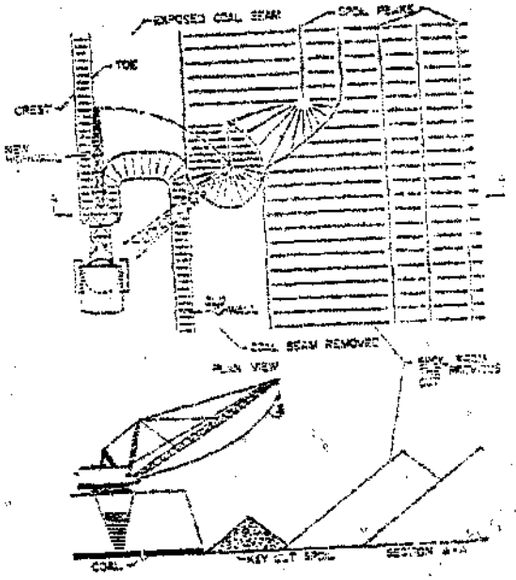
APPENDIX A: SCHEMATIC OF THE DRAGLINE DIGGING MODES



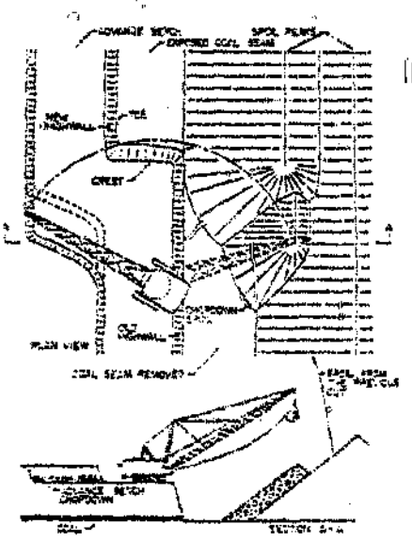
Appendix A1: Plan and cross sectional view of a Box Pit, Side Cut Method.



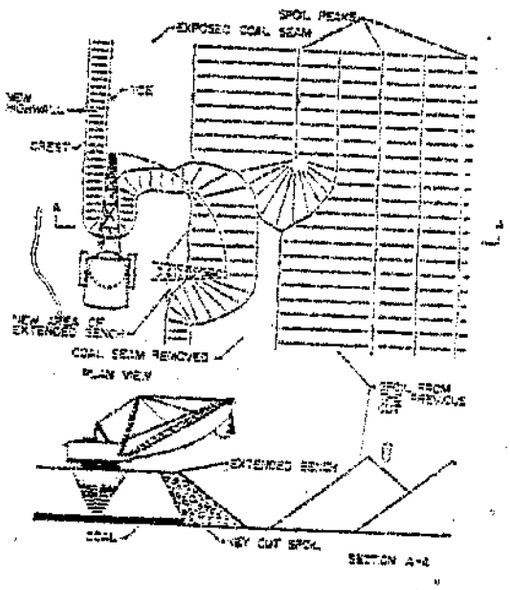
Appendix A2: Dragline using the simple side casting stripping method.



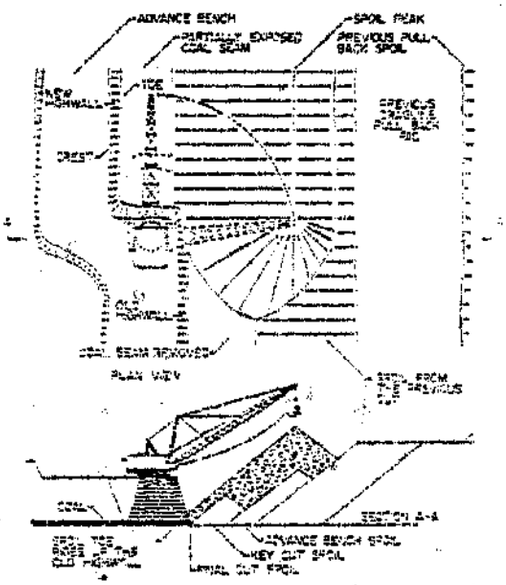
Appendix A3: Plan and cross sectional view of a Simple Side Casting Method illustrating the removal of the initial key cut.



