MINING EQUIPMENT SELECTION FOR A TYPICAL OPENCAST MINE IN THE WITBANK AREA.

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ABSTRACT

A literature survey revealed that numerous articles and papers have been published on different aspects of opencast mining equipment selection but none of these considered the whole range of machines required for any particular mining operation.

This project report aims to provide a foundation on which to base decisions concerning the selection of mining equipment for opencast coal mines in the Witbank Area of South Africa with the object to optimize initial capital outlay and to minimize operating costs.

For the purpose of this study a typical geological section of the area, consisting of soft overburden or topsoil, underlaid by hard sandstone and siltstone overburden and the five main coal seams, invariably split by sandstone partings was selected. For each individual mining activity all the factors influencing the selection of mining equipment are tabulated, followed by a review of possible mining methods and an economical appraisal of the various methods and equipment proposed.

Results indicate that wheel tractor-scrapers, commonly used by operating mines in the Witbank Coalfield, can be succesfully replaced by hydraulic excavators and rear dump trucks for the removal of soft overburden. This mining method can also be extended to include parting removal and coal mining, resulting in the employment of a smaller variety of mining equipment and better utilization of available equipment.

The importance of drill evaluation is highlighted since inadequate fragmentation, the direct result of poor drilling and blasting techniques, extends to other facets of opencast mining.

Draglines, generally used in surface coal mines, prove to be a successful mining method with distinct capital and operating cost benefits.

DECLARATION

I declare that this project report entitled "Mining Equipment Selection for a Typical Opencast Mine in the Witbank Area" is my own work, except to the extent indicated in the acknowledgement and references. It is being submitted as partial fulfilment for the degree of Master of Science in Engineering at the University of the Witwatersrand, Johannesburg. It has not been submitted before for any degree or examination at any other University.

G.A. FOURIE

1st day of August, 1984

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LIST OF ABBREVIATIONS AND SYMBOLS

AL/ANFO	:	Aluminized Ammonium Nitrate Fuel Oil
AN	:	Ammonium Nitrate
ANFO	:	Ammonium Nitrate Fuel Oil
APA	:	Anthracite Producers Association
BCM	:	Bank Cubic Metres (In-situ Volume)
BWE	:	Bucket Wheel Excavator
cm	:	centimetre
GPa	:	Giga Pascal
g/cc	:	grams per cubic centimetre
kg	:	kilogram
km	:	kilometre
m	:	metre
m^2	:	square metre
m^3	:	cubic metre
mm	:	millimetre
M	:	million
min	:	minute
m/s	:	metres per second
MPa	:	Mega Pascal
No	:	Number
R	:	Rands (S.A. Unit of Currency)
RD	:	Relative density
ROM	:	Run-of-mine
RPM	:	Revolutions per minute
t	:	ton
TNT	:	Trinitrotoluene
TCOA	:	Transvaal Coal Owners Association
4U1	:	Number 4 upper 1 seam
4U2	:	Number 4 upper 2 seam
4L	:	Number 4 lower seam

Metric units are used throughout this report.

CHAPTER 1

INTRODUCTION

Capital investment totalling well over R1 500 million has been committed to expansion of the South African coal mining industry's production capacity in the past eight years (Chamber of Mines of South Africa, 1981). This expansion has involved giant new mines, some of which have set both South African and world records for size, production or advanced technology.

Exports have played the key role in a rejuvenation of the industry during the 1970's and, in view of the promising outlook for world markets, will continue to keep coal producers committed to huge expansion during the coming decade (Chamber of Mines of South Africa, 1981).

The bulk of South Africa's coal both for the inland market and for the various overseas users, is produced from the well-known Witbank coalfield. Mining methods have, until the past decade, been restricted to underground mining using conventional bord and pillar methods. During recent years some very large opencast mines have been established in the Witbank Area and a number of projects are at present being evaluated and planned by various mining companies.

In selecting a particular opencast stripping method, the overall objective is to remove the material at the least possible cost. Accomplishment of this goal requires careful consideration of many factors, including geological and topographic information, proximity of the stripping operation to waste disposal areas, possible problems with land reclamation, the types of suitable equipment available and auxiliary equipment requirements.

A literature survey by the author revealed that numerous articles and papers have been published on different aspects of opencast mining equipment selection but none considered the whole range of machines required for any particular mining operation.

For the purpose of this study a typical geological section of the area was selected, consisting of a reddish brown, sandy clay topsoil, which to satisfy mining regulations, must be removed as a separate operation to a minimum depth of one metre and used as replacement material during rehabilitation of the disturbed surface. This topsoil is underlaid by hard sandstone and siltstone layers, and the five main coal seams typical of the Witbank Coalfield.

For each individual mining activity all the factors influencing the selection of mining equipment are tabulated, followed by a review of possible mining methods and an economic appraisal of the various methods and equipment proposed.

Several criteria are used to determine the excavatability of soft overburden (Korhonen et al, 1971; Bieniawski, 1976; Barton, 1974; Jennings, 1973; Weaver 1975). The use of the refraction seismograph as an aid for the determination of the class of excavation in terms of the rippability of materials was developed by the Caterpillar Tractor Company during 1958. This method was further developed by Weaver (1975) to incorporate other geological factors likely to influence the assessment of rippability. Although both methods give an indication of the rippability they do not provide, however, sufficient information on the excavatability of the material and very often a conclusive answer without test trials can not be obtained. Kirsten (1982) developed an excavatability index based on geological characteristics that can be determined by standard engineering tests or inspections and relates this to an excavatability classification system that is applicable for the entire range of natural materials from the weakest soil to the hardest rock. By applying the excavatability classification system of Kirsten (1982) to the topsoil of the Witbank Area, it can be proved that this material can be excavated relatively easily without blasting or ripping.

To evaluate different topsoil mining methods, a typical haulroad profile and pit lay-out were used as the basic inputs to a computer simulation programme developed by Barlow's Tractor Division. (Blenkinsop and Tredrea, 1982). Production rates, capital costs and operating costs for the following machines were calculated:

- scrapers and dozers
- hydraulic shovels and trucks
- front end loaders and trucks

Results obtained indicate that, irrespective of production requirements, the hydraulic excavator and haul truck fleet has definite financial advantages. This mining method can also be extended to include coal mining and the removal of the thin parting between coal seams, resulting in a smaller variety of equipment on the mine which will contribute to better utilization of available machines and standardization of operating and maintenance procedures.

As most of the overburden underlying the topsoil consists of sandstone, siltstone and shale, with uniaxial compressive strengths up to 165 MPa (Steffen, Robertson and Kirsten, 1982), breakage by explosives is essential. To accomplish this, it will be necessary to drill a hole into the rock for placement of the explosives. Investigations by Bauer (1967) and Praillet (1983) indicate that drill penetration rate, which is dependent on rock properties, bit rotation speed and bit pressure and bit diameter, has a strong influence on drill productivities and unit costs. Of these factors, the rock properties to be encountered at a given mine are uncontrollable whereas the rotary speed and pulldown can be varied by the drill operator within design limits of the machine. The first step in selecting a suitable production drill is therefore to determine the uniaxial compressive strength of the rock strata to be encountered during mining operations.

Three drill sizes (152mm, 250mm and 311mm diameter) were analysed to determine and compare capital and operating costs for various annual production rates. From the results obtained it is evident that the small drilling unit has limited application in the Witbank Area, and can only be used successfully in very small strip mining operations. The medium size rotary drill (250mm diameter), with a pull down capability of 65 000 kg provides the mine operator with flexibility and reliability in the drilling of hard overburden. In addition, the capital and operating costs of this unit compare favourably with the other drills, even for very large annual production rates.

A number of alternatives are available to strip broken overburden and to expose the underlying coal seams. The selection criteria to be considered are described when selecting the following mining equipment:

- draglines
- electric rope shovels
- front end loaders
- haul trucks

To illustrate the importance of evaluating alternative mining methods and equipment for overburden stripping, a typical opencast mining operation is analysed for three different types of equipment. Although the initial capital cost of a dragline operation is higher than the other two methods (hydraulic excavators and electric rope shovels) it presents a lower overall unit cost per cubic metre of overburden stripped over the project life. This saving becomes more significant with increasing production rates. Draglines, commonly used in the Witbank Area for overburden stripping, have proved to be very successful, and should be considered as the basis of design for all opencast operations in this area.

The five main coal seams of the Witbank Coalfield are inconsistent in thickness and the individual seams are often split by a sandstone, siltstone or shale parting. As result of this inter-relationship between the coal seams and the interbedded parting, it is important to select mining equipment that will excavate both the coal and parting horizons successfully at an economic cost. Due to the coal and parting hardness, drilling and blasting is required. To minimize the creation of fine coal, which is not acceptable in the final product, the optimum drill hole diameter varies between 75mm and 125mm. A large selection of mining methods is available for loading and hauling the coal and the parting. A comparison between the electric rope shovel and the hydraulic excavator indicate that a substantial cost saving is possible when using hydraulic excavators as primary loaders. The main advantage of the hydraulic excavator is its ability to break out poorly fragmented coal or parting. The lighter weight of the hydraulic excavator, when compared to a similar size rope shovel, is another important factor.

Bottom dump coal haulers provide sufficient capacity to contain the relatively low density coal which enables the rated truck capacity to be fully utilized. They also provide a low discharge height which is advantageous in preventing the creation of excessive fine coal material.

Under certain conditions, e.g. tied collieries, mining equipment can account for up to 70 percent of the initial capital cost required to establish a new opencast mine, while the operating cost of such equipment can represent as much as 80 percent of the total on-mine production costs. These factors will make a large contribution to the overall viability and profitability of a new mining project. This project report aims, therefore, to provide a foundation on which to base decisions concerning the selection of mining methods and mining equipment for potential opencast coal mines in the Witbank Area, in order to optimize initial capital expenditure and to minimize operating costs over the project life.

CHAPTER 2

THE WITBANK COALFIELD

2.1 History

Records indicate that in about the year 1868 coal was mined in the Witbank area by Dutch settlers and local Black tribes for their own domestic use (Graham, 1931).

In 1879, G.W. Stowe discovered coal in the Vereeniging district of the Transvaal, but it was not mined systematically until 1889. The coal was transported by means of ox-wagon to the Kimberley diamond mines - a distance of approximately 500 kilometres. This method of transportation effectively prevented any attempt to produce coal on a large scale, and consequently the total monthly output was restricted to about 1 000 tons per month.

Gold was discovered on the Witwatersrand in 1886 and we owe the commencement of coal mining in the Witbank district on a commercial scale, to this discovery.

In 1894 the railway line from Pretoria was extended to Delagoa Bay (Maputo). This extension, together with the rapid expansion of the gold mines, created a demand for large tonnages of high grade coal. This demand might well be termed the main cause of the development of the great Witbank Coalfield.

By 1907 the South African coal mining industry was well established and sales reached approximately 4,8 million metric tons to the value of about R3,2 million. Most of the collieries in the Transvaal were operating without profit and many actually incurred losses as result of the severe competition in the industry.

The effects of this chaotic situation were described by Brunette (1968), as follows: "One or two 'popular' collieries had more orders than they could fulfill while neighbouring collieries producing exactly the same quality and class of coal but having a slight geographical disadvantage could not find sufficient orders to keep going."

It became obvious that if the coal producers could not get together to put a stop to cut-throat competition and stabilise selling prices in order to cover the costs of production and to ensure a reasonable return on the substantial capital investment, financial disaster of the collieries was inevitable.

The desperate situation in which the coal producers found themselves brought them together, and in 1907 they negotiated an agreement to co-operate in marketing their products, on a profitable basis, that would give them a fair return on their investment and would ensure the consumer a suitable product at a reasonable price. This agreement later became known as THE STANDARD AGREEMENT.

The Transvaal Coal Owners Association (TCOA) came into being in 1907 as a co-operative non-profit marketing organization, but having no actual control over the mining of coal by its member collieries.

Coal sales in South Africa in 1966 amounted to just in excess of 46 million tons with a rate of increase in the order of 4,5% per annum.

The rapid increase in the oil price since 1973 has increased the price of energy to such an extent that demand for coal has again developed.

Since 1970, the TCOA has taken a leading role in the export market, together with the related Natal Associated Collieries and Anthracite Producers Association, starting with the successful negotiation and implementation of the Japanese low ash blend coking coal contract. This provided the base load which made the development of Richards Bay Coal Terminal and its associated rail links possible.

Since then, exports have dominated the TCOA turnover and more especially, its profitability. In 1981, approximately R400 million, out of total coal sales of R615 million, was derived from exports. Slightly less than half of this export turnover was derived from Japanese contracts.

Mining methods in the Witbank Coalfield have until the past decade been restricted to underground mining using conventional bord and pillar methods. During the last decade some very large opencast mines have been established in South Africa and specifically in the Witbank Area producing high quality coal for the export market or large tonnages for thermal power stations in close proximity to the coal fields. Examples of such mines are Optimum Colliery, Kriel, Duvha, Rietspruit, Arnot and Middelburg Mine. A number of projects are at present being evaluated and planned by various mining companies and the establishment of new opencast mines is imminent.

2.2 Geology

The Witbank Coalfield is situated on the irregularly eroded northern margin of the Karoo Sequence (Figure 2.1).

Five main coal seams are present within the coalfield with qualities ranging from low grade bituminous coal, suitable for power generation, to high-grade, steam coal and coking coal.

The basin itself has an uneven floor, and is subdivided by pre-Karoo ridges into a series of sub-basins. These pre-Karoo "highs" have resulted in the non-deposition of the lower seams in places. The total thickness of the Ecca Group sediments in the Witbank Coalfield rarely exceeds 180 metres.

The five main seams of this coalfield are identified, from the base upwards, as numbers 1 to 5. A generalised stratigraphic column is shown in Figure 2.2

The number 1 coal seam, also known as the Bottom Seam, ranges from 1,2 metres to 3 metres in thickness in the northern part of the

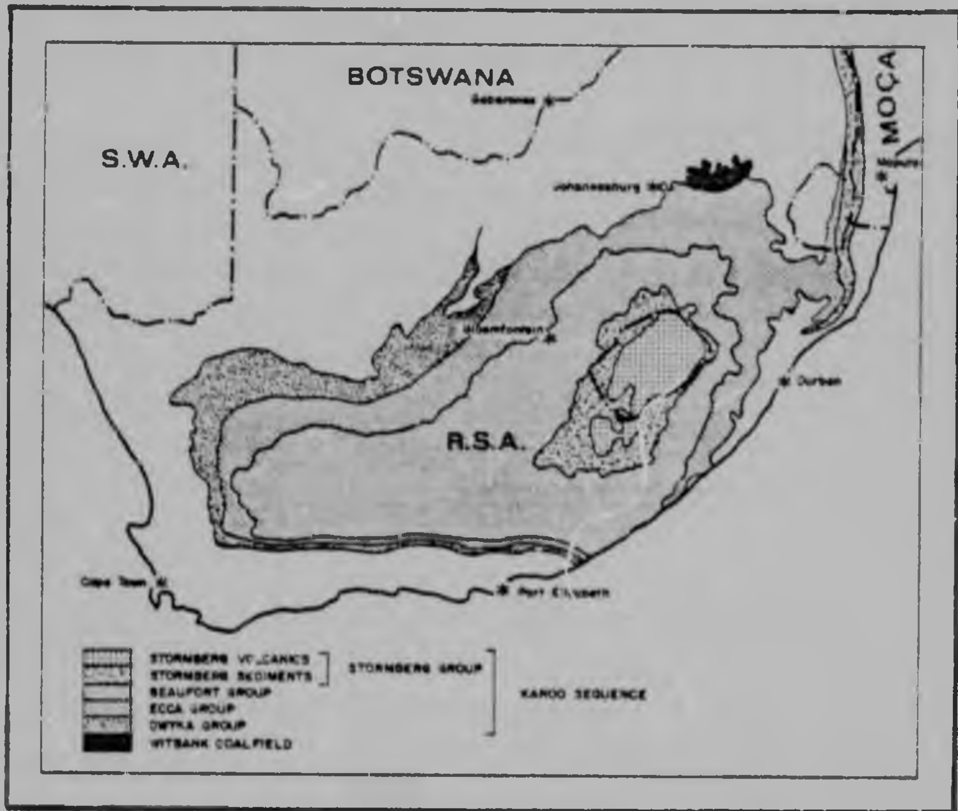


FIGURE 2.1

DISTRIBUTION OF KAROO SEQUENCE AND LOCATION OF THE WITBANK COALFIELD

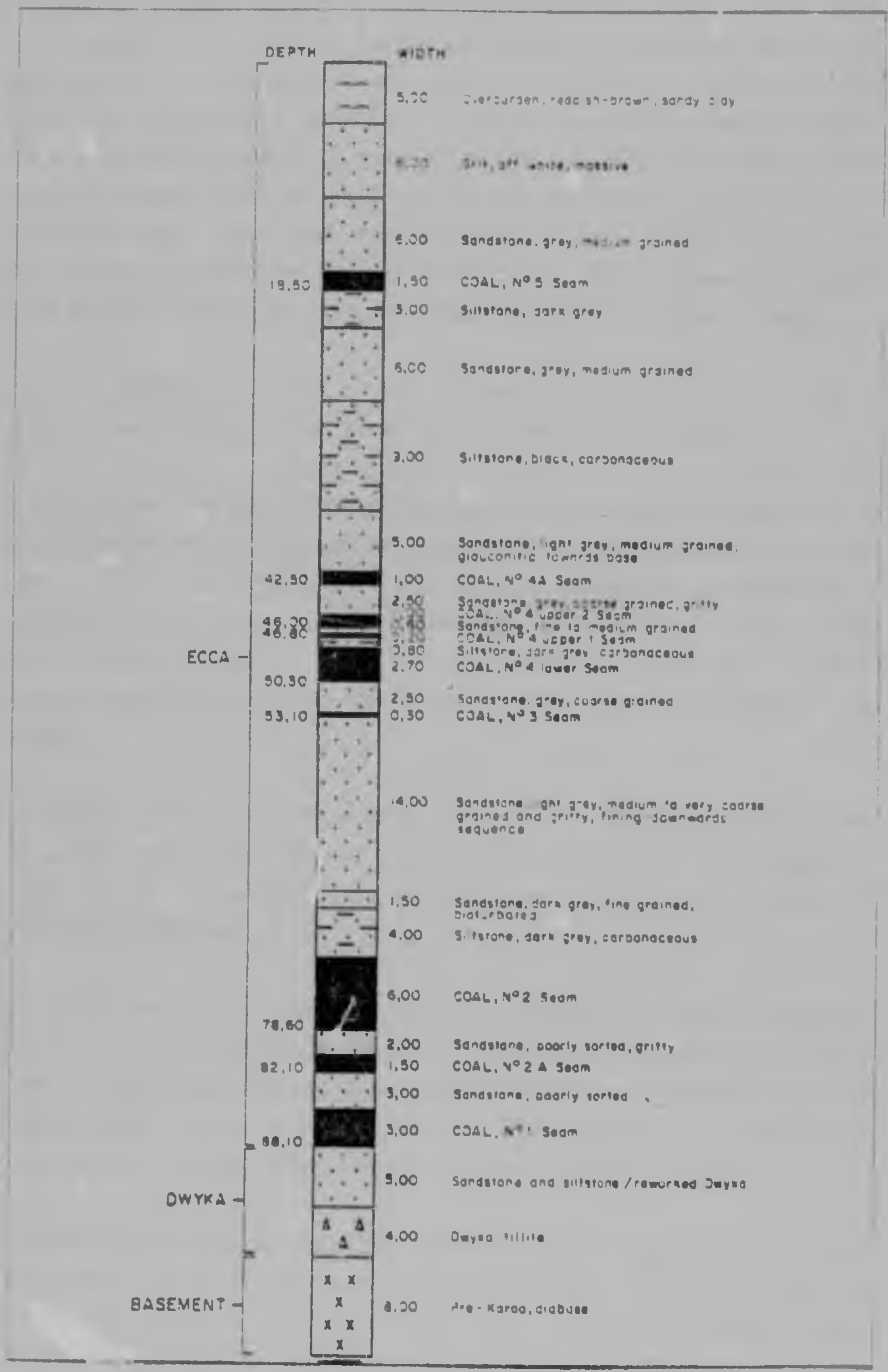


FIGURE 2.2

GENERALISED STRATIGRAPHIC COLUMN

coalfield. Further to the south the seam is much thinner. This seam is situated very close to the Dwyka tillite; it lies either directly on this tillite or is separated from it by up to 2,5 metres of conglomerate, shale or sandstone. In some places the parting between seams Number 1 and 2 is very thin and in restricted areas these seams coalesce; however even in this case the Bottom Seam retains its characteristics. The seam may be absent at some places because of the uneven pre-Karoo floor or pinching out against Basement "highs" and possible erosion prior to the deposition of the Number 2 Seam.

Coarse grained grits form the interburden between Number 1 and 2 seams, and partings within these seams often consist of grit.

The Number 2 seam is the main seam of the Witbank Coalfield. It is a composite seam, with partings and inferior quality coal occurring often within the seam. At the base it comprises approximately 1 metre of bright coal, which is usually followed by dull coal with alternations of bright coal. Locally, where a parting near the base becomes sufficiently thick, the lower seam is termed the Number 2A seam.

Number 3 seam rarely exceeds 1 metre in thickness, and is of no economic importance because of its relative thinness and stratigraphic position. It is separated from seam Number 2 by 10 to 12 metres of shale and sandstone.

Seam Number 4 shows more variation in thickness and quality than any other seam in the Witbank Coalfield. The seam is usually divided into two or three parts, named from top to bottom seam Number 4 A, the Upper Seam Number 4 and the Lower Seam Number 4. Thickness of the parting between seams Number 3 and 4 is about 6 metres.

Because of its high stratigraphic position, Seam Number 5 has been subjected to more denudation than the other seams and occurs in isolated areas of the coalfield. Generally this seam contains more bright coal over its entire thickness than any other seam in the coalfield. The average thickness of Number 5 seam is 1,8 metres and it consists mainly of very bright coal at the base and of a dull upper portion with very bright streaks. The seam is separated from Number 4 Seam by 6 to 14 metres of sandstone and shale.

2.3 Structure

The Witbank Coalfield consists of a shallow basin of sediments deposited on an uneven floor of metasediments and metavolcanic rocks. The sediments rarely dip at angles greater than 5°, although dips tend to increase in the vicinity of Basement "highs". These steeper dips are probably due to differential compaction of the sediments around these "highs". Folding is minimal, with the only known occurrences being monoclinal rolls.

Up to the present, no faults of any considerable magnitude have been encountered in the mine workings of the Witbank Coalfield, and the numerous borehole sections indicate the probability of an equal freedom from disturbance throughout the area. Such faults as have been noted range in throw from a few centimetres to half a metre.

Igneous dykes are also comparatively rare in the northern part of the coalfield, and are usually quite small varying in width between 60 centimetres and 5 metres.

Other structural disturbances are purely of sedimentological origin. Floor rolls are common and probably represent sand deposition in a braided stream type environment, which are very difficul[illegible] predict.

CHAPTER 3

TOPSOIL AND SOFT OVERBURDEN REMOVAL

3.1 Introduction

The first operation conducted during the exploitation phase of surface mining is the removal of soft overburden or topsoil.

According to the South African Mines and Works Act (Act number 27 of 1956), Regulation 5.12.3 ".....all topsoil removed at any opencast mine for the purpose of exposing, working or searching a mineral deposit, shall be deposited at a specially selected site for replacement as topsoil during rehabilitation of the disturbed surface......"

Regulation 5.11 (e) of the Mines and Works Act states that "topsoil" means all cultivable soil material that can be removed mechanically to a depth of one metre without blasting.

In selecting a particular stripping method the primary objective is to remove the material at the least possible cost. Accomplishment of this goal requires the careful consideration of many factors including geological and topographic information, proximity of the stripping operation to waste disposal areas, possible problems of land reclamation, types of suitable equipment available and auxiliary equipment requirements. Normally the size of the coalfield will limit the number of economic stripping methods which need to be considered.

The first part of this section reviews different criteria which are used to classify the excavatability of material, followed by a review of possible mining methods and an economic appraisal of the various alternatives proposed.