Appendix A4: Plan and cross sectional view of the Advanced Bench produced by the Dragline Chopdown Method.

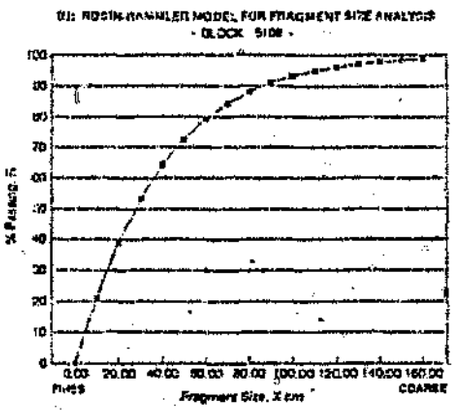


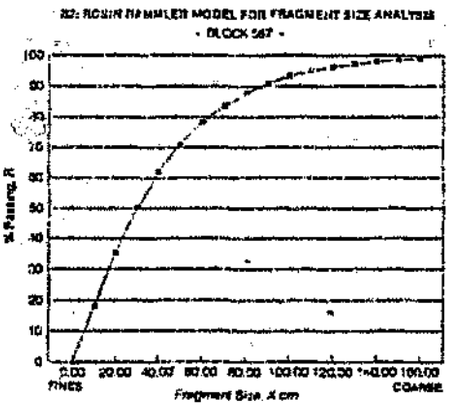
Appendix A5: Plan and cross sectional view of a Dragline producing an extended Bench using material dug from the key.

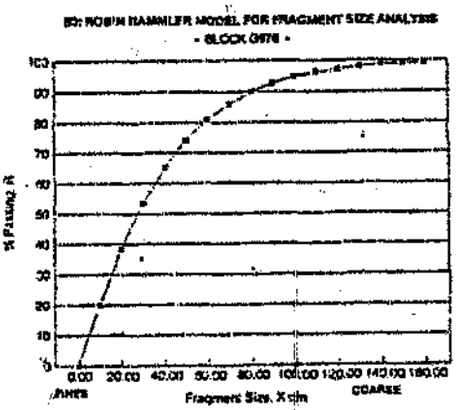


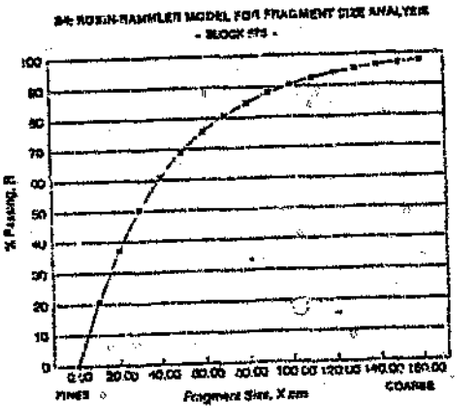
Appendix A5: Plan and cross sectional view of a Dragline performing the last cut of an advance Bench as part of the pull back stripping method.

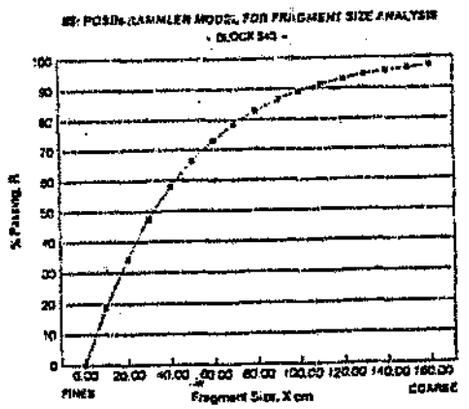
APPENDIX B: Rosin-Rammler Fragmentation Curves