3.2 Excavatability Classification Systems.

A literature survey indicated that several criteria are used to classify the excavatability of materials. (Korhonen et al, 1971; Bieniawski, 1976; Barton, 1974; Jennings, 1973). The following methods are summarized briefly:

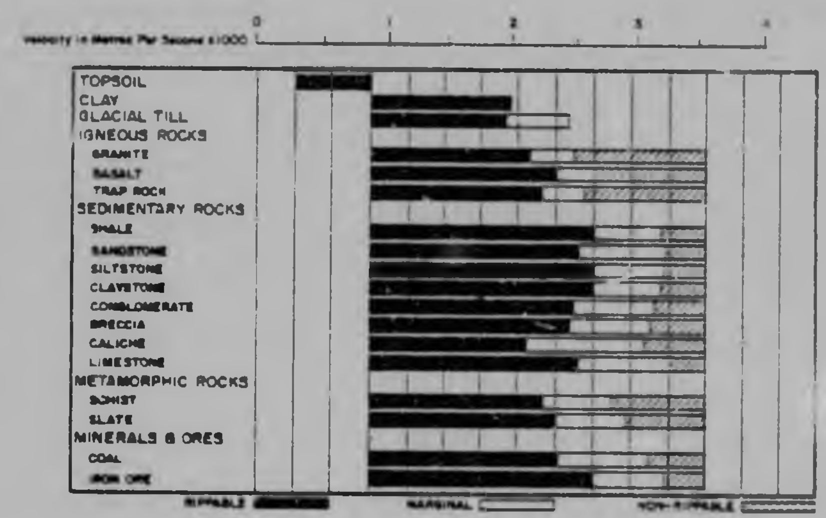
3.2.1 Seismic Velocity

During 1958, the Caterpillar Tractor Company developed the use of the refraction seismograph as an aid for the determination of the class of excavation in terms of the rippability of materials. (Caterpillar Tractor Company, 1978). Weaver (1976) indicates that, by using the refraction seismograph, the seismic wave velocity through various layer of material is measured, from which the degree of consolidation, including such factors as rock hardness, rock stratification, degree of fracturing and degree of weathering can be determined. From this information an indication of the equipment necessary and the method of excavation is obtained. The Caterpillar Performance Handbook (1983) has for many years included information in terms of which various materials have been defined to be rippable, marginally rippable and non-rippable by a range of track-type tractors. Figure 3.1 illustrates the relationship between the ripper performance of a D9L track dozer and seismic wave velocities for different rock types. Weaver (1975) classified the rippability of material based on seismic wave velocities with a heavy tractor as detailed in Table 3.1

The use of seismic wave velocities as a criterion to assess rippability and excavatability is not recommended for the following reasons:

(a) The Caterpillar Tractor Company (1978) states that "the determination whether or not a rock can be ripped is something of an art in itself, and very often a conclusive answer cannot be obtained. Test by trails is the best approach."

(b) Kirsten (1982) compares the seismic velocities for various consistencies of sandstone against drawbar pull for various



Rippers | D9L Ripper Performance
- Multi or Single Shank No.9 Ripper
- Estimated by Seismic Wave Velocities

FIGURE 3.1

RELATIONSHIP BETWEEN RIPPABILITY AND SEISMIC WAVE VELOCITIES FOR CATERPILLAR D9L TRACK-TYPE TRACTOR

TABLE 3.1
VELOCITY RANGES FOR RIPPING WITH A HEAVY TRACTOR

EXCAVATION CHARACTERISTICS	VELOCITY FOR NORMALLY WEATHERED PROFILE	VELOCITY FOR BOULDER SITUATIONS
	m/s	m/s
Easy ripping	450 - 1 200	450 - 900
Hard ripping	1 200 - 1 600	900 - 1 200
Very hard ripping	1 600 - 1 850	1 200 - 1 500
Extremely hard ripping or blasting	1 850 - 2 150	1 500 - 1850
Blasting	> 2 150	> 1 850

classification standards (Figure 3.2) and concludes that "it is evident that the specific limits on seismic wave velocity as given in the TPA Standard Specification (1973) and by Weaver (1975) are at variance and that seismic wave velocity cannot be used as a classifying criterion. This is confirmed indirectly in the proposed SABS Standard Specification (1979) in which the excavation class is defined in terms of the machine characteristics. It is interesting to observe that the seismic wave velocity ranges equivalent to the SAbS Specification are complety different from those given by the TPA Specification and by Weaver."

3.2.2 Geomechanics rating proposed by Weaver (1975)

Weaver (1975) indicates that appart from the seismic wave velocity the following geological factors are likely to influence the assessment of rippability:

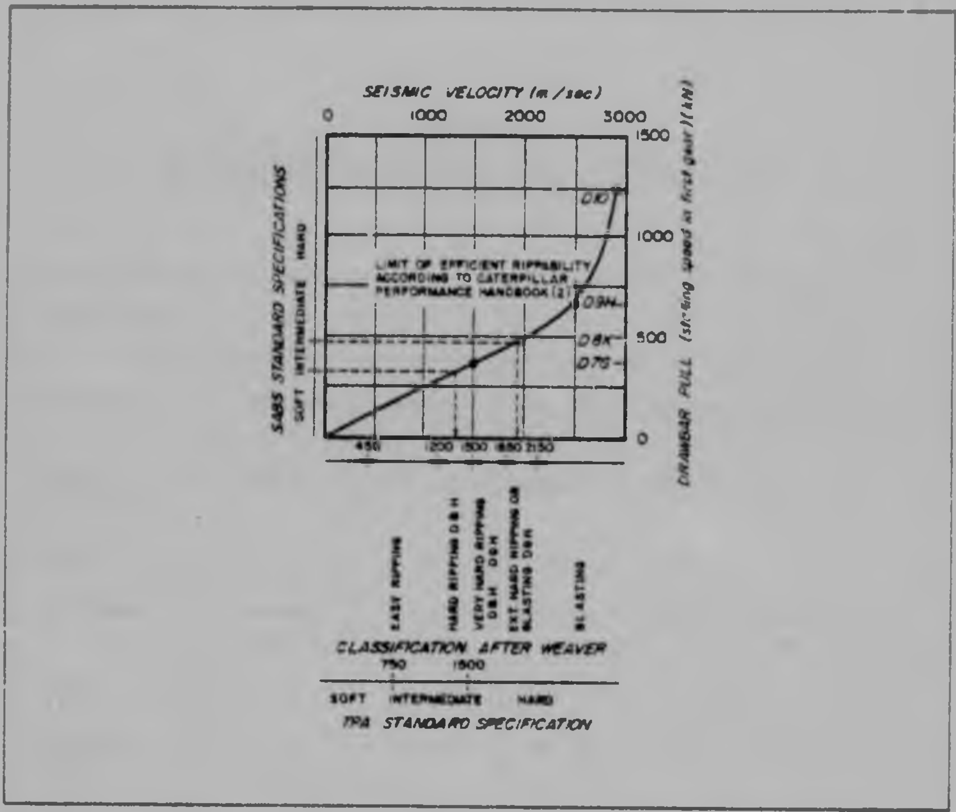


FIGURE 3.2
PLOT OF DRAWBAR PULL AGAINST
SEISMIC VELOCITY AND VARIOUS
CLASSIFICATION STANDARDS

- Rock type
- Rock hardness
- Rock weathering
- Rock structure
- Rock fabric

To incorporate these factors into a single rippability rating factor, Weaver adopted the rock classification system developed by Bieniawski

(1973) and Wickham et al (1972) by assigning ratings to each parameter using a weighted numerical value. The final rock class rating is the sum of the weighted parameters. Table 3.2 summarizes the rippability rating chart as proposed by Weaver.

TABLE 3.2

RIPPABILITY RATING CHART (WEAVER, 1975)

ROCK CLASS	I	II	III	IV	V
Description	Very good rock	Good rock	Fair rock	Poor rock	Very poor rock
Seismic velocity(m/s)	> 2150	2150 - 1850	1850-1500	1000-1200	1200-450
Rating	26	24	20	12	5
Rock hardness	Extremely hard rock	Very hard rock	Hard rock	Soft rock	Very soft rock
Rating	10	5	2	1	0
Joint spacing (m/m)	3000	3000-1000	1000-300	300-50	< 50
Rating	5	5	3	0	0
Joint continuity	Non-continuous	Slight continuous	Separation no gouge	Gouge some gouge	Gouge with gouge
Rating	5	5	3	0	0
Joint gouge	No separation	Slight separation	Separation 1mm	Gouge - 5mm	Gouge - > 5mm
Rating	5	5	4	3	1
Strike & dip orientation	Very un-favourable	Un-favourable	Slightly un favourable	Favourable	Very favourable
Rating	15	13	10	5	3
TOTAL RATING	100-90	90-70	70-50	50-25	25

The advantage of the Geomechanics Classification System is that the effect of various rock characteristics is taken into account in the final rock rating. It is, however, doubtful whether this system, which is based on a summation of the characteristic ratings, could be extended to include the full range of materials from soils through to rocks.

Additional disadvantages which are associated with the summation principle are the inherent limited range of values for the total rating index and the lack of the ability to correlate the nature of the functional dependence of the total rating with any one material characteristic. The limitation on the total rock rating, which follows by definition from the constitutive summation principle, reduces the sensitivity of the rating to any one of the characteristic parameters and this method of rock classification system is not recommended to be used in determining the excavatability of material.

3.2.3 Classification system proposed by Kirsten (1982)

Kirsten (1982) states that the effort to excavate involves two fundamental processess. In principle the material is first broken down in place and then in the second stage it is removed from the excavation. The excavatability of the material may be expressed in terms of the following characteristic parameters which contribute to these processess:

- Strength of parent material
- In situ density
- Degree of weathering
- Seismic velocity
- Block size
- Shape of excavation relative to excavating equipment
- Block shape
- Block orientation
- Joint roughness
- Joint gouge
- Joint separation

In an effort to reduce the number of parameters to be included in the expression for excavatability, Kirsten identified those variables which are dependent upon each other as follows:

(a) The strength, density, weathering and seismic velocity of a natural material are all closely related phenomena, the effects of which could be represented by the Mass Strength Number (Ms)

(b) The block size and relative excavation shape represent the freedom of movement which individual blocks of material have under the action of the excavating appliance. The effect which these parameters have on excavatability can be represented by the Block Size parameter. ($^{RQD}/_{Jn}$)

(c) The effect on excavatability of the block shape and block orientation can be represented by a single characteristic denoting the shape of the block relative to the direction of excavation. This parameter is termed Relative Ground Structure Number (Js)

(d) The effort to separate the individual blocks of material in a tightly fitting arrangement is clearly dependent upon the resistence to movement on the joints. The three parameters of joint roughness, gouge and separation have an effect on this variable and can be combined into a single characteristic: Joint Strength ($^{Jr}/_{Ja}$)

Kirsten (1982) adapted the Norwegian classification system developed by Barton et al (1974) to enable an index to be defined for the entire range of natural materials from the weakest soil to the hardest rock and to allow guidelines in respect of the values for the individual characteristics to be taken from the classification system proposed by Barton.

In terms of the characteristics defined above, Kirsten (1982) expressed the excavatability index (N) as follows:

$$N = Ms\ (RQD/Jn)\ .Js\ .\ (Jr/Ja) \text{ ———— } (3.1)$$

Where Ms = Mass strength number which denotes the effort to excavate the material as if it were perfect, that is, homogeneous, unjointed and dry.

RQD/Jn = Block size number. This term represents the reducing effect which the size of blocks has on the effort to excavate the perfect material.

Js = Relative Ground Structure number which represents the reducing effect caused by the block size and orientation relative to the excavating force on the effort to excavate the perfect material.

Jr/Ja = Joint Strength number. This term represents the reducing effect which the deformability and weakness of the joints have on the effort to excavate the material as a perfect medium.

Every material characteristic in the system proposed by Kirsten should be quantified either empirically by inspection or by rigorous standard testing. Methods to determine the various individual ratings are presented in Appendix 1.

Kirsten (1982) defines the excavation class boundaries in terms of the excavatability index (N) as indicated in Table 3.3

TABLE 3.3
DEFINITION OF EIGHT POINT EXCAVATION
CLASSIFICATION SYSTEM (KIRSTEN, 1982)

MATERIAL TYPE	CLASS	EXCAVATION CLASS BOUNDARIES	DESCRIPTION OF EXCAVATABLITY	BULLDOZER CHARACTERISTICS	
				TYPE	FLYWHEEL POWER (kW)
Soil/ Detritus	1	Less than 0,01	Hand spade	D3	46
	2	0,01 - 0,0999	Hand pick and spade	D4E/D5B	56/78
	3	0,1 - 0,999	Power tools	D6D	104
Rock	4	1,0 - 9,99	Easy ripping	D7G	149
	5	10,0 - 99,9	Hard ripping	D8K	224
	6	100,0 - 999	Very hard ripping	D9H	306
	7	1 000,0 - 9 999	Extremely hard ripping/blasting	D10	522
	8	Larger than 10 000	Blasting	-	-

It should be observed in Table 3.3 that the proposed limits for each category of excavatability are different by an order of magnitude. Some minor inaccuracies or uncertainties in the magnitudes of any of the basic characteristics should not readily result in a change in excavation class unless the index is already close to any of the proposed class boundaries.

To assess the excavatability of soft overburden or topsoil material in the Witbank Area it is recommended to use the classification system proposed by Kirsten (1982) because this system consists essentially of the following elements:

- an expression for the excavatability index in terms of the products of the fundamental engineering characteristics of the ground which affects the process of excavation

- characteristic values for the parameters constituting the excavatability index and

- boundary values for the various excavation class intervals.

The soft overburden of the Witbank Coalfield consist of a reddish brown, sandy clay material, which in core form, is generally not recovered during exploration drilling (Fourie, 1982). The thickness varies from zero, where sandstone outcrops, to depths in excess of 20 metres. The material is often dry, firm and tightly jointed.

By assigning values to the various rock parameters of equation 3.1 then

Ms	=	0,09
RQD	=	35
Jn	=	2,73
Js	=	1,0
Jr	=	1,0
Ja	=	3,0

and N = 0,385
= class 3 (Refer to Table 3.3 above) which can be classified as very easy digging using power tools.

3.3 Equipment Selection

As indicated in section 3.2 the topsoil or soft overburden typical of the Witbank area can be classified as very soft and can be excavated relatively easily without blasting or ripping.

A large selection of mining methods and associated equipment are available for soft overburden stripping. Singhal (1983) compiled an equipment rating chart for topsoil removal which can be used as a guide for equipment selection (Table 3.4).

TABLE 3.4
TOPSOIL REMOVAL VERSUS EQUIPMENT RATING
(SINGHAL, 1983)

1. Should be considered
2. May be considered
3. May be considered under certain conditions
4. May be considered special situation

A. High
B. Moderate
C. Low

		DOZERS	FRONT-END LOADERS	SCRAPERS: ELEVATING	SCRAPERS: PULL-POWER	SCRAPERS: WITH PUSH TRACTOR	DRAGLINE	SHOVEL AND TRUCK	BUCKET WHEEL EXCAVATOR	FRONT-END LOADER AND TRUCK COMBINATION	HYDRAULIC SHOVEL/ TRUCK
Topsoil Thickness (m)	0 - 0,6m	1	1	1	1	1	3			1	1
	0,6 - 1,5	1	1	1	1	1	3	4	3	1	1
Haul distance (m)	0 - 100	1	1	2	2	1	3				1
	100 - 150	2	2	1	1	1	4		3		1
	150 - 300			1	1	1		4		3	2
	300 - 500			1	1	1		4		2	1
	> 500			2	2	2		4		1	1
Flexibility under varied field conditions	Good	A	A	A	A	A	A	A	B	A	A
	Fair	A	A	A	A	A	A	B	B	B	A
	Poor	B	B	B	B	B	A	C	C	C	B

From this table it can be seen that if topsoil to a depth of 1 metre has to be excavated and hauled over a distance of more than 1 kilometre only the following alternatives need to be considered:

- Scrapers
- Hydraulic excavators and trucks
- Front end loaders and trucks

3.4 Financial Evaluation

To assess the various alternatives suggested by Singhal (1983) a typical pit lay-out and haulroad profile were used as the basic input to a computer simulation programme of Barlow's Tractor Division (Blenkinsop and Tredrea, 1982; Fourie, 1982).

Hourly production rates, capital and operating cost estimates were calculated for the following alternatives:

(a) A 31,8 ton capacity rear dump truck (Cat 769 C) loaded by one 280 kW front end loader with a bucket capacity of 5,4 m^3 (Cat 988 B).

(b) A 45,4 ton capacity rear dump truck (Cat 773 B) loaded by one 515 kW front end loader with a bucket capacity of $10,3m^3$ (Cat 992 C).

(c) A 77,1 ton capacity rear dump truck (Cat 777) loaded by one 515 kW front end loader with a bucket capacity of $10,3m^3$ (Cat 992).

(d) A $15,3m^3$ capacity wheel tractor-scraper (Cat 621 B) pushed by one 250 kW track-type dozer (Cat D8L).

(e) A $23,7m^3$ capacity wheel tractor-scraper (Cat 631 D) pushed by one 343 kW track-type dozer (Cat D9L).

(f) A 31,8 ton capacity rear dump truck (Cat 769 C) loaded by one $3,1m^3$ capacity hydraulic excavator (Cat 245 FS).

For each alternative the number of haulers per loader was increased from 1 through to 6. Results of this investigation are presented in Table 3.5, illustrated in Figure 3.3 and detailed in Appendix 2.

TABLE 3.5

PRODUCTIVITY, OPERATING COST AND CAPITAL COSTS FOR VARIOUS SOFT OVERBURDEN MINING METHODS

MACHINES CONSIDERED	NUMBER OF HAULERS PER LOADER/ PUSHER	HOURLY PRODUCTION CAPACITY (BCM)	OPERATING COST PER BCM (CENTS)	CAPITAL COST (R x 10^6)
CAT769C/988B	1	67	235	0,835
	2	128	165	1,180
	3	182	147	1,524
	4	222	145	1,869
	5	243	155	2,213
	6	250	172	2,558
CAT773B/992C	1	101	276	1,483
	2	195	183	1,972
	3	280	157	2,460
	4	350	148	2,949
	5	398	150	3,438
	6	420	161	3,927
CAT777/992C	1	145	208	1,664
	2	274	148	2.334
	3	380	133	3,004
	4	443	137	3,674
	5	466	153	4,343
	6	473	172	5,013
CAT631D/D9L	1	96	221	1,058
	2	194	167	1,587
	3	284	152	2,117
	4	367	147	2,647
	5	445	145	3,177
	6	516	147	3,706
CAT621B/D8L	1	63	237	0,680
	2	125	176	1,000
	3	184	159	1,320
	4	238	153	1,639
	5	288	152	1,959
	6	333	153	2,279
CAT769C/245FS	1	67	142	0,761
	2	128	117	1,105
	3	182	112	1,450
	4	222	116	1,795
	5	243	129	2,139
	6	250	147	2,484

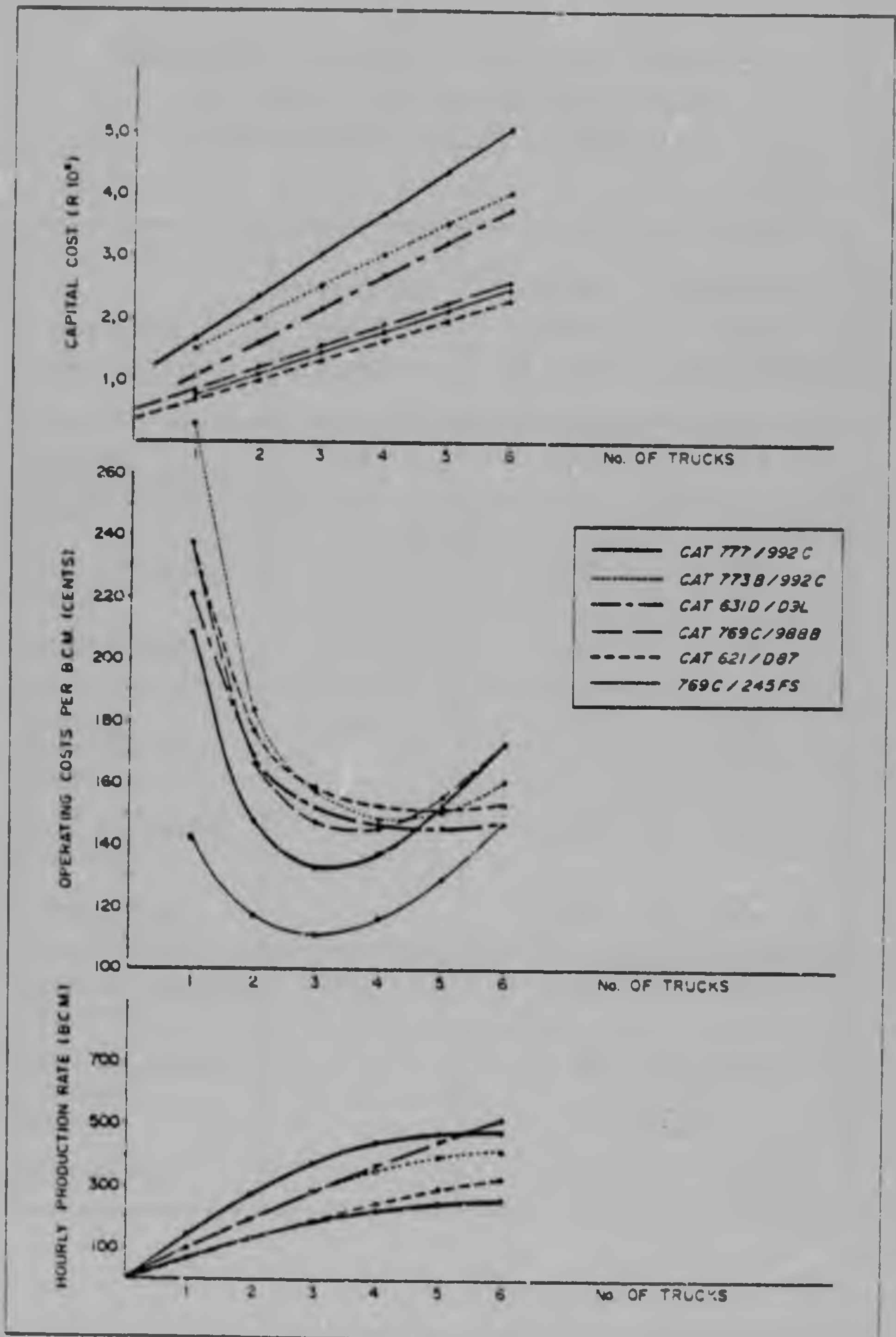


FIGURE 3.3

PRODUCTIVITY, OPERATING COST AND CAPITAL COSTS FOR VARIOUS SOFT OVERBURDEN STRIPPING METHODS

TABLE 3.6

COMPARISON OF CAPITAL AND OPERATING COSTS OF VARIOUS TOPSOIL MINING METHODS FOR DIFFERENT PRODUCTION RATES

MACHINES CONSIDERED	PRODUCTION RATE (BCM/HOUR)	CAPITAL COST (R x 10^6)	OPERATING COST (CENTS/BCM)
Hydraulic excavator/Haul truck	100	0,90	128
Front end loader/ Haul truck		1,05	194
Scraper/Dozer		1,05	220
Hydraulic excavator/Haul truck	200	1,50	112
Front end loader/ Haul truck		1,60	145
Scraper/Dozer		1,60	166
Hydraulic excavator/ Haul truck	300	2,40	115
Front end loader and Haul truck		2,60	155
Scraper/Dozer		2,20	151

3.5 Conclusions and Recommendations

By instituting the excavatability classification system proposed by Kirsten (1982) and by applying rigorous standard tests or empirical quantification by inspection, the excavation class boundaries in terms of the excavatability index (N) can be determined for various soils or rocks in the Witbank Area.

As detailed in section 3.2, the topsoil or soft overburden in the Witbank Area can be defined as very soft and it can be excavated by means of scrapers, hydraulic excavators or front end loaders.