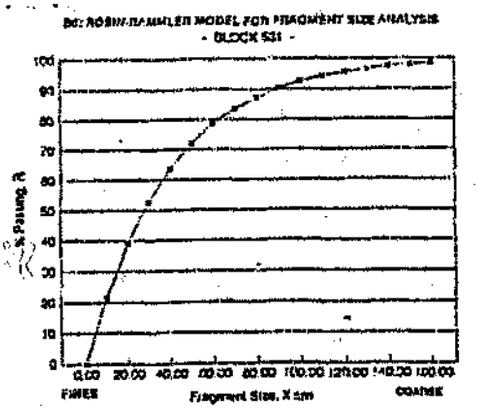


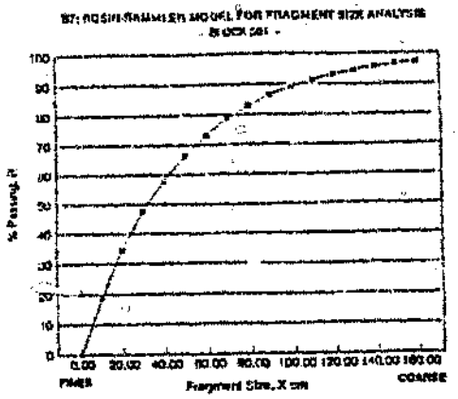








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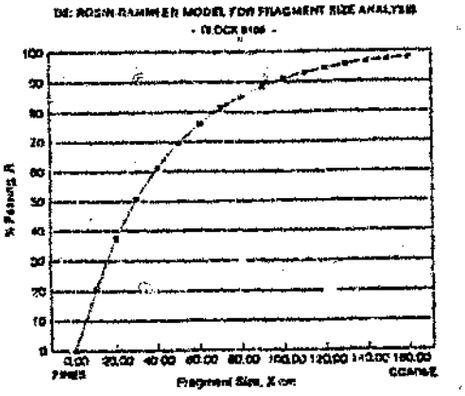
Fragment Size, X cm
FINES
COARSE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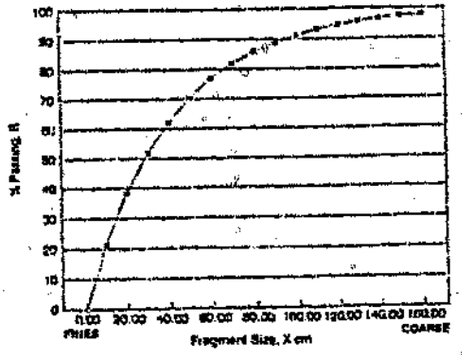
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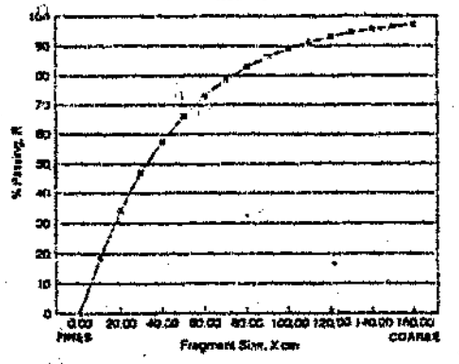
Fragment Size, X cm
FINES
COARSE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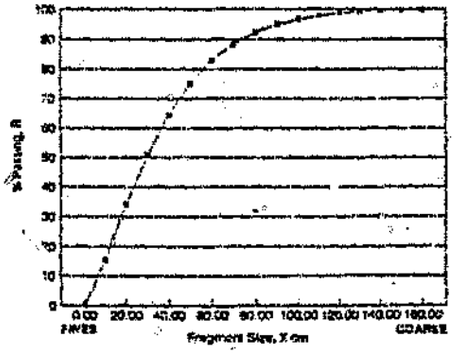
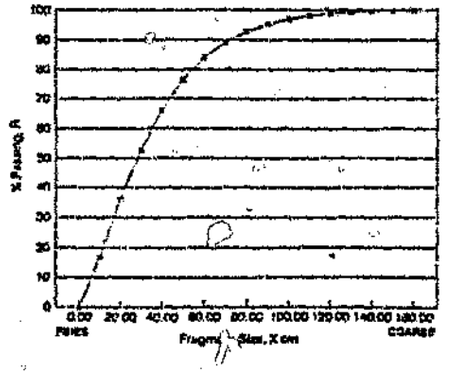
Fragment Size, X cm
FINES
COARSE