By analysing the results obtained in Table 3.5 for different production rates, as illustrated in Table 3.6, it is evident that the hydraulic excavator and rear dump truck combination has definite financial advantages over the other mining methods analysed. This mining method can also be implemented in mining the relatively thin coal seams and in parting removal, [illegible] standardizing mining methods, operations, equipment and maintenance procedures on a single opencast operation; this results in an overall increase in the mine's productivity and corresponding decrease in unit operating costs.

CHAPTER 4

OVERBURDEN DRILLING

4.1 Introduction

Most surface mines require breakage of the overburden using explosives; to accomplish this, it is necessary to drill a hole into the rock for placement of the explosives.

Drilling is also used in surface mining during exploration for obtaining samples and during development of drainage, slope stability and foundation testing. Only in the exploitation phase of surface mining, however, are unique or specialized drilling methods and equipment employed; the discussion in this section is directed primarily to the latter specialised drilling application.

A variety of rocks may be encountered in drilling. Whether they are sandstone, shale or siltstone is usually of less consequence in selecting a drilling method than how resistant they are to penetration and how they occur geologically.

Both consolidated and unconsolidated materials may have to be drilled. While soils and other loose materials do not require blasting, they may, on occasion, have to be penetrated by a drill when they overlie rock, or when they can be economically moved by explosives. The latter practise can be termed "controlled trajectory blasting."

4.2 Classification of Rock Drilling Methods

A classification of drilling methods, for surface or any other kind of mining, can be made on several bases. These include size of hole, method of mounting, and type of power. The scheme which is considered most logical to employ is based on the form of rock attack, or mode of energy application.

(a) Thermal Attack: Although other principles are known and could be employed, the only method of thermal penetration having practical application today is flame attack with the jet piercer or channeler. Because of its ready capability of forming various shapes of openings, jet burners are used not only to produce blastholes but also to chamber them and to cut dimension stone. This method of rock penetration is only employed on a small number of hard taconite mines in Northern America and Canada.

(b) Fluid Attack: Internal rupture of rock by bursting is attractive, but the end result is more likely to be fragmentation than penetration. Jet action or erosion appears to be the most feasible method, but application is limited.

(c) Sonic Attack: Sometimes referred to as vibratory drilling, this method is presently used as a form of high frequency percussion.

(d) Chemical Attack: The chemical reaction, because of the time element, is not at present considered a practical production method.

(e) Mechanical Attack: The application of mechanical energy to rock can be performed basically in only two ways: by percussive or rotary action. Combining the two actions results in a hybrid method termed rotary-percussion. The mechanical category comprises by far the majority of rock penetration methods in existence today. In surface mining, large percussion drills and roller-bit rotary machines are most widely used.

Studies by Hartman (1968) indicated that the relative cost in soft and medium rock formations with rotary (roller-bit) drilling, is definitely more favourable than other methods, as indicated in Figure 4.1.

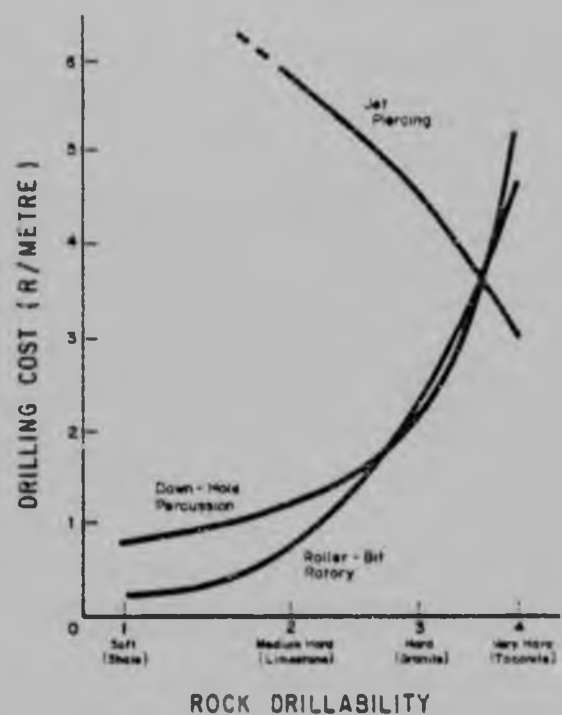


FIGURE 4.1
EFFECT OF ROCK DRILLABILITY ON DRILLING COST FOR VARIOUS PENETRATION METHODS
(HARTMAN, 1968)

Rotary and percussion drilling are sufficiently competitive in the medium-to-hard range to make the choice dependent on the particular circumstances.

Praillet (1983) identified various fields of application of drilling machines as illustrated in Figure 4.2. He calls this the diagram of the four kingdoms. Each kingdom represents the ideal field application of a method and its limits.

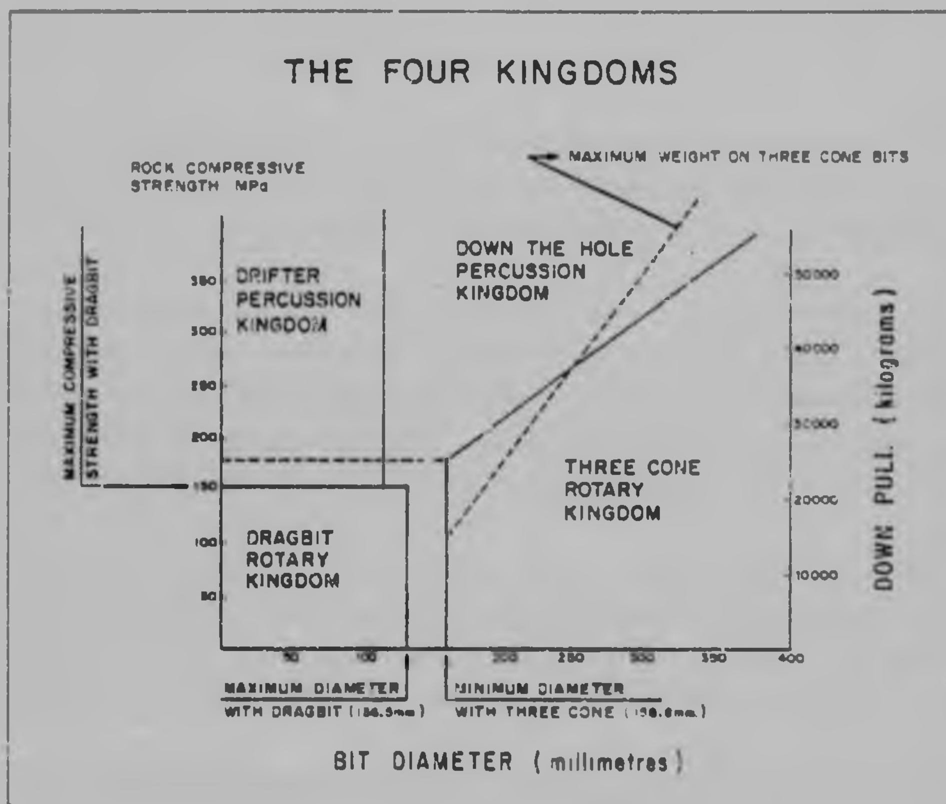


FIGURE 4.2
THE FOUR KINGDOMS OF DRILLING (PRAILLET. 1983)

- Kingdom one is the kingdom of rotary drilling with drag bits.

- Kingdom two outlines the field of application for out-of-the hole drifters.

- Kingdom three represents the ideal field of application for rotary machines using tricone bits, and

- Kingdom four is the ideal field of application for down-the-hole drills.

As most of the overburden in the Witbank Coalfield consists of sandstone, siltstone or shale with an uniaxial compressive strength ranging from 62 to 165 MPa (Steffen, Robertson and Kirsten, 1982) and production output requires large hole drilling, rotary drilling with tricone bits is recommended and will be investigated in more detail.

The drilling effectiveness is highly dependent on the quality of the drill evaluation. This discussion will concentrate on the parameters that must be considered to evaluate drill requirements for a new mine.

The importance of correct drill evaluation extends into other facets of opencast mining. Inadequate fragmentation, the direct result of poor drilling and blasting techniques, results in oversized, hard digging for the excavating equipment, while excessively large drilling units will be under utilized, resulting in an inefficient operation.

The first section of this chapter details the various aspects of drilling to be considered before selecting a particular machine size and the second section will represent an evaluation of different equipment sizes for the various scales of operation in the Witbank Coalfield.

4.3 Factors Influencing Selection of Drilling Equipment

(a) Rotary Drill Penetration Rate

The penetration rate is the main factor that influences drill productivity and has a strong influence on unit costs.

Several years ago, extensive field surveys were conducted in the iron ore industry of North America with a view to relating rotary drill performance to rock properties. These studies (Bauer and Calder, 1967) showed that a good correlation could be obtained between penetration rate and rock uniaxial compressive strength, provided sufficient tests were conducted to obtain a sufficiently meaningful rock strength. Instrumented field tests also indicated that the penetration rate could be linearly correlated with the weight per millimetre of bit diameter and with the rotary speed.

The results of this work can be expressed by the following empirical equation: (Bauer and Calder, 1967 and Bauer 1971).

$$P = (84,48 - 28 \log Sc) \cdot \frac{W}{\varnothing} \cdot \frac{RPM}{17614} \quad \text{——(4.1)}$$

Where P = Penetration rate (m/h)

Sc = Uniaxial compressive strength, MPa.

$\frac{W}{\varnothing}$ = Weight per millimetre of bit diameter, in Kilograms.

RPM = Revolutions of drill pipe per minute.

By applying this formula to typical rock properties encountered in the Witbank Area and by comparing the results with actual production rates obtained on operating mines (Fourie, 1983) it is obvious that this formula gives distorted results as detailed in Table 4.1.

TABLE 4.1

COMPARISON BETWEEN THEORETICAL AND ACTUAL DRILL PENETRATION RATES

Down Pressure	20 000 kg
Bit Diameter	250,825 mm
Rock Compressive Strength	153 MPa
Bit Rotation	100 RPM
Calculated Penetration Rate using equation 4.1	10,55 m/h
Actual Penetration Rate	37 m/h

Praillet (1983), through various field tests determined that the penetration rate with rotary drills in softer formations can be calculated using the following empirical formula:

$$P = \frac{63.8675 \times W \times RPM}{C^2 \times D^{0,9}} \qquad (4.2)$$

Where	P	=	Penetration Rate (metres/hour)
	W	=	Pull down (kilograms)
	RPM	=	Revolutions of drill pipe per minute
	C	=	Rock compressive strength (MPa)
	D	=	Bit diameter (millimetres)

Both Bauer and Praillet (equations 4.1 and 4.2) determined that penetration rate is a function of

- Rock properties i.e. compressive strength
- Bit rotation speed
- Down pressure on bit and
- Bit diameter

The effect on penetration rate caused by each of these items will be discussed in detail below.

(i) Rock properties

The rock property factor in these equations is uncontrollable for a given mine, whereas the rotary speed and pulldown can be varied by the drill operator within the design limits of the machine. The first step in selecting a suitable production drill is therefore to determine the uniaxial compressive strength of the rocks and strata to be encountered during mining operations. Geotechnical laboratory testing on cores from a number of boreholes drilled in the Witbank Area were done by Steffen, Robertson and Kirsten (1982). Although the amount of testing carried out was less than originally programmed, inspection of the rock cores showed that there was little variation in the consistency of the different rock horizons over the test site, and the results obtained (Table 4.2) can be classified as being representatitve of rock strata encountered in the area.

TABLE 4.2
VARIATIONS OF ROCK PROPERTIES FROM LABORATORY TESTS

PARAMETER	MEAN MPa	STANDARD DEVIATION	RANGE MPa
Uniaxial compressive strength, siltstone (MPa)	86	15	61 - 106
Uniaxial compressive strength, sandstone and grit (MPa)	94	25	66 - 156
[illegible]odul[illegible] of elasticity [illegible]stone, [illegible]and grit.	15	15	3 - 41
Ten[illegible] (MPa) Silt[illegible]tone and [illegible]	6 1/2	3	1 - 9
Intact cohesion (MPa) siltstone, sandstone grit	11	4	7 - 15
Effective angle of friction, carbonaceous siltstone	23	6	16 - 27
Effective angle of friction, sandstone and grit.	33	4	28 - 37

Figure 4.3 illustrates penetration rates as a function of rock compressive strength for a rotary speed of 100 RPM and the recommended bit load using formula (4.2) developed by Fraillet (1983).

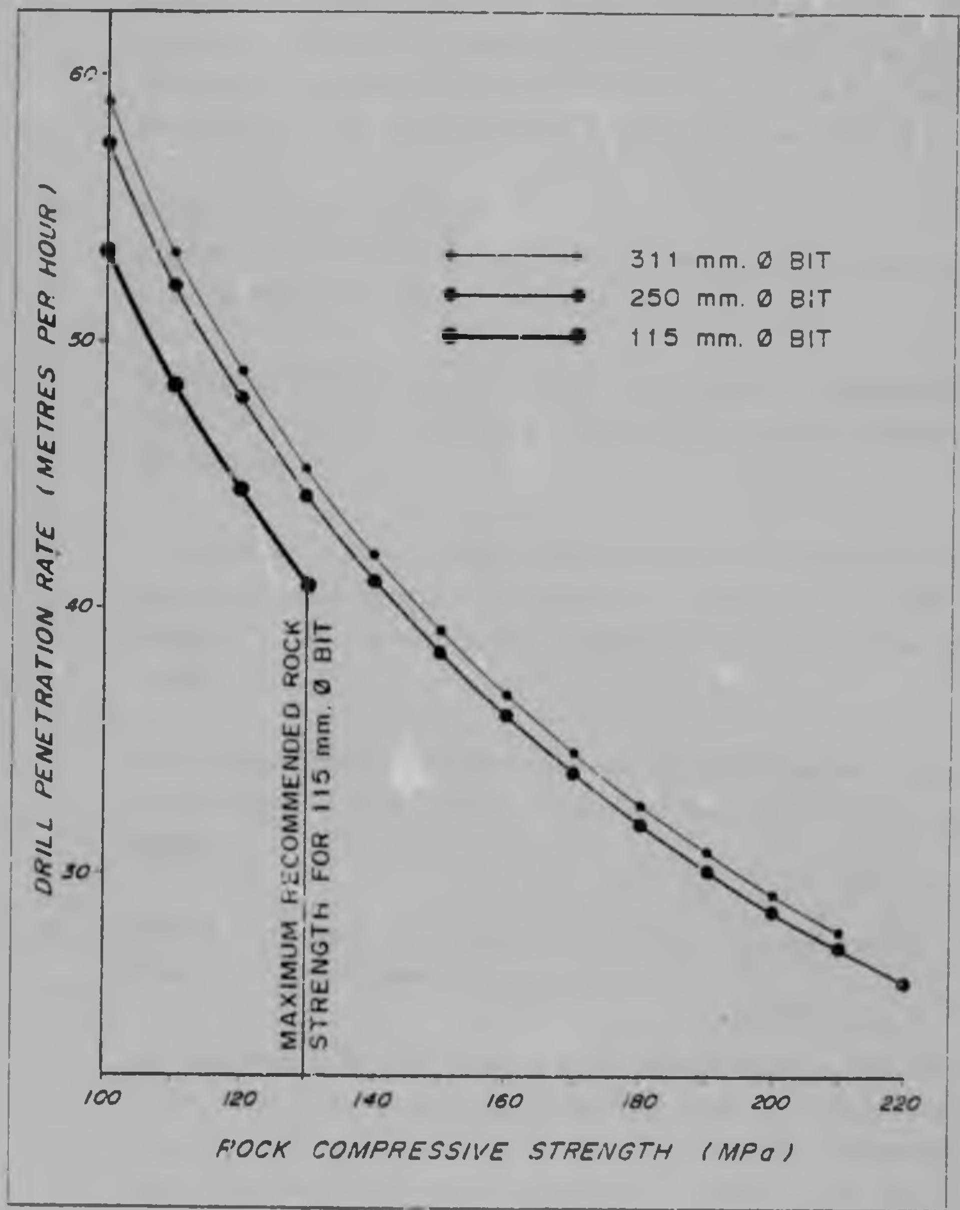


FIGURE 4.3
RELATIONSHIP BETWEEN PENETRATION RATE AND ROCK COMPRESSIVE STRENGTH

(ii) Rotary drill rotation speed

The rotary drive motors turn the drill tool string thus turning the drill bit at the bottom of the hole. This action brings the successive lines of drill bit compacts into contact with the base of the hole. As the rotary speed increases, so the number of contacts increases as well as the penetration rate. The limit of rotary speed is determined by the design parameters of the various drilling machines (Table 4.3).

(iii) Rotary drill pulldown weight

Figure 4.4 illustrates the pulldown and propel arrangement for a large rotary drill. (Bucyrus-Erie 1979).

A portion of the machine weight is applied by the pulldown motor via the pulldown chains, rotary head and drill stems to the drill bit.

It is obvious that the larger the diameter of the drill bit, the larger and more resistant the supporting bearings in the roller cones will be, thus allowing a greater downpull on the bit itself.

An empirical formula to determine the maximum allowable load on any roller bit is calculated by Praillet (1983) to be as follows:

Maximum pulldown (in kilograms) = $0{,}57D^2$ ————— (4.3)
Where D = Bit diameter (in millimetres)

For instance, a 251mm three cone bit will be able to support a maximum pull down force of 35 900 kg, while a 158mm three cone bit will only be able to support 14200 kg. Exceeding those maximum forces would immediately destroy the bit as indicated in Figure 4.5 below. (Bucyrus Erie, 1979).

(iv) Drill bits

The tricone rotary drill bit has evolved from the drag bit and

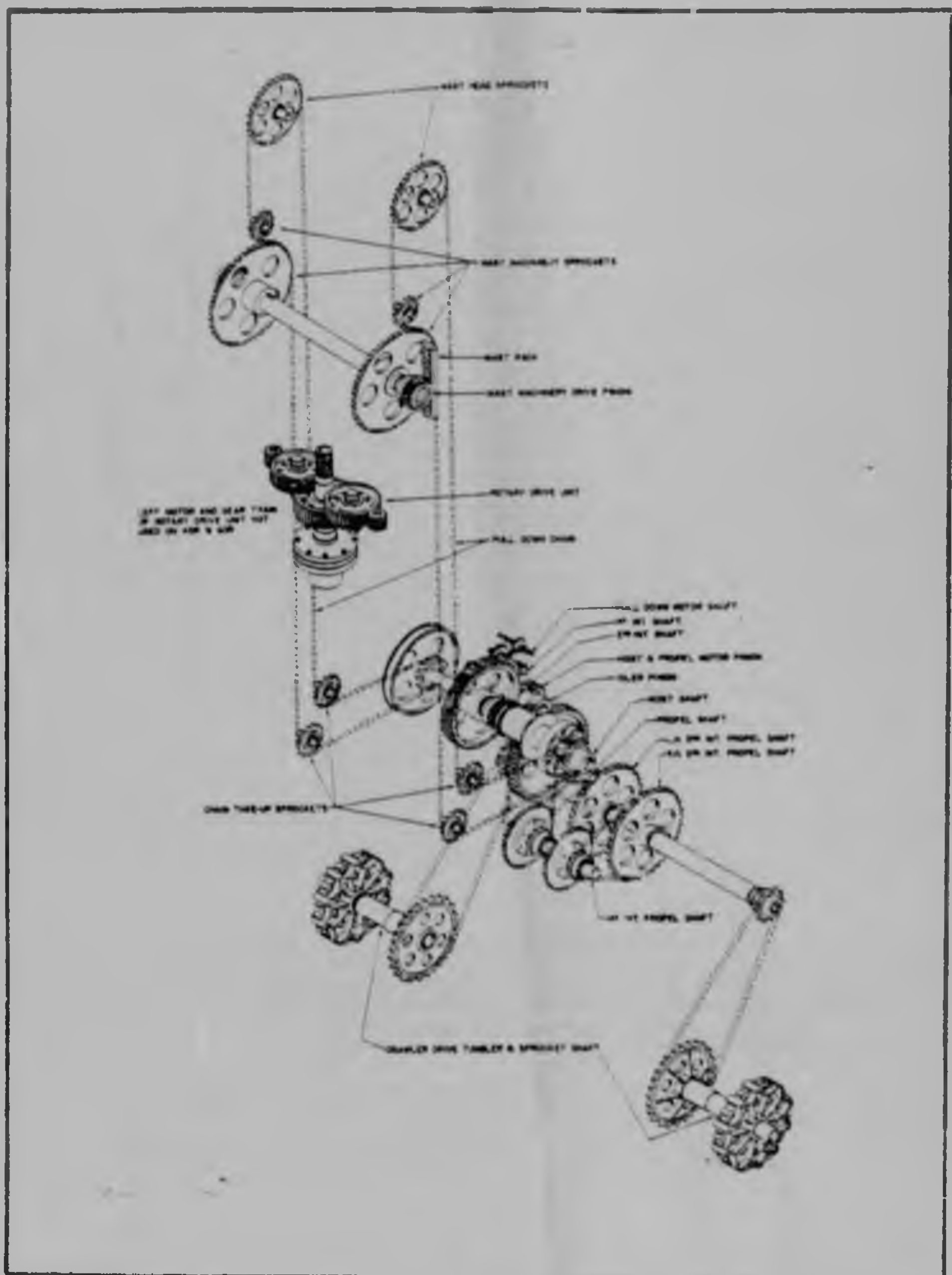


FIGURE 4.4

PULLDOWN AND PROPEL ARRANGEMENT FOR A BUCYRUS-ERIE 60-R ROTARY DRILL

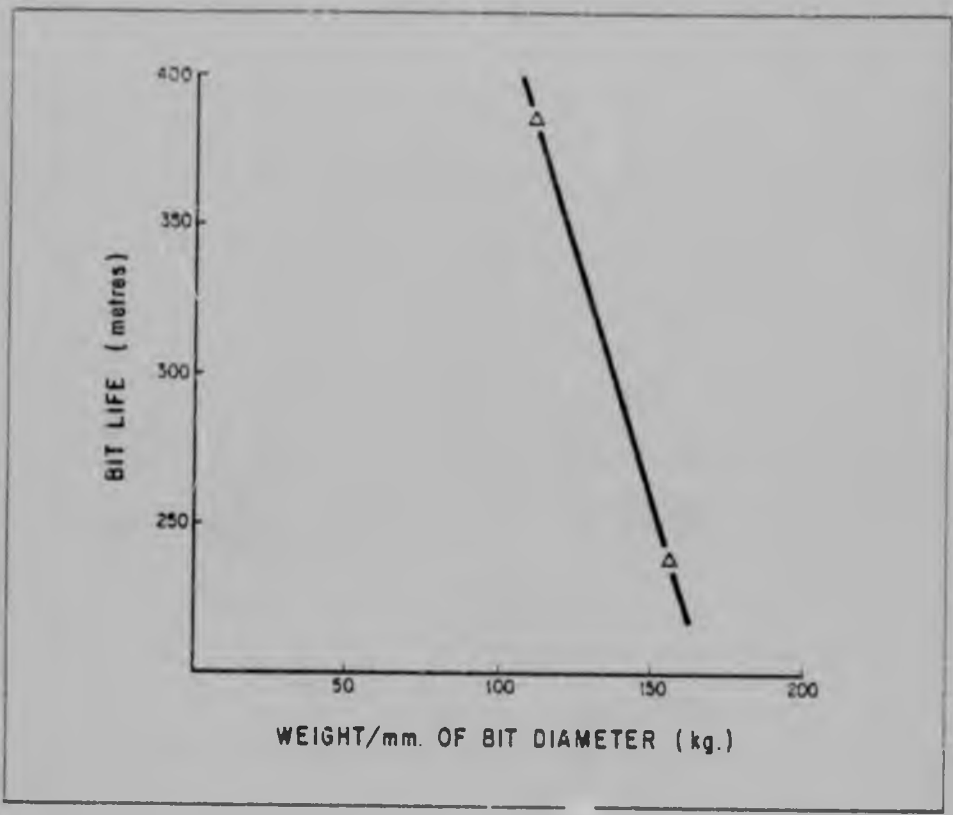


FIGURE 4.5
BIT LIFE VERSUS PULL-DOWN WEIGHT FOR
A 251 mm DIAMETER TRICONE BIT

the two cone bit. Figure 4.6 illustrates the various components of a tungsten insert tricone rotary bit.

Not all tricone bits are designed to drill in any kind of rock, since the main objective is to sink the bit tooth completely in the rock to be drilled. It is obvious that if the rock is very soft, those bits should have very long teeth to penetrate deeper; and conversely, if the rock is very hard, short teeth

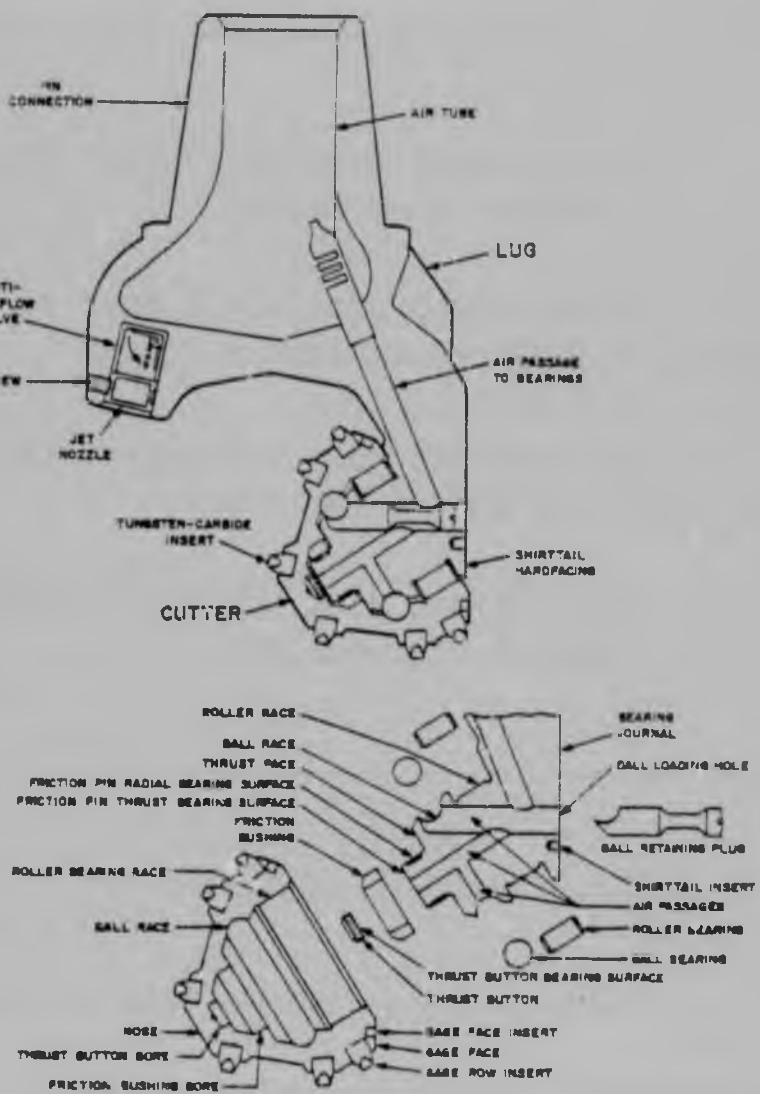


FIGURE 4.6

TRICONE ROTARY BLAST HOLE BIT COMPONENTS should be preferable in order to avoid premature tooth failure. Although it does not always appear very clearly in bit manufacturer's catalogues, there are four classes of tricone bits as illustrated in Figure 4.7 below.

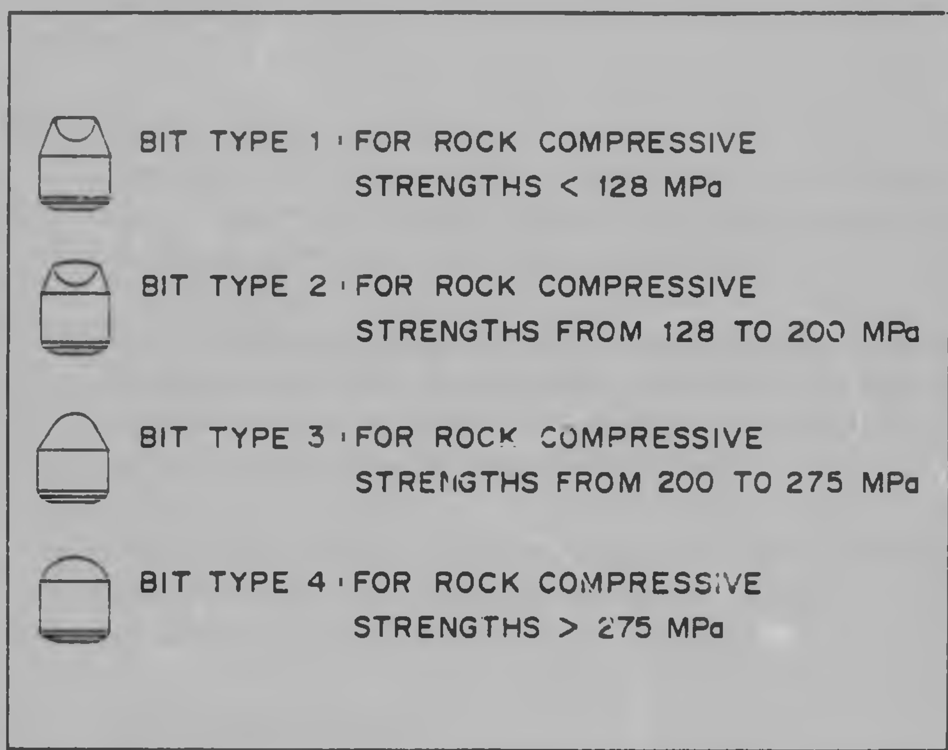


FIGURE 4.7
TRICONE BIT CLASSIFICATION SYSTEM

Praillet (1983) classified these four categories as follows:

Class 1. Bits designed with long steel teeth and for rock compressive strengths up to 128 MPa.

Class 2. These bits are designed with tungsten carbide, relatively long teeth for rocks with a compressive strength ranging between 128 and 200 MPa.

Class 3. Class 3 bits have medium length carbide insert teeth and are to be used for rocks with a compressive strength between 200 and 275 MPa.

Class 4. These bits have very short round tungsten carbide teeth. They are suitable for rocks with a compressive strength greater than 275 MPa.

(b) Rotary Drill Air Requirements

Compressed air used on a blast hole drill serves two purposes. One is to cool the bearing surfaces of the bit and the other is to clean and remove cuttings from the blast hole.

If an insufficient quantity of air is provided, premature failure of the bit will occur. Approximately 20% of the air is forced through the roller cones for cooling purposes by adjusting the air pressure across the bit using the bit nozzles.

The air volume is the primary requirement for bailing cuttings from the hole. Air velocity up the hole is dependent on the air volume per minute as well as the hole annulus.

(c) Drilling Depth Capability

To obviate the need to change or extend drilling rods during normal production drilling activities, it is important to select a drilling unit that is capable of drilling to the required depth within a single pass.

Isopach plans, defining the thickness of overburden and interburden to be encountered during the mining programme, need to be drawn up to determine the minimum, mean and maximum drilling depths required.

(d) Mounting

The method of mounting is determined by the type of surface area to be encountered during drilling operations and the required manoeuvreability of the drilling unit.

Truck or rubber wheel type mounting allows for a high degree of flexibility and manoeuvreability of the drilling rig but is most unsuitable for rugged terrain.

Crawler mounted rotary drills, although more difficult and slower to move, allow a more stable machine, that can be used on any surface area.

Crawler type drills can also be positioned closer to the highwall crest than truck mounted drills, when drilling front row holes.

4.4 Drill Evaluation

The evaluation will start with the review of some of the large diameter rotary blast hole drills available. Table 4.3 lists representative rotary blasthole drills that are available from various manufacturers. This is not a complete list but is meant to show the range of equipment available from which the mining engineer will have to select the right machine for any particular operation.

Three drill sizes (152mm, 250mm and 311mm diameter) will be compared to determine the capital and operating costs for various annual production rates.

4.4.1 Estimated Production Rates

As stated in section 4.3, rotary blasthole drill production rates are dependent on the type of rock formation encountered, the type and size of drill bit used, thrust or pulldown exerted by the drill on the bit and drill compressor size. Other factors that will influence productivity are operator skill, type of terrain, distance between holes and the drills' capability of drilling the required hole with or without changing drill steel (single-pass capability).

TABLE 4.3

MANUFACTURER'S BASIC SPECIFICATIONS OF LARGE DIAMETER ROTARY BLAST HOLE DRILLS

MANUFACTURER	MODEL NO	MOUNT	HOLE DIA mm	PULLDOWN Kg x 10^3	ROTATION SPEED (RPM)	POWER UNIT DRILLING
Bucyrus-Erie	2450-R	Truck	220	16	0-125	Diesel
	40R	Crawler	220	23	0-92,5	Electric
	45-R	Crawler	279	35	0-92,5	Electric
	60-R	Crawler	311	63	0-92,5	Electric
	61-R	Crawler	445	65	0-92,5	Electric
Chicago Pneumatic	T630	Truck	200	15	0-120	Diesel
	C630	Crawler	200	15	0-120	Diesel/Elec.
	T630FF	Truck	200	20	0-190	Diesel
	T7000	Truck	220	20	0-180	Diesel
	C7000	Crawler	220	20	0-180	Diesel/Elec.
	T750	Truck	270	25	0-100	Diesel
	T8000	Truck	311	30	0-130	Diesel
Davey Rousselia	M5CTD	Crawler	152	15	30-240	Diesel
	M8A	Truck	200	15	40-250	Diesel
	M8TD	Truck	210	20	0-220	Diesel
Driltech	D40K	Crawler	222	20	0-200	Diesel
	D40K	Truck	222	20	0-200	Diesel
Gardner Denver	RDC-16B	Crawler	130	8	0-250	Diesel
	GD-60	Truck	250	30	0-94	Diesel/Elec.
	GD-120	Crawler	381	60	0-120	Elec./Diesel
	GD-130	Crawler	432	65	0-120	Elec./Diesel
Ingersoll-Rand	Damco 3000	Crawler	143	10	70-375	Diesel
	Damco 5000	Crawler	165	13	70-375	Diesel
	Damco 6000	Crawler	200	20	70-375	Diesel
	T-4	Truck	200	17	0-100	Diesel
	T-5	Truck	250	30	0-111	Diesel
	DM-4A	Crawler	200	22	0-100	Diesel
	DM-6	Crawler	270	45	0-100	Diesel
	DM-7	Crawler	311	45	0-100	Diesel
Joy (Robbins Division)	RR10-S	Crawler	270	30	29-156	Diesel
	RR11-E	Carwler	311	40	32-350	Electric
	RR15-E	Crawler	381	60	0-200	Electric
	H-100	Crawler	270	45	23-154	Electric
	RRT-50	Truck	220	25	25-140	Diesel
	RRT-60	Truck	270	30	25-140	Diesel
	RRT-70	Truck	270	35	25-140	Diesel
Marion	M-4	Crawler	311	50	0-115	Electric
	M-5	Crawler	381	60	0-115	Electric
Reed	SK-15	Crawler	160	10	0-330	Diesel
	SK-35	Truck	200	15	0-135	Diesel
	SK-40	Crawler	220	20	0-135	Diesel
Schramm	C42-B	Crawler	143	7	0-113	Diesel
	C42H-B	Crawler	143	7	0-113	Diesel
	T64H-B	Truck	171	15	0-43	Diesel
	C64H-B	Crawler	171	16	0-86	Diesel
	T66-B	Truck	200	14	0-86	Diesel
	C66-B	Crawler	200	14	0-86	Diesel
	T985H-A	Truck	200	19	0-86	Diesel
	C985H	Crawler	200	24	0-95	Diesel
	C9120	Crawler	220	24	0-95	Diesel
Portadrill	TG-600	Truck	200	17	0-145	Diesel
	TM6	Truck	200	20	0-130	Diesel
	CM6	Crawler	200	25	0-130	Diesel

As indicated in Table 4.2, the uniaxial compressive strengths of overburden strata, typical of the Witbank Area, range from 61 MPa (siltstone) to 156 MPa (sandstone). To ensure that the selected drilling unit can drill in all kinds of rock to be encountered, it will be necessary to select the rock type in the proposed opencast limits with the highest compressive strength as the basis of design.

The total annual drill production rate is also dependent on the following factors:

- Drill patterns
- Drill availability and utilization

(a) Drill Patterns

The basic equations (Explosives Today, 1972) to determine the drilling pattern for a given powder factor are as follows:

$$B \times S = \frac{r.\ Mc}{k} \qquad \text{——————} \quad 4.4$$

and

$$Mc = \frac{XD^2}{1{,}27} \qquad \text{——————} \quad 4.5$$

Where

B	=	Burden (metres)
S	=	Spacing (metres)
r	=	Ratio of charge length to bench height - normally 0,6
H	=	Bench height (metres)
L	=	Charge length (metres)
Mc	=	Explosive charge density (g/m)
k	=	Powder factor (g/m^3)
X	=	Explosive density (g/cc)
D	=	Blasthole diameter (mm)

From equations 4.4 and 4.5 the required drill pattern for various blast hole diameters can be calculated to obtain a given powder factor.

If it is assumed that slurry explosives will be used at a density of 1,15 g/cc, the respective drill patterns for the various drill sizes to yield a powder factor of 0,5 kg/BCM will be as calculated in Table 4.4.

TABLE 4.4
DRILL PATTERNS FOR DIFFERENT DRILL SIZES
TO YIELD A POWDER FACTOR OF 0,5 Kg/BCM

	DRILL HOLE DIAMETER (mm)		
PARAMETER	152	250	311
r	0,60	0,60	0,60
X	1,15	1,15	1,15
D^2	23 104	62 500	69 721
k	500	500	500
B x S	25,11	67,90	105,10
B (m)	5	7,90	9,80
S (m)	5	8,60	10,80

(b) Drill Availability and Utilization.

A very important factor to be considered when calculating equipment requirement is the actual operating time of the unit. This operating time is dependent on machine availability and machine utilization.

The following equations can be used to obtain the percent availability and utilization for the drill fleets to be evaluated.

$$\% \text{ Availability} = \frac{T - M}{T} \times 100\% \quad (4.6)$$

Where T = Possible available time

M = Maintenance time

$$\% \text{ Utilization} = \frac{O - D}{O} \times 100\% \quad (4.7)$$

Where O = available operating time

D = operating delays

(c) Annual Production Rates

By using the results obtained in Table 4.4 and formulae 4.2; 4.4 4.6 and 4.7 the annual production rates for the different drills under consideration can be calculated as shown in Table 4.5

4.4.2 Financial evaluation

(a) Drill Operating Costs

The estimated hourly operating cost for the three drills types under consideration is presented in Table 4.6.

The labour cost listed is based on a drilling crew consisting of a drill operator and a helper as well as indirect labour costs such as supervision and holiday allowances.

The use of either electric or diesel-electric power for operating the drill depends on the nature of the operation. It is highly recommended that electric power be utilized when the drill is required to cover relatively short distances between drill-hole patterns since the diesel-electric generator is not required, repair and overhaul costs are thus reduced and the current high costs of petroleum fuel are avoided. If, however, the drill is to be moved over long distances between drill patterns, the savings generated with the electric drill might be offset by the time required to change the electric feed cable

$$\% \text{ Availability} = \frac{T}{T} \times 100\% \quad (4.6)$$

Where T = Possible available time
M = Maintenance time

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TABLE 4.5
ANNUAL DRILL PRODUCTION RATES

PARAMETER	DRILL HOLE SIZE (mm) 152	250	311
Drill hole pattern (m)	5 x 5	7,9 x 8,6	9,8 x 10,8
(BCM per metre hole)	25	67,94	105,84
Compressive Strength of Rock (MPa)	156	156	156
Maximum penetration rate (m/h)	37,73	65,29	271,4
Ideal penetration rate (m/h)	35,13	36,93	37,73
Hourly production rate (BCM)	878,25	2509,02	3993,34
Possible available time per year (hours)	7200	7200	7200
% Availability	80	80	80
% Utilization	80	80	80
Operating hours per year	4608	4608	4608
Annual maximum drill production (BCM x 10^6)	4,05	11,56	18,40

TABLE 4.6
ESTIMATED HOURLY OPERATING COSTS FOR VARIOUS OVERBURDEN DRILL SIZES

		DRILL SIZE		
		115mm	250mm	311mm
Labour	(R)	3,15	3,15	3,15
Electric Power	(R)	6,00	8,40	12,00
Repairs	(R)	19,00	21,83	25,00
Bits	(R)	28,10	18,46	34,30
Total hourly operating cost	(R)	56,25	51,84	74,45
Capital cost	(R)	580 000,00	974 000,00	1 300 000.00
Expected operating life (hours)		50 000,00	70 000,00	70 000,00
Depreciation cost	(R)	11,60	13,91	18,57
Total hourly owing & operating cost	(R)	67,85	65,75	93,02
Hourly Production Rates (BCM)		878,25	2 509,02	3 993,34
Drilling cost per BCM	(cent)	7,726	2,62	2,33
Unit drilling cost as % of lowest		332	112	100

or provide an auxiliary moving unit. For this study it is assumed that only electric power will be used to operate the various drill units under investigation.

The repair and maintenance costs include repairs due to breakdowns as well as those for preventative maintenance.

The related equipment costs are a catchall group which include such things as warehousing and storage costs of spare parts.

Drill bits are the primary consumable material cost in drilling. Bit costs vary with the type of formation to be drilled, the pulldown pressure applied, operator skill, the type of bit used and the rotary speed of the drill. Bit costs are estimated from a typical operating mine with similar rock types and properties (Fourie, 1983).

(b) Drill Capital Costs

Drill capital costs are based on quotes received from leading equipment manufacturers and include such items as delivery, erection and commissioning fees. (Fourie, 1983).

TABLE 4.7
INITIAL CAPITAL COST ESTIMATES -
ROTARY DRILLS

DRILL SIZE	152mm	250mm	311mm
No. of units required	5	2	2
Capital cost per unit	R580 000	R974 000	R1 300 000
Total capital cost	R2 900 000	R1 948 000	R2 600 000
Initial capital as % of lowest	149%	100%	133%

4.5 Conclusions and Recommendations

Figure 4.8 illustrates the results calculated in Tables 4.6 and 4.7.

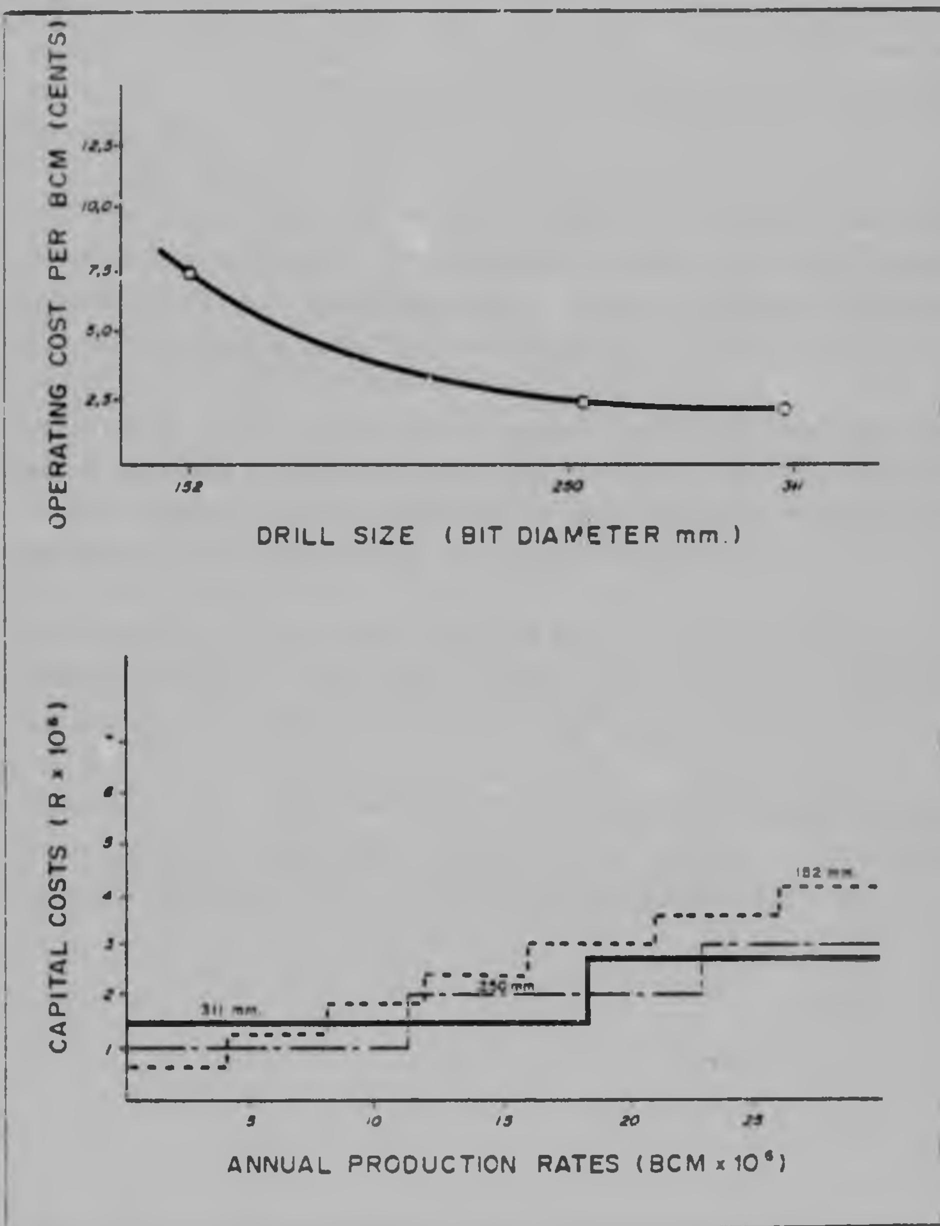

FIGURE 4.8
CAPITAL AND UNIT OPERATING COSTS
FOR VARIOUS DRILL SIZES

From this figure it is evident that the small drilling unit (152 mm diameter drill bit) has limited application in strip mining operations in the Witbank Area. The only advantage of this machine is that it provides a slightly lower capital cost for smaller mines where the overburden stripping requirements are less than 4 million cubic metres per annum. This saving is however outweighed by the higher unit operating cost.

250 mm rotary drills are in general use in the Witbank Area by various mining companies. The operating cost per cubic metre drilled is some 290% less than that of the 152 mm machine and 12% more than the unit drilling cost of a 311 mm drill.

As a result of the large drilling capacity of the 311 mm unit, this unit is only used on the very large opencast mines and even then, the limited number of units required for a single operation can result in severe production losses due to major drill breakdowns.

It is therefore recommended to select 250 mm rotary blasthole drills as they provide the mining operation with greater flexibility, reliability and will minimize drill moves from one drilling area to another.

These units can be selected to provide up to 65 000 kg of pull down force which is more than adequate for penetration of the hard sandstone and siltstone layers encountered in the Witbank Coalfield.

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CHAPTER 5

OVERBURDEN STRIPPING

5.1 Introduction

The continued increase in coal consumption throughout the world, coupled with diminishing shallow coal reserves, is forcing surface mines to greater depths; consequently, stripping operations are becoming a larger portion of the total operating cost and are therefore ever more critical.

Historically, to combat increasing costs and deeper depths, larger equipment was automatically selected as the "obvious" solution. However, a more reliable and efficient system may consist of smaller equipment that has greater flexibility and proven production capabilities.

The size of some stripping equipment, particularly large draglines and stripping shovels, has reached a plateau in the last 10 years because of the economics of utilizing very large machines. The main limitations appear to be a lack of suitable applications for this very large equipment, coupled with lack of flexibility and serious maintenance and transport problems. The economic savings inherent in larger machines have been at least temporarily outweighed by operating problems.

It is thus important, when selecting stripping methods and equipment, that all types and sizes of available equipment be carefully considered to include capital and operating cost estimates and long range economic comparison before choosing the "obvious" method and equipment.

A large selection of mining methods and associated equipment are available for overburden stripping. Singhal (1983) compiled an equipment rating chart (Table 5.1) which can be used as guide in selecting alternatives to be investigated in more detail for a specific project.