% Passing, R
Fragment Size, X cm
FINES
COARSE


% Passing, R
FINES
COARSE

APPENDIX C : ASSESSMENT OF OPERATORS' PERFORMANCE

BLAST DATE	BLOCK NUMBER	DIGGING PERIOD	DATE	OPERATOR		
				NIGHT	MORNING	AFTERNOON
23/2/93	3205109	23/2/93-13/3/93	23/2	A	B	C
			24/2	D	B	C
			25/2	D	E	E
			26/2	D	A	E
			27/2	A	A	C
			28/2	F	C	B
			01/3	D	A	B
			02/3	D	E	B
			03/3	C	A	B
			04/3	C	G	B
			05/3	C	G	B
			06/3	C	G	B
			07/3	C	G	B
			08/3	C	G	B
			09/3	C	G	A
			10/3	C	G	A
			11/3	B	G	A
			12/3	B	C	A
			13/3	B	C	A
26/2/93	320587	13/2/93-23/3/93	13/3	B	C	A
			14/3	B	C	A
			15/3	B	C	G
			16/3	B	C	G
			17/3	A	C	G
			18/3	A	B	G
			19/3	A	B	G
			20/3	A	B	G
			21/3	A	B	G
			22/3	A	E	C
			23/3	A	E	C
03/2/93	320576	23/3/93-30/3/93	23/3	A	E	C
			24/3	G	E	C
			25/3	I	E	C
			26/3	G	A	C
			27/3	G	A	C
			28/3	G	A	C
			29/3	G	E	B
			30/3	G	A	H

BLAST DATE	BLOCK NUMBER	DIGGING PERIOD	DATE	OPERATOR		
				NIGHT	MORNING	AFTERNOON
10/2/93	320575	30/3/93-8/4/94	30/3	G	A	H
			1/2	C	I	H
			2/2	C	G	D
			3/2	C	G	H
			4/2	C	G	D
			5/2	C	G	D
			6/2	C	G	D
			7/2	H	G	D
			8/2	H	G	D
18/3/93	320543	8/4/93-15/4/93	8/4	H	G	D
			9/4	H	C	D
			10/4	H	C	D
			11/4	H	C	D
			12/4	H	C	G
			13/4	H	H	G
			14/4	A	E	G
			15/4	A	E	I
25/3/93	320531	15/4/93-3/5/93	15/4	H	G	D
			16/4	A	E	E
			17/4	A	H	G
			18/4	A	H	I
			19/4	A	E	C
			20/4	A	E	C
			21/4	G	E	C
			22/4	G	E	C
			23/4	G	A	C
			24/4	G	A	C
			25/4	G	A	C
			26/4	I	A	B
			27/4	G	A	B
			28/4	C	A	B
			29/4	C	G	B
			30/4	C	E	B
			1/5	H	G	B
			2/5	C	G	B
			3/5	C	E	A

BLAST DATE	BLOCK NUMBER	DIGGING PERIOD	DATE	OPERATOR		
				NIGHT	MORNING	AFTERNOON
30/3/93	320501	3/5/93-8/5/93	3/5	C	E	A
			4/5	C	E	A
			5/5	B	E	A
			6/5	B	J	G
			7/5	B	C	D
			8/5	B	C	D
01/6/93	3206108	4/6/93-22/6/93	4/6	B	K	G
			5/6	B	K	G
			6/6	B	K	G
			7/6	B	K	G
			8/6	B	E	G
			9/6	E	E	A
			10/6	A	B	A
			11/6	A	B	G
			12/6	A	B	G
			13/6	A	B	G
			14/6	A	B	K
			15/6	A	L	K
			16/6	G	H	K
			17/6	G	A	K
			18/6	G	A	K
			19/6	G	A	K
			20/6	G	A	G
			21/6	G	A	B
			22/6	G	A	B
8/6/93	320678	22/6/93-29/6/93	22/6	G	A	B
			23/6	K	A	H
			24/6	K	G	B
			25/6	K	G	B
			26/6	K	G	B
			27/6	K	G	B
			28/6	E	G	A
			29/6	K	G	A
11/6/93	320676	29/6/93-8/7/93	29/6	K	G	A
			30/6	B	G	B
			1/6	B	K	A
			2/6	B	K	G
			3/7	B	K	G
			4/7	H	K	G
			5/7	H	K	G
			6/7	B	K	G
			7/7	A	K	G
			8/7	A	B	G
18/6/93	320665	8/7/93-19/7/93	8/7	A	B	G
			9/7	A	B	G
			10/7	B	B	G
			11/7	A	B	G
			12/7	A	B	K
			13/7	A	B	E
			14/7	G	A	K
			15/7	G	A	G
			16/7	G	E	E
			17/7	G	E	B
			18/7	G	E	K
			19/7	G	A	B