TABLE 5.1
EQUIPMENT RATING : OVERBURDEN REMOVAL
(SINGHAL, 1983)

1. Should be considered
2. May be considered
3. May be considered under certain conditions
4. May be considered special situation

A. High
B. Moderate
C. Low

		DRAGLINE	SHOVEL	SHOVEL & TRUCK COMB	FRONT-END LOADERS	DOZERS	FRONT END LOADER & TRUCK COMB.	SCRAPERS: BUCKET WHEEL & EXCAVATOR	SCRAPERS: ELEVATING	SCRAPERS: FULL POWER	SCRAPERS: WITH PUSH TRACTOR	HYDRAULIC SHOVEL
Thickness	0-10m	2	1	1	1	1	1	1	1	1	1	1
	10-20m	1	1	1	1	1	1	1	1	1	1	1
	20-30m	1	1	2	3	3	3	2	2	2	2	2
	30m	1	2	3	4	4	4	2	3	3	3	3
Characteristics	Poor Fragmentation (Blocky-Large Breakout Force)	3	1	1	3	1	3	-	-	-	-	1
	Moderately Blocky	2	1	1	2	1	2	-	2	2	2	1
	Good Fragmentation (Low Breakout Force)	1	1	1	1	1	1	3	1	1	1	1
	Unconsolidated	1	1	1	1	1	1	1	1	1	1	1
Transport Distance	15 - 30m	1	1	-	1	1	-	2	-	-	-	1
	30 - 100m	1	-	2	1	1	-	1	-	-	-	1
	100 - 150m	2	-	1	2	2	3	1	3	3	3	1
	150 - 300m	-	-	1	-	-	1	-	1	1	1	2
	300m	-	-	1	-	-	1	-	1	1	1	1
Coal Seam Support Characteristics	Good(Hard)	1	1	1	1	1	1	1	1	1	1	1
	Moderate	1	2	2	1	1	1	2	1	1	1	1
	Poor(Soft)	1	4	4	2	2	2	4	1	1	1	1
Segregation Capacity	-	A	C	A	A	B	A	A	A	A	A	A
Production Capability	-	A	A	B	A	A	B	A	A	A	A	A
Flexibility Under Varied Field Conditions	Good	A	A	A	A	A	A	B	A	A	A	A
	Fair	A	B	B	B	A	A	B	A	A	A	A
	Poor	A	C	C	C	B	B	C	B	B	B	B
Mobility	-	B	B	B	A	A	A	C	A	A	A	A

Overburden overlying the number 4 coal seam that separates the numbers 4 and 2 coal seams in the Witbank Area consists of a hard sandstone and carbonaceous siltstone. The total thickness ranges from 6 metres to more than 40 metres depending on topography and localized geology.

By using the equipment rating chart proposed by Singhal (1983) the following mining methods will be investigated:

- Draglines
- Electric Shovels and Trucks
- Hydraulic Excavators
- Bucket Wheel Excavators

The first section of this chapter details the various aspects to consider when selecting any of these mining methods and in the latter section a financial appraisal of the various methods proposed is presented.

5.2 Draglines

Rather than increasing the size of draglines as has been prevalent in recent years, the trend is now towards refining and improving the existing intermediate size machines and constructing longer booms with smaller buckets to reduce costly rehandle of spoil. This is particularly true where deeper stripping is involved.

Dragline bucket sizes range from less than $1m^3$ to $138m^3$ and boom lengths up to 122 metres. The selection and specification of a particular dragline for a particular application has been the topic of many technical papers and presentations. (Mooney, 1979; Morey, 1979; Speake et al, 1977; Stewart, 1982).

Figure 5.1 (Bucyrus Erie, 1976) is a typical dimensional diagram of a walking dragline, the major dimensions being designated by letter identification as detailed in Table 5.2.

The primary dimensions required for dragline selection are:

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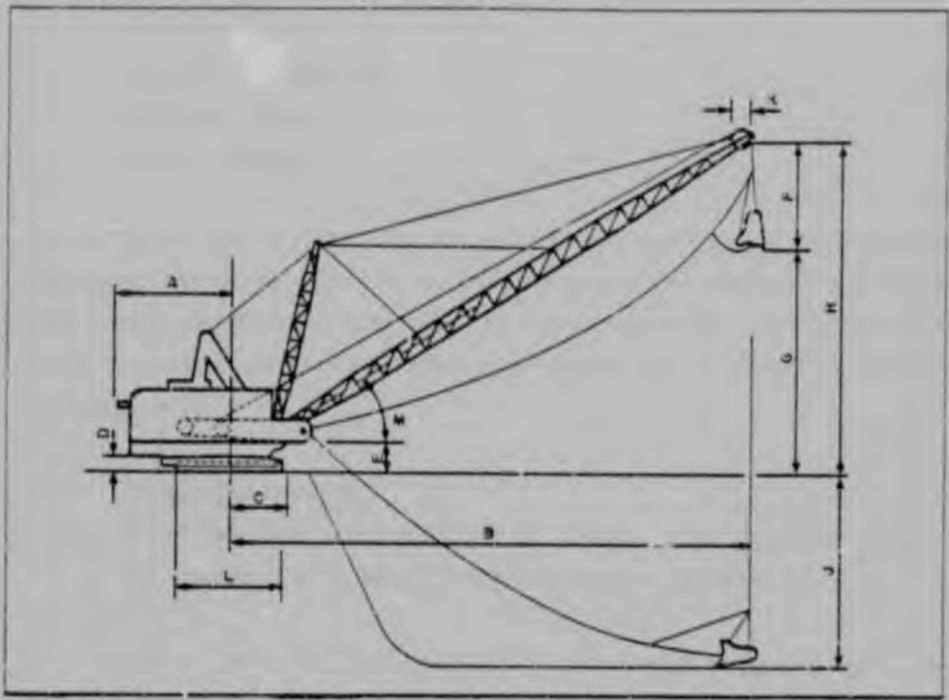

FIGURE 5.1
DIMENSIONAL DIAGRAM OF A
WALKING DRAGLINE (BUCYRUS ERIE, 1976)

TABLE 5.2
DRAGLINE DIMENSION TERMINOLOGY
AS ILLUSTRATED IN FIGURE 5.1

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

- Dragline Reach
- Cut Depth Capability
- Stacking Height
- Bucket Capacity

The first three factors are dependent on the stripping method with associated overburden cut depth, coal thickness and proposed pit width. The bucket capacity is controlled by the planned coal production rate which in turn dictates the required stripping rate, depending on the stripping ratio.

(a) Dragline dimensions

There are two main methods of obtaining the required dragline dimensions, these being graphically or by computation.

Both use the "cut diagram" or simple range diagram as illustrated in Figure 5.2 for either actual measurement or derivations.

Figure 5.3 illustrates a pit cross section with the relevant geometrical parameters being labeled.

The parameters used in the analysis are:-

β	=	Highwall slope with the horizontal in degrees.
θ	=	Spoil pile slope with the horizontal in degrees.
D	=	Overburden depth
OR	=	Dragline operating radius
P	=	Dragline positioning
RF	=	Dragline reach factor
S	=	Spoil pile swell factor
W	=	Pit width
h	=	Height of the spoil pile peak above the top surface of the coal
T	=	Coal thickness

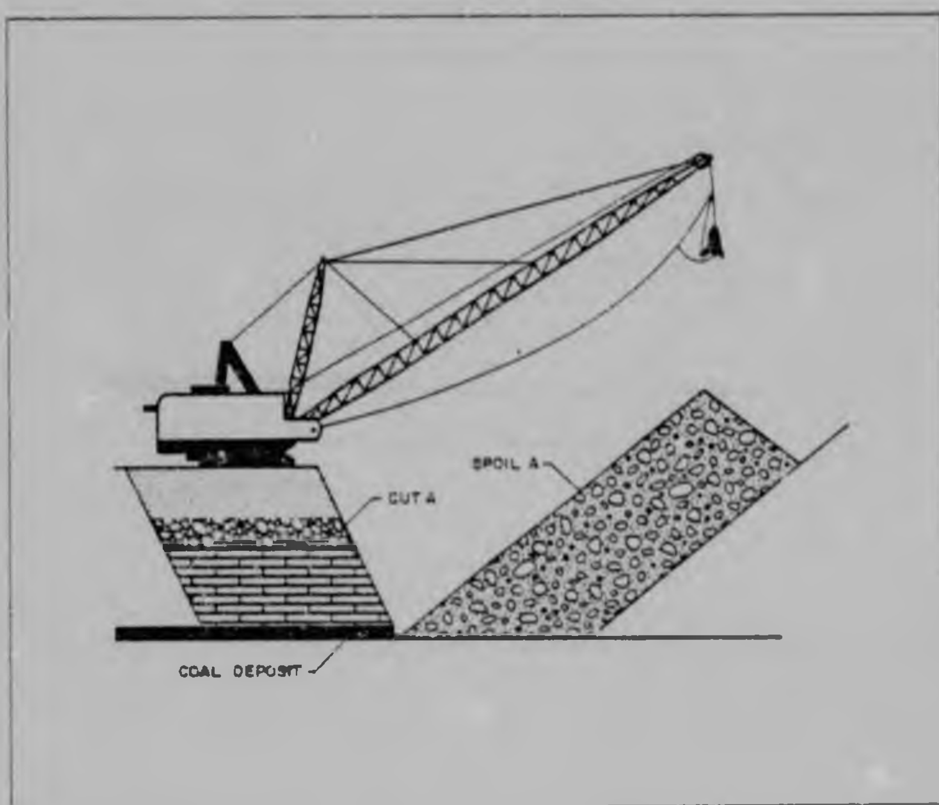

FIGURE 3.2
SIMPLE SIDE CASTING CUT DIAGRAM
OR RANGE DIAGRAM (BUCYRUS ERIE, 1976)

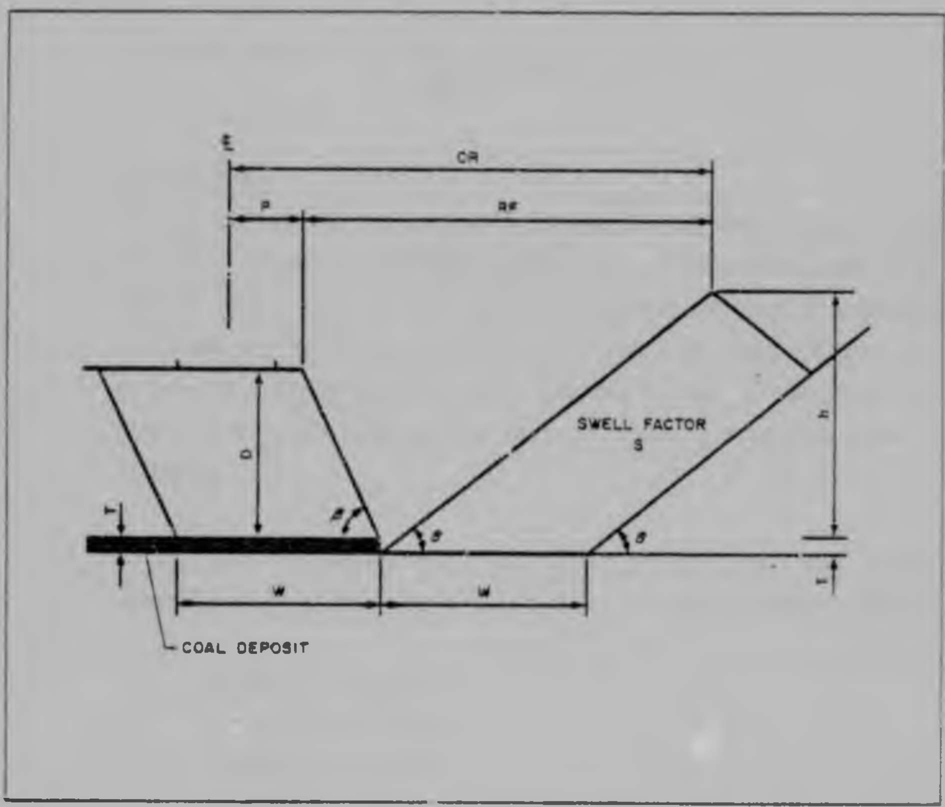

FIGURE 5.3

SIMPLE SIDE CASTING RANGE DIAGRAM SHOWING RELEVANT GEOMETRICAL PARAMETERS

By using the simple casting range diagram indicated in Figure 5.3, Bucyrus Erie (1976) calculated the reach factor (RF) and the stacking height to be as follows:

$$\text{Reach factor (RF)} = \frac{D(1 + S)}{\tan\theta} + \frac{W}{4} + \frac{D}{\tan\beta} - \frac{T}{\tan\theta} \quad (5.1)$$

$$\text{Stacking height} = \frac{RF - D\cot\beta}{\cot\theta} - T - D \quad (5.2)$$

(b) Bucket Size

Having calculated the physical dimensions of the dragline itself, the other equally important parameter to be determined is the bucket size. The bucket size or capacity is usually expressed in terms of "rated cubic metres". The required capacity is primarily dependent on the proposed coal production and overburden ratio, that is the rate at which overburden must be stripped.

Once the stripping rate has been established the bucket capacity can be calculated using data for the following factors:

- Dragline cycle time
- Material swell factor
- Bucket fill factor
- Dragline scheduled operating time
- Dragline availability
- Dragline utility

Having established or estimated these factors the bucket capacity can now be calculated from the following formula: (Bucyrus Erie, 1976)

$$\text{Bucket Capacity} = \frac{Y \times C \times (1 + S)}{O \times A \times U \times B} \quad (5.3)$$

Where Y = Annual overburden stripping requirements

		(bank cubic metres)
C	=	Cycle time - (seconds)
S	=	Swell factor - (decimal fraction)
A	=	Dragline availability - (decimal fraction)
U	=	Dragline Utilization - (decimal fraction)
B	=	Bucket fill factor - (decimal fraction)
O	=	Operating time per year (seconds)

(c) Dragline selection

The two previous sections have outlined the calculation of the parameters necessary to make the machine selection. Before this can be done, the following factors must also be taken into account.

- Maximum suspended load of dragline
- Bucket weight
- Relative density of overburden material to be stripped.

A large range of draglines are available as illustrated in Table 5.3 below. This list only represents the range of machines supplied by Bucyrus Erie, while other manufacturers such as Marion, Harnischfeger and Rapier supply similar units.

This machine selection did not take any account of digging depth or dumping height. In general these factors are less critical. A final selection procedure would be much more detailed and, as such, too lengthy to be fully discussed in this presentation. Various computer programmes, developed by mining companies and consulting engineers are available to assist the mining engineer in selecting a particular machine for a specific operation. (Brian, 1978; Fluor, 1981).

TABLE 5.3

STANDARD MACHINE SELECTION TABLE - DRAGLINES

REFERENCE NUMBER	BOOM LENGTH(m)	BOOM ANGLE(deg.)	OPERATING RADIUS(m)	REACH FACTOR (m)	MAX.SUS. LOAD(kg)
800-W with 12m tub					
1	59	35	55	45,4	58967
2	59	30	57	48,2	56700
3	67	35	61	51,5	52160
4	67	30	64	54,6	49895
5	75	35	67	57,6	47627
6	75	30	70	61,3	45360
7	81	35	72	62,8	43100
8	81	30	76	66,4	40800
1260-W with 15m tub					
9	69	30	66	54,1	86180
10	72	34	66	54,1	86180
11	72	30	68	56,8	81650
12	79	38	68	86,8	81650
13	79	30	75	63,2	72570
14	87	38	75	63,2	72570
15	87	30	81	70,0	58970
1300-W with 15m tub					
16	72	34	66	54,1	102000
17	72	30	68	56,8	97500
18	79	38	68	56,8	97500
19	79	30	75	63,2	88450
20	87	38	75	63,2	88450
21	87	30	81	70,0	81630
1350-W with 16m tub					
22	81	38	73	61,6	108862
23	87	30	84	72,5	95250
24	87	38	78	65,8	102100
25	92	30	89	77,1	90700
26	92	38	82	70,0	97520
27	98	30	94	81,7	83900
28	98	38	86	74,1	92900
1370-W with 18m tub					
29	81	38	73	60,2	140600
30	87	30	84	71,2	127000
31	87	38	78	64,5	138300
32	92	38	82	68,4	129300
33	92	38	82	68,4	129300
34	98	30	94	80.3	113400
35	98	38	86	72,7	124700
1500-W with 19m tub					
36	81	38	73	59ml	150000
37	87	30	84	70,0	136100
38	87	38	89	74,6	129270
39	92	30	52	67,3	140600
40	92	38	82	67,3	140600
41	98	30	94	79,2	122500
42	98	38	86	71,6	133800
1570-W with 20m tub					
43	87	30	74	69,3	170100
44	87	38	77	62,3	181400
45	94	30	91	75,7	156300
46	94	38	84	68,4	170100
47	99	30	95	79,7	142900
48	99	38	87	72,08	156500
49	105	30	100	85,1	129000
50	105	38	92	76,9	143000
2560-W with 20m tub					
51	84	30	84	68,6	192800
52	84	35	80	64,7	201900
53	84	38	77	62,3	208700
54	90	30	89	73,8	181400
55	90	35	84	69,6	1905000
56	90	38	82	66,8	197000
2570-W with 23m tub					
57	87	30	86	69,3	23800
58	87	35	82	65,1	249500
59	87	38	79	62,1	260800
60	94	30	93	75,7	215500
61	94	35	88	71,5	226800
62	94	38	85	[illegible]	238000

5.3 Electric Rope Shovels

Following the same trend as draglines, the bucket size of stripping shovels has been decreasing. The largest existing shovel is a 138 cubic metre machine erected in 1965, yet the largest unit manufactured since that time is a $96m^3$ shovel built in 1969; this represents a reduction in capacity of 31 percent.

The selection of shovels and trucks for most mining applications requires the determination of numbers of units as well as unit size.

Contrary to the selection of strip mining equipment, no constraint is superimposed on the machine dimensions with respect to bank height, spoil area or spoil pile shape. The shovel dumping height and reach are a function of the truck dimensions and method of operation, both of which can be selected. The initial shovel selection process, then, essentially depends on the required production. Thus the number and size of shovel units are first determined with which the number and size of trucks are matched.

(a) Determination of the Required Number and Capacity of Shovels

Figure 5.4 is a typical shovel working dimension diagram, the major dimensions being designated by letter identification. Table 5.4 identifies the terminology for the various dimensions designated by the letter code in Figure 5.4.

Of all these dimensions, those of most significance for the mine operator are the dumping height, dumping radius and operator's eye level which help to describe the size of truck that can be loaded, and the maximum cutting height which helps determine the bench height.

The selection of size and number of shovels to satisfy a given production target is a complex task and depends on the following factors:-

- Maximum daily production required

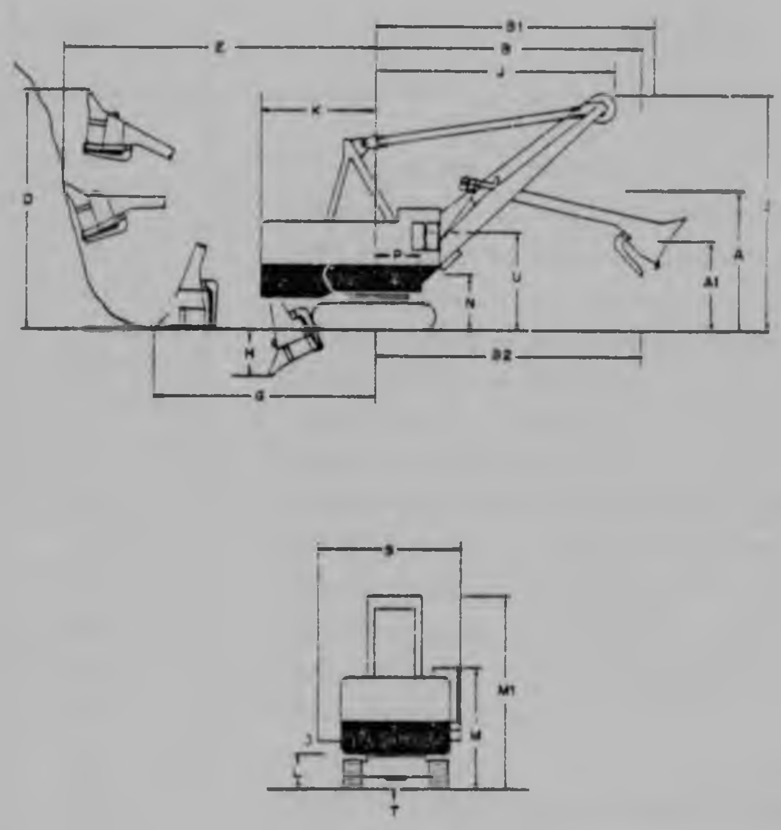

FIGURE 5.4

SHOVEL WORKING DIMENSIONS (BUCYRUS ERIE, 1979)

TABLE 5.4
SHOVEL DIMENSIONS AS ILLUSTRATED IN
FIGURE 5.4

CODE	DIMENSION
A	Dumping height - maximum
A_1	Dumping height at maximum radius - B_1
B	Dumping radius at maximum height - A_1
B_1	Dumping radius - maximum
B_2	Dumping radius at 5m dumping height
D	Cutting height - maximum
E	Cutting radius - maximum
G	Radius of level floor
H	Digging depth below ground level - maximum
I	Clearance height - boom point sheaves
J	Clearance radius - boom point sheaves
K	Clearance radius - revolving frame
L	Clearance under frame - to ground
M	Clearance height top of house
M_1	Height of A-frame
N	Height of boom foot above ground level
P	Distance - boom foot to center of rotation
S	Overall width of machinery house and operator's cab
T	Clearance under lowest point in truck frame
U	Operator's eye level

- Truck size and haul road profile
- Surge in the system
- Shovel operating method
- Overburden characteristics

To translate a required shovel capacity into shovel size, the following factors must be estimated or calculated.

- Digging hours per day or year
- Shovel work cycle
- Dipper fill factor
- Material characteristics and swell factor
- Dipper factor

(i) Calculation of digging hours

In a given period of time the number of hours a scheduled shovel will actually be digging and loading will be determined by the length of any delays. Delays may be broken down into two types; fixed and variable.

Fixed delays are known as to type and duration and are normally a result of method of operation or management. Typical delays that can be classified under this heading are:

- Shift change over time
- Lunch breaks
- Lubrication

The variable type of delay can be anticipated but its exact occurance and duration is unpredictable. Examples of such delays are time spent waiting for trucks, tractor clean-up, moving the shovel, changing teeth and adapters, operating adjustments to the shovels and other minor running repairs. The following scale (Bucyrus Erie, 1979) may be used in estimating the variable delay time for a shovel operation:

Excellent 92% (55 minutes/hour)

Good	83%	(50 minutes/hour)
Fair	75%	(45 minutes/hour)
Poor	67%	(40 minutes/hour)

Generally a "good" value of 83%, or a 50 minute working hour, is taken as an estimate of shovel performance, this figure being based on a good match between haul units and shovel and a good shovel working system such as the double back-up method.

The shovel utilization can now be calculated. The utilization then describes the portion of the available time that the shovel is actually digging.

(ii) The shovel work cycle

The work cycle of a shovel is defined as the time required to make one complete cycle of dig, swing, trip the dipper and return to the bank.

Table 5.5 illustrates estimated work cycle times in seconds for a range of Bucyrus Erie shovels. The chart incorporates the three variables, maximum suspended load at the point sheaves, shovel boom length and material diggability. The swing angle used in compiling this table, was 90°

The other factors which affect the shovel work cycle are:

Suspended load

This includes the dipper weight plus the weight of the loaded material. As the suspended load increases, the work cycle time increases.

Diggability

This expresses the resistance of the material to the dipper passing through the bank. Diggability may be classified as easy, medium, hard or very hard. For the first two classifications, the work cycle time is considered to be

TABLE 5.5

ESTIMATED WORK CYCLE TIMES FOR DIFFERENT SHOVELS OPERATING IN VARIOUS DIGGING CONDITIONS

SHOVEL	MAX.SUSPENDED LOAD (kg)	BOOM LENGTH (m)	WORK CYCLE TIME* (SECONDS)			
			EASY DIGGING	MEDIUM DIGGING	HARD DIGGING	VERY HARD DIGGING
155B	27 200	11,6	25.8	25.8	29.8	34.8
	20 400	15,8	26.2	26.2	N.A.	N.A.
195B	38 600	12,6	26.3	26.3	30.3	35.3
	29 500	17,2	29.0	29.0	N.A.	N.A.
	25 400	21,3	32.0	32.0	N.A.	N.A.
280B	43 100	15,2	28.0	28.0	32.0	37.0
	36 300	24,4	33.7	33.7	N.A.	N.A.
290B	47 600	14,3	28.0	28.0	32.0	37.0
	40 800	24,4	33.7	33.7	N.A.	N.A.
295B	59 000	15,2	30.0	30.0	34.0	39.0
	52 600	21,3	32.0	32.0	N.A.	N.A.
	40 000	25,9	34.0	34.0	N.A.	N.A.

* Based on a 90° swing arc. N.A. - not applicable

unaffected and is used as the standard base for assigning a given work cycle. For hard digging, 4 seconds is added to the work cycle and for very hard digging, an additional 5 seconds is added.

(iii) The dipper fill factor

In addition to the work cycle, the diggability of the material also affects the fill factor of the shovel dipper. The fill factor is the ratio of the actual loose cubic metres loaded in the dipper to the dipper rated capacity. This factor varies according to the ease with which material will flow into the dipper.

An estimate of the typical fill factors that can be expected in various types of digging are as follows:

Easy digging	0,95 to 1,00
Medium digging	0,90 to 0,95
Hard digging	0,80 to 0,90
Very hard digging	0,70 to 0,80

(iv) Material Characteristics

When determining the bucket size of a particular mining shovel, the following material characteristics should also be considered:

- Density of rock
- Swell factor of rock

Swell factor is defined as the weight ratio of a loose (or broken) cubic metre of material to the weight of one cubic metre of rock in-situ.

(v) Shovel and truck size matching

A shovel's productivity is affected very little by the size of the trucks being loaded as long as the truck capacity is not less than three times the shovel dipper capacity (based on physical and operational experience) and truck spotting time is small.

Truck spotting time can be minimized by the use of a shovel/truck arrangement such as the double back-up system.

5.4 Hydraulic Excavators

In recent years, the use of hydraulic excavators in mining applications has been increasing. Front-end-loaders, cable shovels, and hydraulic excavators all have their strong and weak points in actual use and when choosing a piece of equipment, the mine operator must evaluate what kind of equipment best suits his needs.

The main trend in hydraulic excavators over the past few years has been an increasing size. The three largest available models, the

O & K RH300, the Demag H241, (both West Germany) and the Poclain 1000 (France) have maximum bucket capacities of 34, 21, and 16,8 cubic meters, respectively. These large machines, coupled with high selectivity, mobility, and bucket control features common to all hydraulic excavators, present a serious challenge to cable shovels. The P & H 2200 (United States), a new machine with a maximum bucket capacity of 30,6 cubic metres, will be the first non-European hydraulic excavator in this large bucket capacity range.

Another important trend in hydraulic excavators is the use of electric power as the prime mover instead of diesel engines. This is particularly significant in the larger machines. In most surface coal mine applications the hydraulic excavator's movement during the work shift is limited. Thus the laying of an electrical cable from the mine's power supply to the excavator does not reduce the machine's flexibility. As oil prices continue to rise, power from electricity rather than diesel fuel will be an important cost saving element.

In practice, hydraulic shovels are generally used when loading is done on the same level as the machine. Backhoes are commonly used for loading from a level lower than that on which the machine is resting. An important difference between hydraulic excavators and front-end loaders and cable shovels is that when loading, the direction of the penetration force can be precisely controlled. The force necessary for digging can be applied in a down-ward, horizontal, and upward direction as illustrated in Figure 5.5.

This enhances the selectivity of the loading operation. In recent years the visibility from the cab to further improve the ability of the operator to use the hydraulic shovel as a precision digging machine has been increased. The comfort of the operator has been a important design consideration.

Production Estimates.

The loading cycle of a hydraulic excavator is similar to that of an electric rope shovel and therefore the same criteria used in selecting

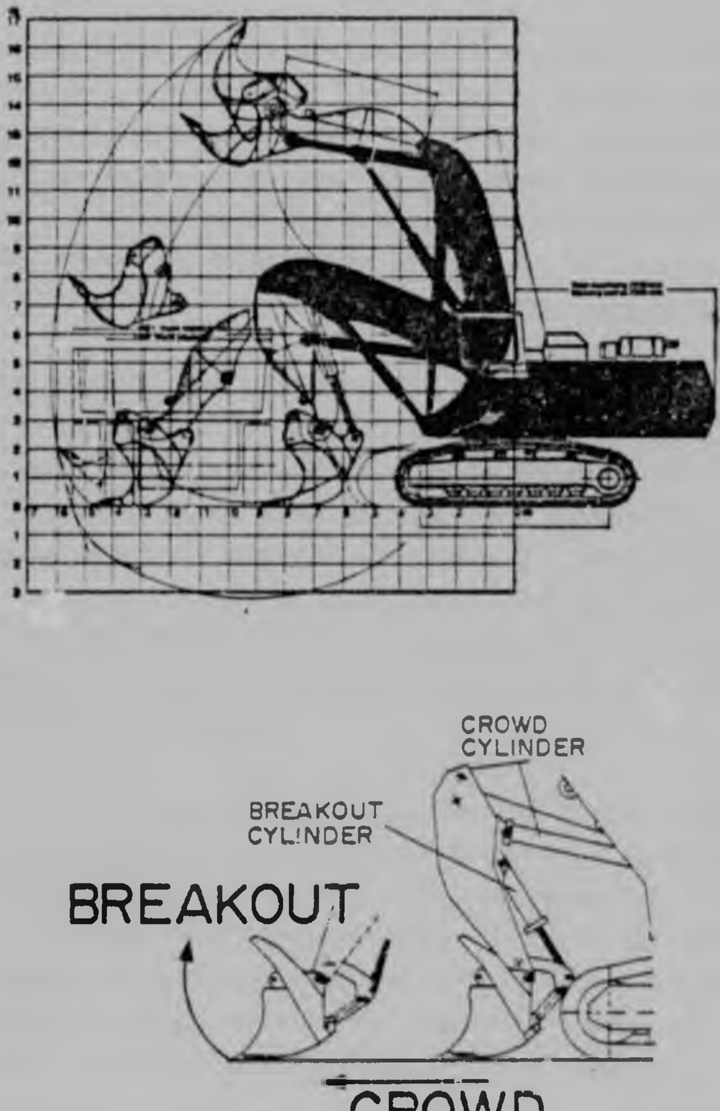

FIGURE 5.5
HYDRAULIC EXCAVATOR - DIG MOTION

a particular unit will apply.

5.5 Front-end Loaders

Although front-end loaders are primarily utilized for loading coal, gravel, rock or other materials into trucks or hoppers, the larger wheel loaders available today are being utilized to strip overburden, particularly in shallow cover, which is often found in contour mining operations. The tremendous increase in power and weight of wheeled loaders has made their substitution possible in the performance of functions previously carried out by shovels, draglines and dozers.

Due to its versatility and high speed, the larger loader can be used in virtually every phase of the stripping operation except blasting. It is common today for the front end loader to handle overburden removal, the loading of coal trucks, clean up, road work, reclamation and other functions, all during the same day and in one pit. Its high speed also enables it to roam between nearby pits. The front-end-loader has offered not only versatility to the surface mine, but also high productivity and in many cases reduced costs for the various functions it may be required to perform.

The large wheel loader has enabled improved pit layout due to its ability to move overburden over greater distances. It has without doubt improved coal recovery through less degradation in clean up work and the ability to recover coal which otherwise might have been covered up by the routine casting operations of shovels and draglines. Rubber-tired machines do not spall the top of the coal bed or contaminate it with dirt as do crawlers.

Costs vary widely from function to function, even in the same pit, as conditions change rapidly, particularly the amount and character of rock in the overburden. Where two types of machines provide similar productivity, the owning and operating expenses of each type determine the unit cost differences in material moved.

Loader tyre costs are often the largest variable while the costs of tracks and dozer components of crawler tractors vary the most. One is then often faced with the old cost problem of tyres versus tracks when assigning or choosing the most appropriate machine for a particular job. In general, soil, clay and ordinary shale are not destructive to tractor components; under these conditions life is good and costs are favorable. Massive sandstone can cause these costs to increase dramatically.

Proper operator training, operator attitude and good housekeeping with a loader working on a smooth floor, such as the top of a coal bed, can provide good tyre life and low costs. This applies regardless of the overburden content, even if massive coarsely broken sandstone is encountered. The latter condition may cause a sharp increase, however, in wear on buckets and bucket teeth.

The successful operator using wheel loaders as a primary stripping tool has learned some key operating procedures. Among these are spending a little extra time to get a full bucket load before transporting the material 200 to 300 metres. Carrying the bucket one metre off the ground provides the greatest stability and the best visibility for the driver. Efforts made to prepare and maintain a good haul road and to keep spillage cleared away from the tyres, together with some extra effort in preparing the bank with better blasting practices, are rewarded with increased production and lower costs.

(a) Front End Loader Selection

As for the rope shovel and hydraulic excavator, selection of front end loaders for a particular operation depends on:-

- required production
- loader cycle time
- payload per cycle
- required bucket size

(i) Required production

The production rate of a wheel loader should exceed by a small margin the production capability of the other critical units in the earthmoving system, since the availability of wheel loaders is normally lower than that of other loading equipment.

(ii) Loader Cycle Time

Caterpillar (October 1982) determined through their experience on various wheel loaders that basic cycle time of 0,45 - 0,55 minutes are considered reasonable when hauling loose granular material on a hard, smooth operating surface.

Material type, pile height, and other factors may tend to either increase or reduce production.

The following values, (Table 5.6) as determined by Caterpillar (1982) for many variable elements are based on normal operations. Adding or subtracting any of the variable times from the basic cycle time, will give the actual cycle time. Using the actual job conditions and the above factors, the total cycle time can be estimated.

(iii) Payload per cycle

The required payload per cycle is determined by dividing the required hourly production rate by the number of cycles per hour.

(iv) Bucket selection

After the required payload per cycle has been calculated, the payload should be divided by the loose cubic metre material weight to determine the amount of loose cubic metres required per cycle.

Therefore:

$$\text{Bucket size} = \frac{\text{Volume required/cycle}}{\text{Bucket fill factor}}$$

TABLE 5.6

FRONT END LOADER - CYCLE TIME FACTORS

		MINUTES ADDED (+) OR SUBSTRACRED (-) FROM BASIC CYCLE
MATERIALS		
Mixed	+	0,02
up to 3mm	+	0,02
3mm to 20mm	-	0,02
20mm to 150mm		0,00
150mm and over	+	0,0[illegible]d up
Bank or broken	+	0,04 and up
PILE		
Conveyor or Dozer piled 3m and up		0,00
Conveyor or Dozer piled 3m or less	+	0,01
Dumped by truck	+	0,01
MISCELLANEOUS		
Common ownership of trucks and loaders	Up to	- 0,04
independently owned trucks	Up to	+ 0,04
Constant Operation	Up to	+ 0,04
Inconsistent operation	Up to	+ 0,04
Small targer	Up to	+ 0,04
Fragile target	Up to	+ 0,05

It is important to distinguish between the various bucket ratings used for specifying front end loader capacity. It is standard to use the SAE bucket rating.

SAEJ742 (Oct 1979) specifies that the addition of any auxiliary spill guard to protect against spillage of material which might injure the operator will not be included in bucket capacity ratings. For buckets with irregular shaped cutting edges (vee edge) the strike plane should be drawn at one-third of the distance of the protruding portion of the cutting edge. "Struck capacity" is defined as that volume contained in a bucket after a load is levelled by drawing a straight edge resting on the cutting edge and the back of the bucket. "Heaped capacity" is a struck capacity plus that additional material that would heap on the struck load at a 2:1 angle of repose with the struck line parallel to the ground.

(b) <u>Alternative Selection method</u>

Another method of selecting the right front end loader and bucket to meet production requirements is to use nomographs developed by the Caterpillar Tractor Company (1982), presented in Appendix 3. This method is much quicker and easier than the normal calculation because it does not require as many calculations, yet the accuracy is about the same within the normal limits of input data.

5.6 Haul Trucks

In selecting the appropriate fleet of haultrucks the mining engineer will need to make a haulage study to determine not only the most suitable method of hauling material but also the most effective and economical type of equipment to use for the particular operation.

Apart from a wide range of truck capacities from which to select, the following different truck types must also be considered for a specific application.

(a) Rear dump-, bottom dump and side dump trucks.

(b) Mechanical drive versus electrical drive.

In recent years, the analytical methods used for the selection of the various haulage units have undergone a considerable change. Several vehicle manufacturers and a few of the mining companies employ electronic digital computer programs to carry out haulage studies to determine the optimum truck size for a specific application. The haulage computer program simulates the travel of the vehicle over the described haul roads by calculating incremental velocities, travel time, and the travel distances from the known torque vs. speed characteristics of the vehicle. This is, in a sense, a numerical integration of the vehicles' performance and, of course, varies for each and every application. These travel times are then combined with easily estimated fixed times such as loading, turning, dumping spotting, and any indicated delay times, resulting in a total cycle time for the vehicle. A typical example is included in Appendix 4.

Truck productivity decreases as haul distance or travel time increases. Figure 5.6 illustrates this trend for truck sizes of 85, 120, 180 and 210 tons.

A level haul profile has been used to construct this figure, with truck speeds of 32 km per hour. If the haul had been on adverse grades, the relationship would remain the same but truck productivity would decrease more rapidly with haul distance.

Studies by Bucyrus Erie (1979) indicate that the average hourly operating cost per ton decreases with an increase in truck size as illustrated in Figure 5.7.

By incorporating the cycle time, which is a function of the travel distance, Bucyrus Erie (1979) derived at a relationship between unit haulage cost and haul distance as shown in Figure 5.8.

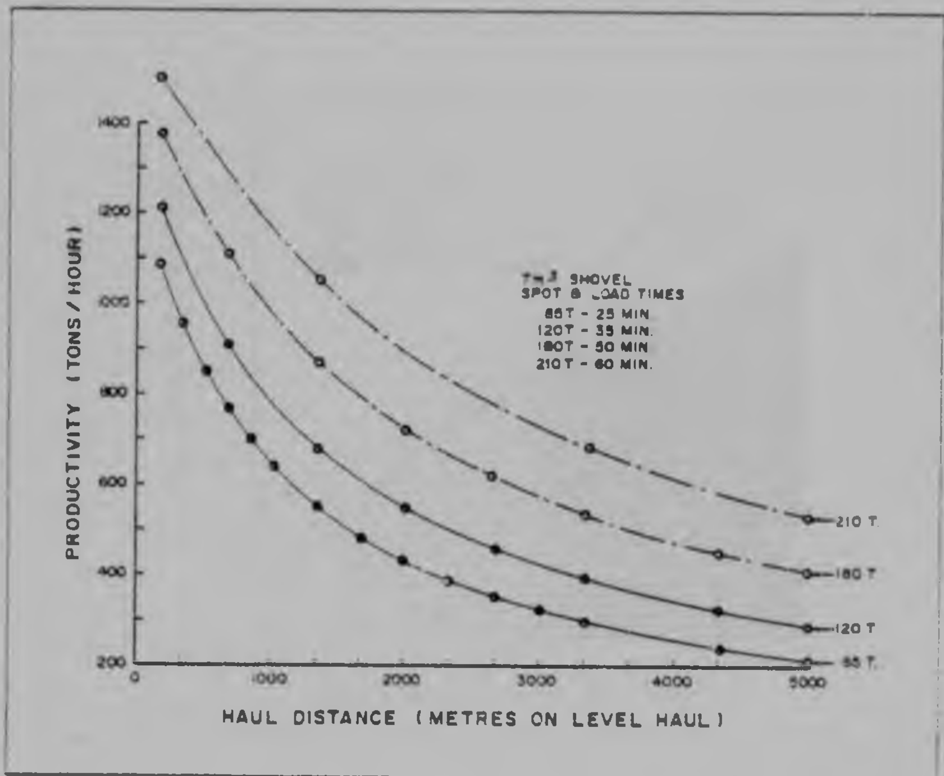

FIGURE 5.6
HAULAGE TRUCK PRODUCTIVITY VERSUS
HAUL DISTANCE

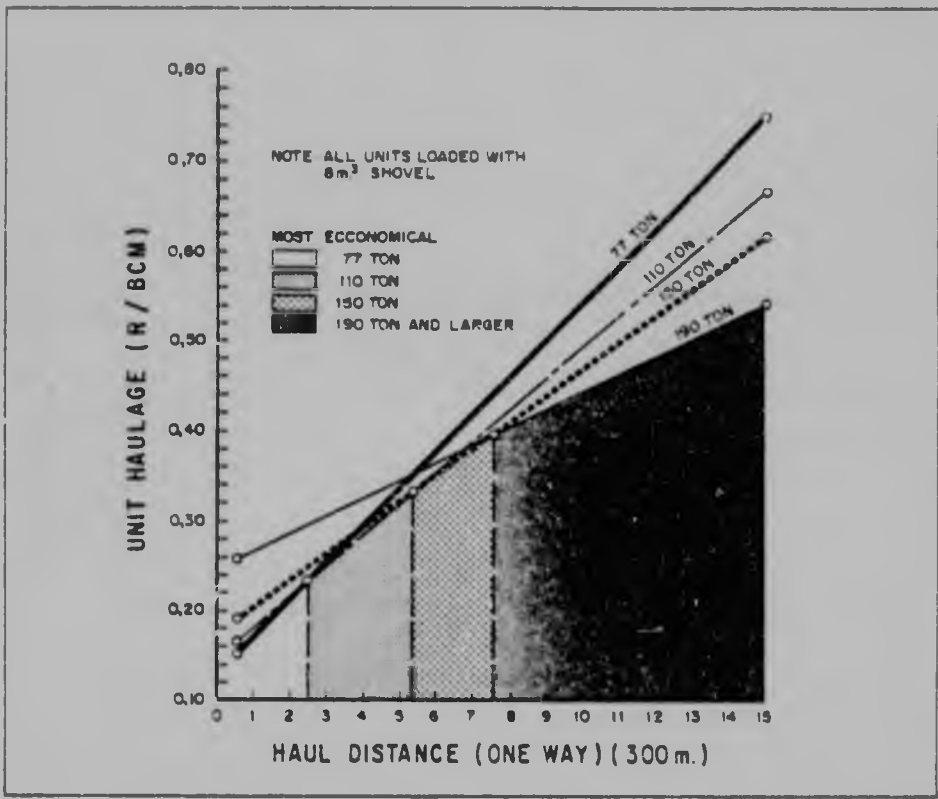

FIGURE 5.7
THE RELATIONSHIP BETWEEN HAULAGE
COST AND HAUL DISTANCE FOR
DIFFERENT TRUCK SIZES ASSUMING A
TRAVEL SPEED OF 32 kph AND LEVEL HAUL

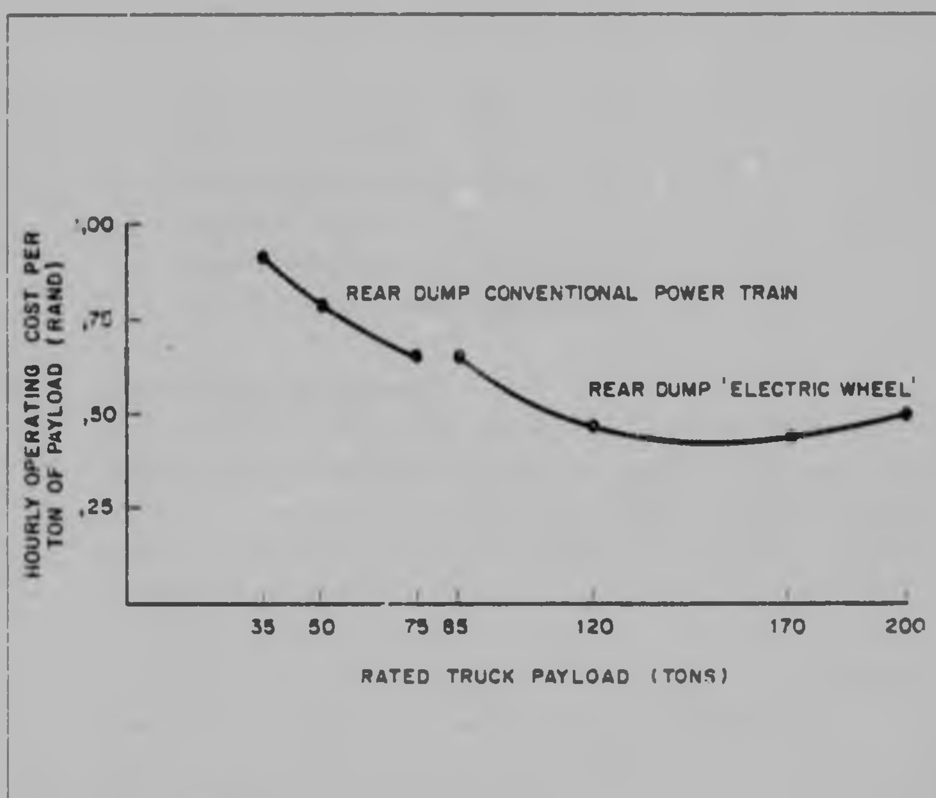

FIGURE 5.8
TOTAL TRUCK OPERATING AND OWNERSHIP COST VERSUS TRUCK PAYLOAD

A most interesting aspect of this figure is that the larger trucks only become economic on long hauls or travel times and in fact the smaller unit (85 ton) is the most economic on flat hauls up to one kilometre in one direction.

(a) <u>Truck fleet size</u>

The number of haulage units required for a system is perhaps the most difficult part of the shovel/truck selection process. The factors affecting this decision are:

- Total required production.
- Production capacity per truck.
- Mechanical availability of units.
- Operating method.
- Synchronization of the entire load, haul and dump system (simulation).

(i) <u>Total required production</u>

As outlined in the shovel selection, the size of shovel, the number of operating shovels and the shovel fleet size must first be calculated. Each operating shovel requires a certain number of trucks to keep it occupied such that its productive capacity or requirement can be achieved. For a given size of truck, the number required will depend on the speed in which a complete cycle can be completed. The number of trucks to be operated per shovel is commonly found by dividing the total cycle time by the spot and load time. This is referred to as the shovel/truck match number.

$$\text{Match number} = \frac{\text{Total Cycle time}}{\text{Spot and load time}}$$

The less variation in the mean time for each entity of the cycle, the more valid or applicable is the match number. If load time, haul time or dump time are very erratic or variable, then queueing of the trucks will occur somewhere in the cycle. As a result, the actual shovel production will be less than estimated. To overcome this situation, more trucks can be used than is indicated by the match number. This will maximize the shovel production but will decrease the productive capacity of the haulage fleet. Alternatively, more shovels can be operated with less trucks which decreases the productive capacity of the shovels but maximizes the haul fleet productivity. Figure 5.9 is a typical illustration of the relationship between production rate and the number of trucks assigned to the shovel. To determine the most economic operating point, the cost of operating the shovel and trucks must be incorporated into the analysis.

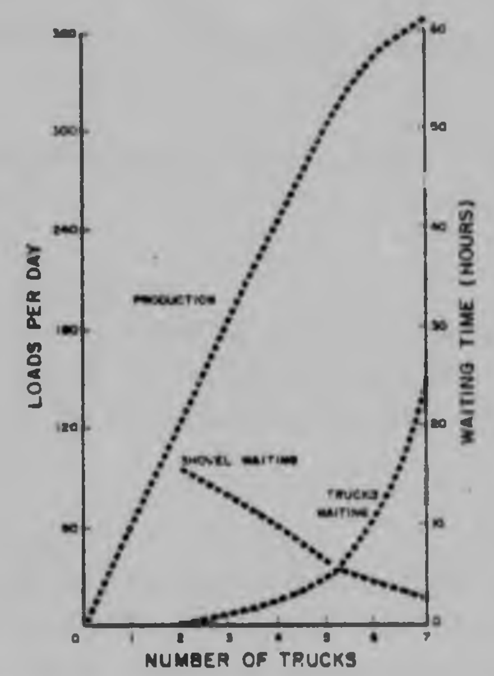

FIGURE 5.9
PRODUCTION VERSUS NUMBER OF TRUCKS FOR
ONE SHOVEL LOADING IN GOOD DIGGING
(BUCYRUS ERIE, 1979)

(ii) Truck Productivity

The productive capacity of a truck is controlled by the size of the unit, load time, travel time (haul and return), dumping time and the inherent fixed and variable delays in the operation.