APPENDIX D: DESIGN PARAMETERSOF BLAST 320531

- RATED AS THE POOREST BLAST -

PARAMETERS	QUANTITY
1. Drill Pattern (S x B)	11 X 10.2
2. Hole Diameter (d)	250m
3. Standard Deviation (w)	0.3
4. Hole Length (H)	27.5m
5. Charge Length (L)	17m
Rock Properties	
6. Young Modulus (Y)	15 GPa
7. Rock density	2.57
8. Poison ratio	0.25
9. Uniaxial Comprssive Strength	80
10. Rock Mass Description (RMD)	50
11. Density influence (RDI)	14.25
12. Hardness factor	5
13. Rock Hardness index	4.16
Explosive Properties	
14. Powder factor (Kg/m3)	0.48
15. Kg Explosive / hole	1234.3
16. RWS of explosve	83.3 @ 1.31 RD
17. RWS of TNT	131
18. Bulk density	1.398 @ 1.31 RD
B. Other Parameters	
19. Explosive type	Emulsion (RD of 1.31)
20. Decking material	5m Ash
21. Number of Holes /blast	
22. Delays	
(a) Between rows	84ms
(b) In-hole	Top = 500, Bottom = 500
23. Boosters	336 x 454kg
24. Stemming	5m drill chippings
25. Crest burden	6.m
26. Toe burden	8.5
27. Presplit	3m
28. Tie-in	V1 configuration
29. Detaonator	1 x 6D

APPENDIX E: DESIGN PARAMETERS - RECOMMENDED CHANGES

PARAMETERS	EXISTING PRACTICE	RECOMMENDED CHANGES
A. Parameters for fragment size analysis		
1. Drill Pattern (S x B)	11X10.5m, 10.5x10.5m, 11 x 10.2m, 10 x 10.2m	9.5 x 8.5
2. Hole Diameter (d)	250m	250m
3. Standard Deviation (w)	0.3	0.3
4. Hole Length (H)	28m	28m
5. Charge Length (L)	17m	17m, 22m
Rock Properties		
6. Young Modulus (Y)	15	15
7. Rock density	2.57	2.57
8. Poison ratio	0.25	0.25
9. Uniaxial Comprssive Strength	80	80
10. Rock Mass Description (RMD)	50	50
11. Density influence (RDI)	14.25	14.25
12. Hardness factor	5	5
13. Rock Hardness index	4.16	4.16
Explosive Properties		
14. Powder factor (Kg/m3)	0.39 - 0.49	0.49
15. Kg Explosive / hole	1167 - 1449.4	1139
16. RWS of explosve	83.3 @ 1.31 RD	82.5 @ 1.15 RD
17. RWS of TNT	131	131
18. Bulk density	1.398 @ 1.31 RD	1.323 @ 1.15 RD
B. Other Parameters		
19. Explosive type	E-6000 (RD of 1.31)	Emulsion (RD of 1.15)
20. Decking material	3 - 5 Air, Full Column	5m Ash
21. Number of Holes /blast	62 - 168	
22. Delays		
(a) Between rows	42ms, 67ms, 84ms	84ms
(b) In-hole	Top = 525, Bottom = 500 Top = 500, Bottom = 500	350ms or Top = 500, Bottom = 450
23. Boosters	(170 - 336) x 454kg	80 x 454kg
24. Stemming	5m drill Chippings	5m drill chippings
25. Crest burden	5, 5.75, 6m	6m
26. Toe burden	8.5	8.5
27. Presplit	3m	3m
28. Tie-in	V1 configuration 1 x 6D	V1 configuration 1 x 6D
29. Detaonator	V1 configuration 1 x 6D	