Load Time

For a given size of truck matched to a shovel of a given dipper capacity, the load time can be calculated by:

$$\text{Number of Passes} = \frac{\text{Truck Capacity (CY heaped as 2:1)}}{\text{Dipper Capacity} \times \text{Dipper Fill Factor}}$$

$$\text{Load Time(min)} = \frac{\text{Passes} \times \text{Work Cycle}}{60}$$

The number of passes must be rounded to the nearest whole number.

Spot at Loading Unit

Associated with loading is the time required for the haulage unit to spot in the loading position. Table 5.7 gives average values for the different types of haulage unit and the operating conditions.

Travel Time

The travel time includes the time required for the loaded truck to haul from the shovel to the dump point and the time to return empty from the dump point to the shovel. The manufacturer provides a performance or gradeability chart for each of its models based on weights, gear ratios and tyre data. From this chart, vehicle speed and braking range can be determined.

The effective grade or gradeability is defined as the ability of a vehicle to negotiate a given grade taking into account both the physical grade of the haul road and rolling resistance.

TABLE 5.7
AVERAGE HAULAGE UNIT SPOTTING TIMES
AT THE SHOVEL

	SPOTTING TIME AT THE LOADING UNIT (MINUTES)		
OPERATING CONDITIONS	BOTTOM DUMP TRACTOR-TRAILER	REAR DUMP	SIDE DUMP TRACTOR-TRAILER
Favourable	0,15	0,15	0,15
Average	0,50	0,30	0,50
Unfavourable	1,00	0,50	1,00

Turn, Spot and Dump Time

Turning, spotting and dumping time depends upon the type of haulage unit and the specific operating conditions. As a guide, average values for the different types of trucks under the indicated conditions are shown in Table 5.8.

TABLE 5.8
AVERAGE HAULAGE UNIT TURN, SPOT AND DUMP TIMES

OPERATING CONDITIONS	TURN, SPOT AND DUMP TIME (MINUTES) BOTTOM DUMP TRACTOR-TRAILER	REAR DUMP	SIDE DUMP TRACTOR-TRAILER
Favourable	0,3	1,0	0,7
Average	0,6	1,3	1,0
Unfavourable	1,5	1,5 - 2,0	1,5

Operational Delay Factors

To establish the productive capacity of a haulage unit, an estimate of the time spent actually hauling must be made. For a given period of time (shift, day, etc.), there will be operational delays that reduce the productive capacity of a haulage unit. These delays, as for the shovel, can be classified into two categories:

- Fixed delays.
- Variable delays.

Operating Conditions	Percentage Working Time	Working Minutes/Hour
Excellent	92%	55
Average	83%	50
Unfavourable	67%	40

The percentage working time is commonly known as the job efficiency.

(b) Production Capacity Per Truck

Once truck size (rating and capacity), cycle time, and actual operating times are known the haulage unit productivity can be calculated. This productivity can be calculated using one of three methods, with each method having a distinct meaning or use:

(i) Theoretical Productivity

The theoretical productivity is the tons or cubic metres per hour produced by an operating unit if no delays were encountered. This implies a 100% potential which is rarely if ever, achieved.

Therefore:

$$\text{Tons per Hour (TPH)} = \frac{60 \times (\text{Truck Rating})\ \text{tons}}{(\text{Cycle Time})}$$

or Bank Cubic metres per Hour (BCM/hr)

$$= \frac{60 \times (\text{Truck Rating})}{(\text{Cycle Time}) \times (\text{Swell Factor}) \times (\text{Density})}$$

(ii) Average Productivity

The average productivity is the tons or cubic metres per hour produced by an operating unit taking into account fixed and variable delays. This rate should be applied to the desired period of time (shift, day, etc.) to estimate total production.

$$\text{TPH} = \frac{(U - D) \times 60 \times E \times TR}{U \times C \times SF \times M}$$

Where

U	=	Unit of time (Say 8 hours)
D	=	Fixed delays (hours)
E	=	Job efficiency
TR	=	Truck rating (tons)
C	=	Cycle time (minutes)
M	=	Material density (tons/bank cubic metre)
SF	=	Swell factor

(iii) Peak Productivity per Hour

The peak productivity per hour is the tons or cubic metres per hour produced by an operating unit taking into account variable delays only. This rate would be used to determine the number of haulage units to be assigned to a shovel to achieve a given required production.

$$TPH = \frac{60 \times (\text{Job Efficiency}) \times (\text{Truck Rating})}{(\text{Cycle Time}) \times (\text{Swell Factor}) \times (\text{Density})}$$

(c) Truck Mechanical Availability

In general, the status of a haulage unit at any given time would be:

- Operating
- Mechanical/Electrically unavailable
- Spare

The term with the most definitions in the mining industry is "mechanical availability". For purposes of clarification, one of the most common definitions will be used for this presentation. Mechanical availability will thus be defined as the time that a unit of equipment is operated divided by the total possible time less spare time. Spare time is removed from the calculation on the basis that had the machine been operating instead of spare, then the same ratio of operating and maintenance downtime would have been obtained.

% Mechanical Availability =

$$\frac{(\text{Operating time} + \text{Operating delays}) \times 100}{\text{Total time} - \text{Spare time}}$$

Spare time on haulage units or any equipment may be the result of absenteeism in the scheduled work force, fluctuating mechanical availability or planned.

As a general rule, the number of spare mechanically available units necessary to maintain a required fleet is 10% of the scheduled units to be operated. The size of the fleet, therefore, may be calculated by:

Total Number of Units (Fleet) =

$$\frac{\text{Required Operating Units} \times 1.1}{\text{Mechanical Availability}}$$

To conclude, it should be remembered that if the average mechanical availability of a truck fleet is say 80%, this does not mean that there will always be 8 out of 10 trucks mechanically available.

Consider the case of a truck fleet of 10 trucks with a mechanical and electrical availability of 70%. The number of trucks available at any one time can be calculated by assuming that the probability that the truck is operational is equal to the overall availability and using the binomial distribution. Table 5.9 lists the individual probabilities for each number of trucks and Figure 5.10 illustrates the resulting histogram.

TABLE 5.9
PROBABILITY OF AVAILABLE TRUCKS FOR A FLEET SIZE OF 10 AND A MECHANICAL AVAILABILITY OF 70%

NUMBER OF TRUCKS	PROBABILITY THAT AT LEAST THE STATED NUMBER OF TRUCKS ARE AVAILABLE (%)
10	2,82
9	14,93
8	37,28
7	63,96
6	83,97
5	95,27
4	98,94
3	99,83
2	99,99
1	100,00
0	100,00

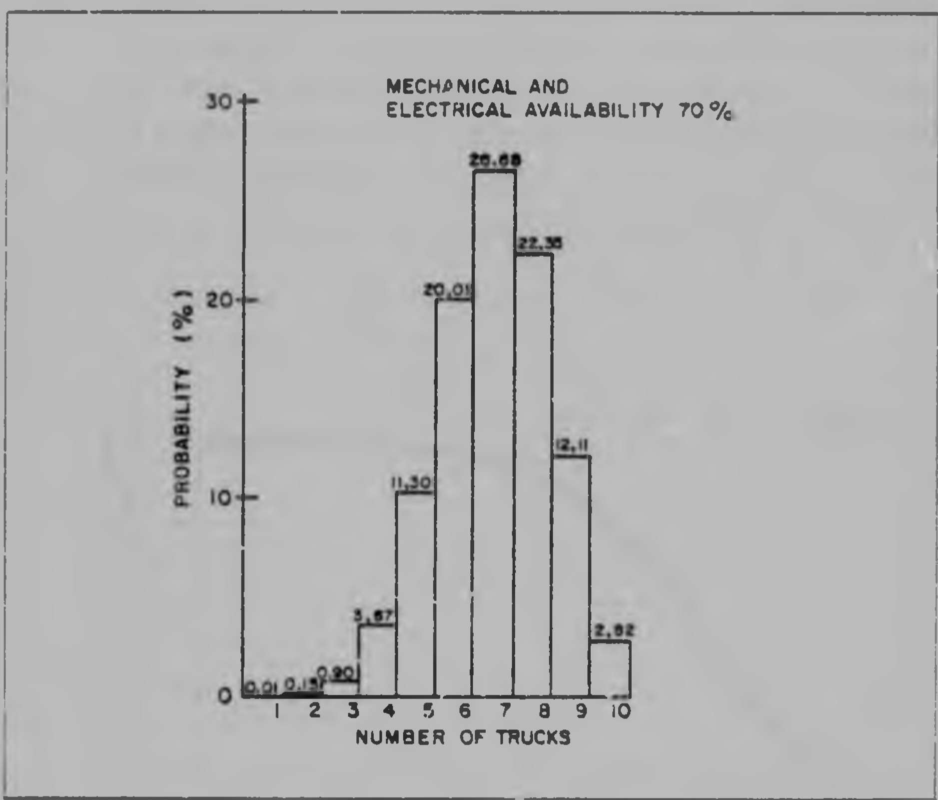

TABLE 5.10

TRUCK AVAILABILITY HISTOGRAM

Table 5.9 presents the cumulative probabilities for the same example, i.e. 10 trucks with a fleet mechanical and electrical availablity of 70%. There is only a probability of approximately 64% that 7 or more trucks will be operational. These figures bear out the previous statement that approximately 10% extra capacity is required to ensure the calculated production.

Figure 5.11 presents a visual representation of the cumulative probabilities. It is interesting to note that 99% of the time 3 or more trucks will be available. On the other hand, 1% of the time, maybe 3 or 4 days per year, only one or two trucks will be operational.

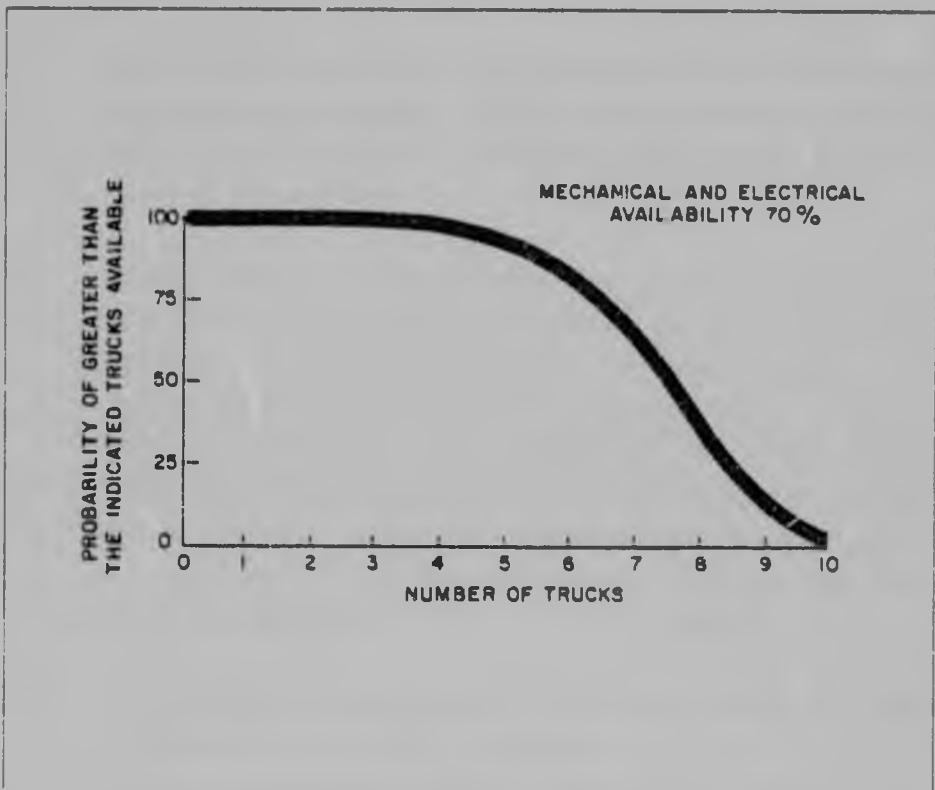

FIGURE 5.11
CUMMULATIVE TRUCK OPERATIONAL
PROBABILITY CURVE

5.7 Financial Evaluation

To illustrate the importance of evaluating alternative mining methods and equipment, a typical opencast operation will be analysed, consisting of a single coal seam of 4 metres, overlaid by 30 metres of sandstone overburden. The basic assumptions and parameters used in the evaluation are as follows:

(a) All overburden material is assumed to have a swell of 130% for construction of dragline range diagrams and to determine bucket loads Where reference is made to overburden depths, it is assumed that topsoil has been removed by other methods. No allowance is made for dragline spoil stability as it relates to spoil height.

(b) This calculation assumes a pit of sufficient length to eliminate any operating constraints. The pit width is assumed to be 40 metres in all examples. Highwall slopes of 70^{o} and spoil slopes of 38^{o} are used.

(c) Working Schedule: The equipment used in these comparisons is scheduled on the basis of 7416 hours per year, which is equivalent to 309 working days at 24 hours per day. For dragline operations a total of 8400 scheduled hours per year is used, as exemptions were granted by the office of the Government Mining Engineer to operate these units on a 7 day per week basis. Operating efficiencies of 85% for dragline excavators and 70% for shovels, hydraulic excavators and front end loaders are used.

(d) For draglines an availability of 85% is used and for all other equipment a factor of 80% is assumed.

(e) The economic comparison is based on the present value of operating costs and original investment, inflation at 10% and escalation at 10% per annum, and a 20 year investment period.

In all instances, it is assumed that the equipment can be erected in two years. Forty percent of the investment is arbitrarily assigned to the first year and 60 percent to the second. The economic rating illustration compares the most desirable alternative, expressed as 100, with the other alternatives.

Operating costs include labour and electric power. No differentiation is made for the various classes of labour because it is felt the accuracy will not materially affect the results.

An allowance of R5 000 per hectare covers those cases where dragline spoil piles are created and must be levelled. This is not intended to be the total reclamation cost, but only the cost of levelling the peaks to a gentle roll. Where spoil dumping is carried out by trucks, a cost of R2 500 per hectare is used.

This analysis covers the following mining methods:

- Draglines
- Shovels and Trucks
- Hydraulic Excavators and Trucks.

for the following annual production rates: 5; 10 and 15 million cubic metres of overburden per year.

Detailed calculations are presented in Appendix 5 and summarized in Table 5.10 below:

TABLE 5.10

ECONOMIC RATINGS OF ALTERNATIVE OVERBURDEN STRIPPING METHODS FOR VARIOUS ANNUAL PRODUCTION RATES

	OVERBURDEN STRIPPING METHOD		
	DRAGLINES	SHOVEL/ TRUCK	HYDRAULIC EXCAVATOR/ TRUCK
Annual production rate (BCM)	5×10^6	5×10^6	5×10^6
Economic rating	100	223	225
Annual production (BCM)	10×10^6	10×10^6	10×10^6
Economic Rating	100	258	261
Annual Production rate (BCM)	15×10^6	15×10^6	15×10^6
Economic Rating	100	293	296

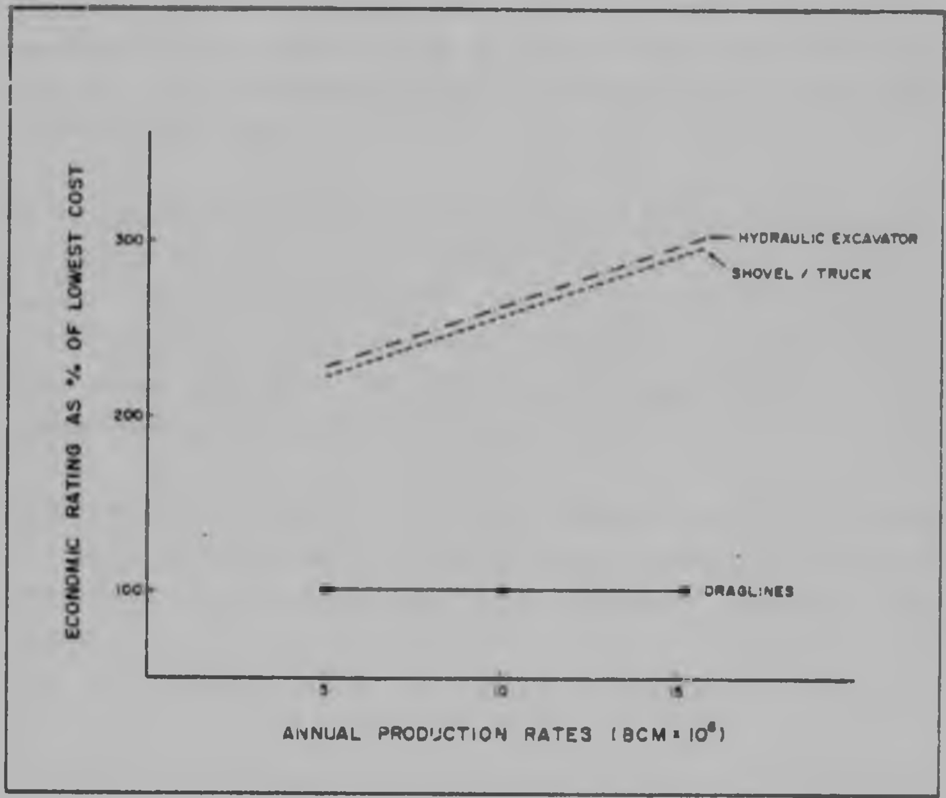

FIGURE 5.12
COST COMPARISON OF ALTERNATIVE MINING METHODS
AT DIFFERENT PRODUCTION RATES

5.8 CONCLUSIONS AND RECOMMENDATION

Figure 5.12 illustrates the results of the evaluation study as summarized in Table 5.10

From this graph it is evident that even if the initial capital costs to purchase draglines are higher than that for the other two mining methods, this alternative presents a lower unit cost per cubic metre over the project life.

It is also significant from the graph that draglines become more favourable than other mining methods for larger annual production rates. This is mainly because both the rope shovel and hydraulic shovel methods depend on haultrucks to transport overburden from the face to the spoil pile. These trucks consume large quantities of fuel which cause operating costs to increase.

Draglines are commonly used in the Witbank Area for overburden stripping as illustrated in Figure 5.13 and should therefore be considered as the basis of design for all opencast operations in this region.

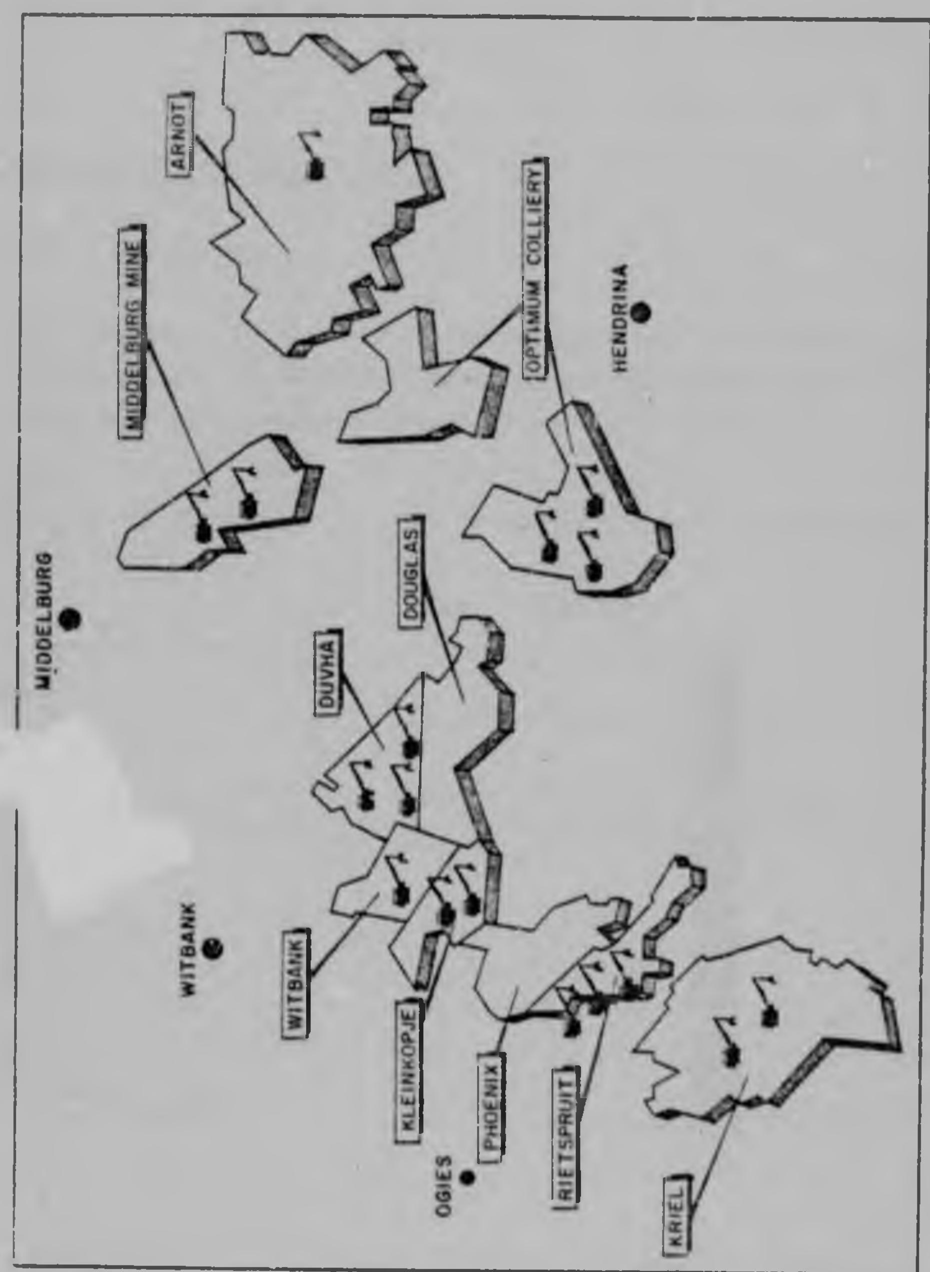

FIGURE 5.13

POPULATION OF LARGE DRAGLINES IN THE WITBANK AREA

CHAPTER 6

COAL AND PARTING REMOVAL

6.1 Introduction

The five main coal seams of the Witbank Area are inconsistent in thickness and the individual seams are often split by sandstone, siltstone or shale parting as illustrated in Figures 6.1 and 6.2

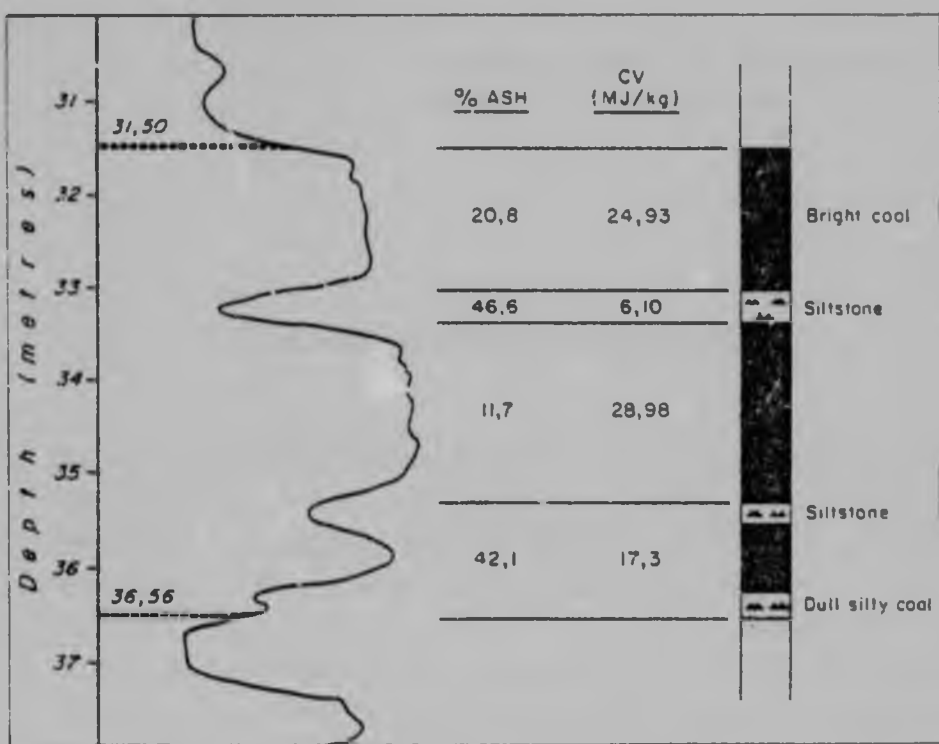

FIGURE 6.1

GEOLOGICAL VARIATIONS WITHIN SEAM 2 FROM WIRE-LINE δ -δ LOG, CORE TEST RESULTS AND BOREHOLE LOGS

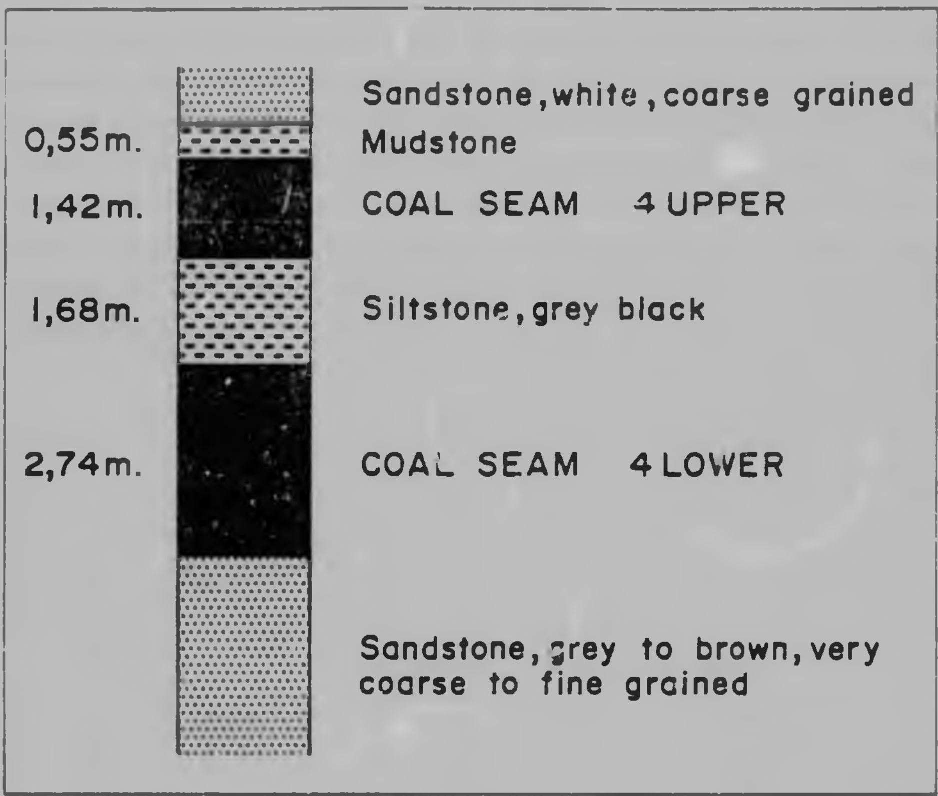

FIGURE 6.2
TYPICAL GEOLOGICAL SECTION OF SEAM 4

As result of the inter relationship between the various coal seams and the interbedded parting it is important to select a mining method and the associated equipment that will excavate both the coal seams and the parting horizons successfully at an economical cost.