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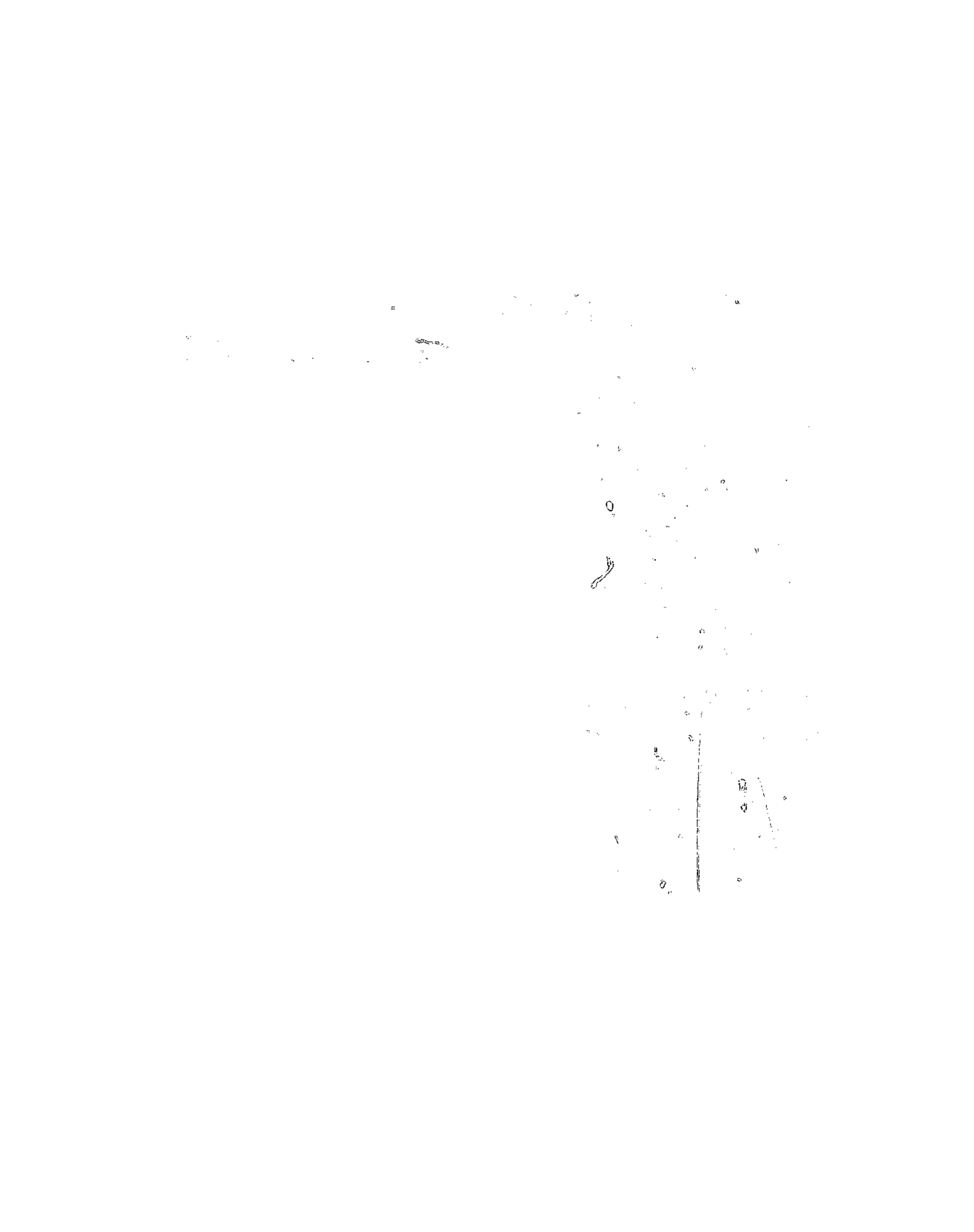

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