The largest percentage of coal produced in the Witbank Area is sold to customers who require strict quality control measures, and specifications and size grading of the final product are therefore of utmost importance.

In selecting a particular mining method for the various coal seams (and parting horizons) the ultimate aim is not only the removal of the coal at the lowest possible cost but also the mining method that will generate the least amount of fines. The difference in the listed price for 'B' grade coal on the inland market for various product sizes are presented in Figure 6.3 (Government Gazette, December 1983). From this graph it can be seen that the price of the larger product sizes is some 17,2% higher than that of the fine product. This price differential can have a marked effect on the overall profitability of any mining project.

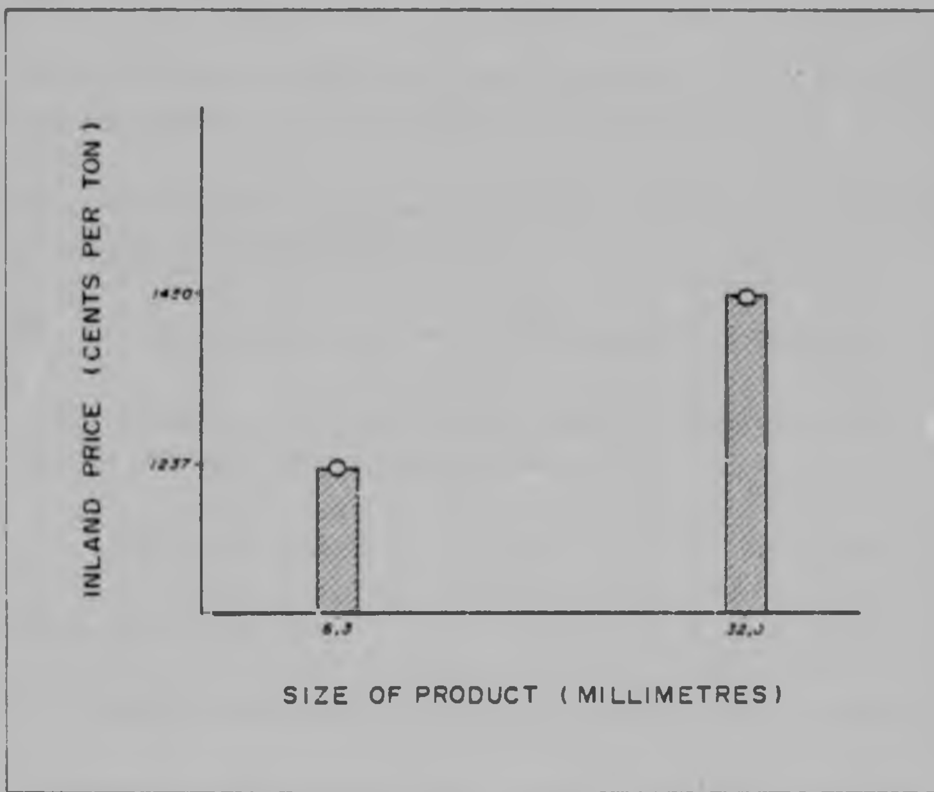

FIGURE 6.3
PRICE DIFFERENCE FOR DIFFERENT PRODUCT SIZES OF 'B' - GRADE COAL

The first section of this chapter illustrates the importance of in-pit parting separation, followed by the various aspects of coal breaking to be considered before selecting a particular mining method. Finally the loader and coal haulage system selection are detailed.

6.2 Economics of in-pit parting separation

The economics of in-pit parting separation have been assessed using the in-seam parting for Seam 4 as an example, as illustrated in Figure 6.2.

The total costs including mining, hauling and plant discard rehandling have been estimated for the following two situations:-

(a) mining the 4 upper seam, parting and 4 lower seam as three separate mining horizons and

(b) mining both seams and the included parting as a single unit.

If it is assumed that the parting has a variable thickness of 't' metres then the cost per ROM ton for alternative (a) above is

$$(0{,}77 + 0{,}19t) \text{ Rands} \qquad \text{——————} \qquad (6.1)$$

and for alternative (b) is

$$(0{,}67 + 0{,}29t) \text{ Rands} \qquad \text{——————} \qquad (6.2)$$

as detailed in Appendix 6.

The break-even point in the two mining methods is therefore at that point where equation 6.1 is equal to equation 6.2

$$\therefore\ 0{,}77 + 0{,}19t = 0{,}67 + 0{,}29t$$

and $t = 1{,}0$ metres, as graphically illustrated in Figure 6.4

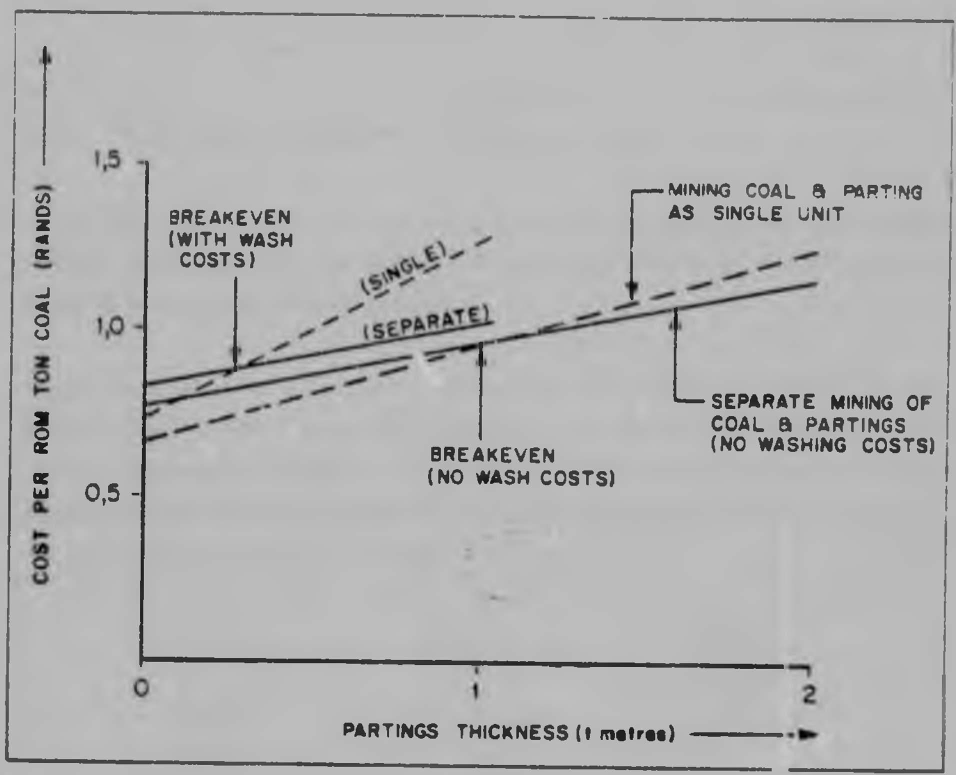

FIGURE 6.4

ESTIMATION OF BREAK-EVEN PARTING THICKNESS FOR SELECTIVE MINING

Similarly it can be proved (as indicated in Appendix 6) that if coal washing costs are included, the maximum thickness of parting that can be economically tolerated in the ROM feed to the washing plant is 0,29 metres.

The selected coal and parting mining methods must therefore also be capable of mining thin layers successfully.

6.3 Coal Breaking

King et al (1979) states that "in this country (South Africa), one of the factors inhibiting the use of machines which simultaneously cut and load is the high strength and abrasiveness of the local coals.

Equipment designed overseas principally for use in the United States of America, the United Kingdom and Europe where coal seams are generally much weaker than here, is in most cases technically and economically inferior to conventional South African working methods which make use of explosives to break the coal."

Although this statement by King et al (1979) applies to underground mining methods and the use of continuous miners, it can also be applied to surface mining methods.

Investigations by Steffen, Robertson and Kirsten (1982) on the geotechnical properties of the Witbank coal seams indicate that the mean uniaxial strength is 30 MPa. Schmidt hammer tests and penetrometer results by King et al (1979) indicate values up to 68 and 40 kN respectively as illustrated in Figure 6.5

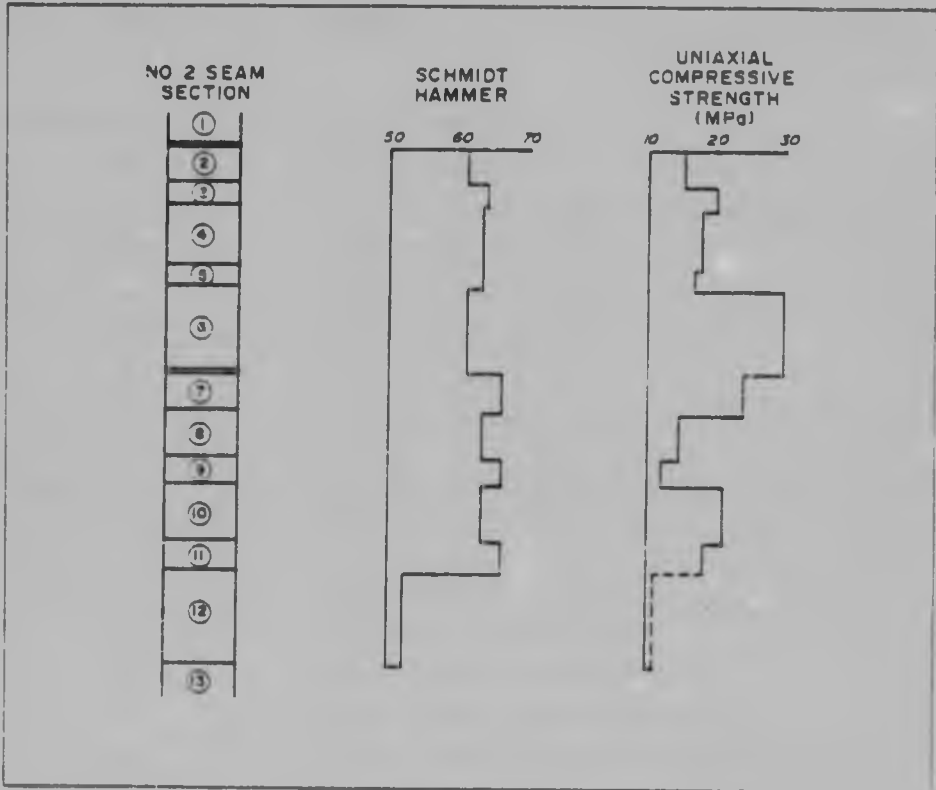

FIGURE 6.5

SCHMIDT HAMMER TESTS AND PENETROMETER RESULTS FOR NO 2 COAL SEAM (KING ET AL, 1979)

All these results and information indicate that breaking of the coal seam by explosives is required.

Similarly, the excavatability of the siltstone and sandstone parting can be determined by using the classification system proposed by Kirsten (1982). Results from a geotechnical investigation by Steffen, Robertson and Kirsten (1982) indicate the excavatability index (N) to be 1300 for siltstone and 5160 for sandstone partings, corresponding to extremely hard digging conditions which will require blasting prior to excavation.

To determine the optimum drill size for satisfactory fragmentation, the following formulae apply (Cunningham, 1982).

$$B \times S = \frac{r\ X\ D^2}{1{,}27\ k} \qquad \text{———————— (6.3)}$$

Where	B	=	Burden (metres)
	S	=	Spacing (metres)
	r	=	Ratio of charge length to bench height
	H	=	Bench height (metres)
	L	=	Charge length (metres)
	K	=	Powder factor (g/m^3)
	X	=	Explosive density (g/cc)
	D	=	Blasthole diameter (m)

and X =

$$X = A\left(\frac{Q}{Vo}\right)^{-0{,}8} Q^{1/6} \frac{(115)^{2/3}}{E} \qquad \text{———————— (6.4)}$$

where	X	=	Mean fragment size (cm)
	A	=	Rock factor (=6 for coal)
	Q	=	Explosive mass per blasthole (kg)
	Vo	=	Volume broken per blasthole (m^3)
	E	=	Relative weight strength of explosives, taking E = 100 for ANFO

Table 6.1 summarizes the results for a 6 metre coal seam using formulae 6.3 and 6.4, where ANFO explosive is used to obtain a powder factor of 0,2 kg per cubic metre.

TABLE 6.1
RELATIONSHIP BETWEEN BLASTHOLE DIAMETER AND MEAN FRAGMENTATION

HOLE SIZE (mm DIA)	DRILL PATTERN (BURDEN x SPACING)	MEAN FRAGMENTATION (cm)	NUMBER OF HOLES PER 100 000 TONS
20	0,8 x 0,9m	27,8	15 432
40	1,7 x 1,8m	34,5	3 631
60	2,6 x 2,6m	39,5	1 644
75	3,2 x 3,2m	42,6	1 025
100	4,3 x 4,3m	46,9	600
125	5,4 x 5,4m	50,5	381
175	7,6 x 7,6m	56,5	192
200	8,7 x 8,7m	59,1	146

Figure 6.6 illustrates the results obtained in Table 6.1 above.

From this graph it can be seen that holes smaller than 60mm diameter will not be considered feasible alternatives for the following reasons:

- The smaller the blasthole, the finer the product and this can result in excessive fines which might not be acceptable to the consumer.

- The number of holes to be drilled for a given tonnage

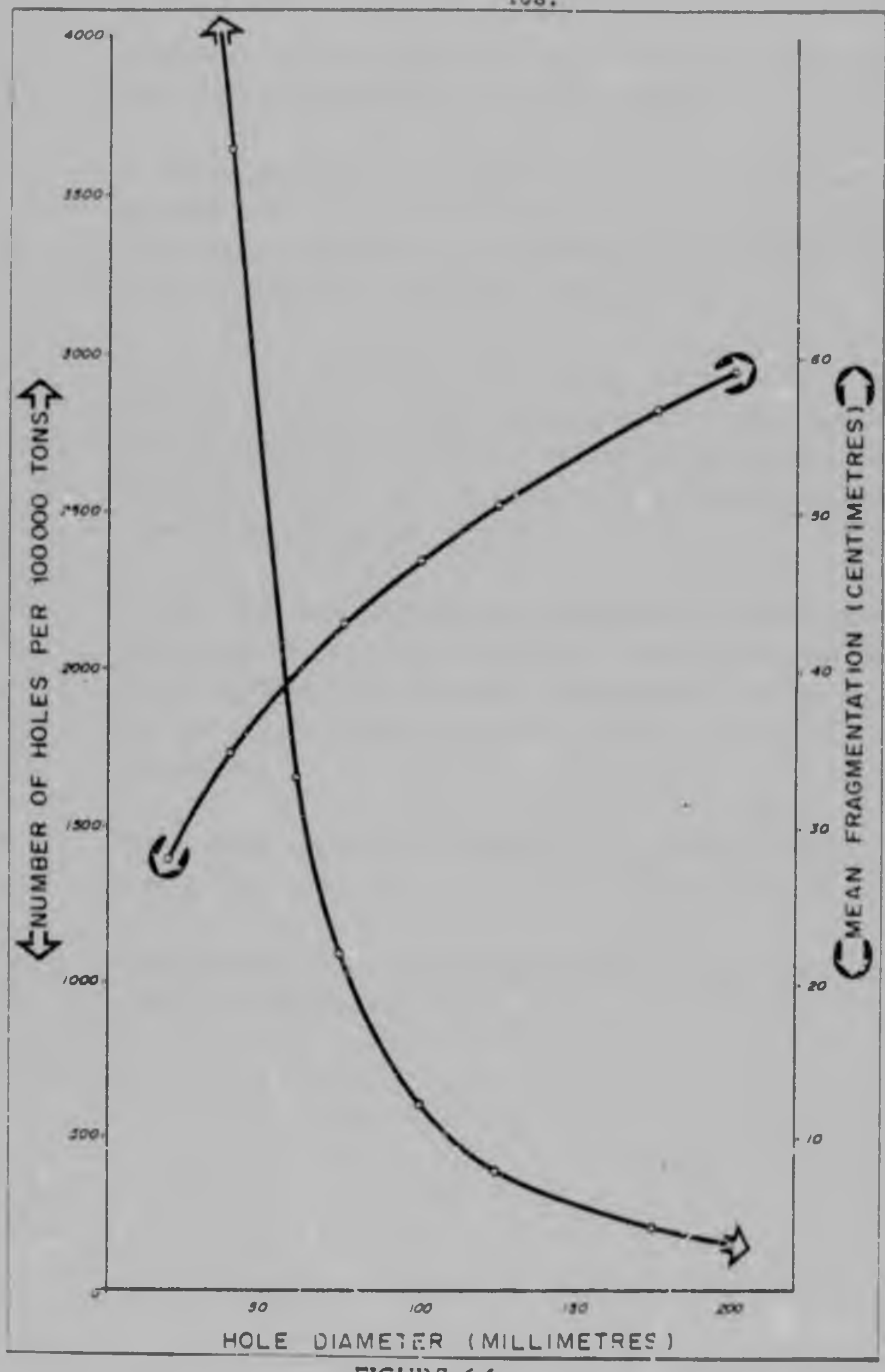

FIGURE 6.6

RELATIONSHIP BETWEEN DRILL HOLE SIZE, NUMBER OF HOLES AND MEAN FRAGMENTATION FOR COAL SEAMS requirement is very large; thus a large number of drilling units, each with its own drilling crew will be required.

If the average moving and set-up time between blastholes is assumed to be 1 minute, the total unproductive time generated by a 20mm diameter drill is 257 hours per 100 000 tons drilled, compare to 6 hours for a 125 mm drill.

- If, due to wet conditions, water resistant slurry explosives are used at any stage during the mining operation the explosive will fail to detonate in drillholes smaller than 75mm diameter, since the critical diameter required for propagation is about 75mm. (Clay et al, 1960)

- On the other hand, large diameter blastholes (> 175mm) require drilling patterns where the burden (or spacing) exceeds the bench height or coal thickness. Fragmentation will be along the line of least resistance and poor blasting results will be experienced.

The optimum drill hole size range is thus between 75 and 125 mm.

The following table summarizes a few machines that will satisfy this requirement.

TABLE 6.2
BASIC SPECIFICATION OF COAL DRILLS

MANUFACTURER	MODEL NO.	HOLE SIZE (mm DIA.)	PULLDOWN (Kg)	ROTATION SPEED
Gardner - Denver	RDC - 16B	125	7 200	0 - 250
Gardner - Denver	GD - 25/35C	127 - 152	15 876	0 - 162
Ingersoll-Rand	Damco 3000	140	8 636	70 - 375
Schramm	C42 - B	140	6 681	0 - 113
Schramm	C42H - B	140	6 681	0 - 113

6.4 Coal and Parting Loading

A large selection of mining methods and associated equipment are available for coal loading and transport. Singhal (1983) compiled the following equipment rating chart (Table 6.3) which can be used as a guide in selecting alternatives for any specific project.

From this table it can be seen that shovels and hydraulic excavators provide the best alternatives in the relatively hard seams encountered in the Witbank Coalfield where fragmentation is normally poor.

To evaluate the two loading systems a typical double seam mining operation of a 42m wide cut will be analysed consisting of a lower seam 1,2 metres thick and an upper seam of 4 metres thick. The seams are separated by a hard and abrasive shale layer, varying in thickness from 0,6 to 1,2 metres. This is typical of the No. 4 seam or 1 and 2 seam mining horizons.

TABLE 6.3
EQUIPMENT RATING : COAL LOADING
(SINGHAL, 1983)

1. Should be considered
2. May be considered
3. May be considered under certain conditions
4. May be considered special situation

A. High
B. Moderate
C. Low

						SCRAPERS			
		SHOVELS	(RUBBER TYRED) FRONT-END LOADERS	HIGH LIFTS (TRACKS)	HYDRAULIC SHOVEL/ BACKHOE	SELF-LOADING FULL-POWERED	ELEVATING	UNDER-POWERED WITH TRACTOR	BUCKET WHEEL EXCAVATOR
Coal seam thickness	0,3 - 1,0m	2	1	1	2	1	1	1	4
	1,0 - 1,5m	1	1	1	1	1	1	1	3
	1,5 - 3m	1	1	1	1	2	2	2	2
	3 - 7,5m	1	2	2	1	3	3	3	1
	> 7,5m	1	3	3	1	4	4	4	1
Fragmentation or ease of digging	Soft or highly fragmented	1	1	1	1	1	1	1	1
	Moderately fragmented	1	1	1	1	2	2	2	3
	Hard or Low fragmentation	1	3	2	1	4	3	4	
Floor conditions	Very soft	4	1	1	1	1	1	1	3
	Moderate	1	1	1	1	1	1	1	2
	Hard	1	1	1	1	1	1	1	1
Mobility		B	A	A	A	A	A	A	B
Flexibility under varied field conditions	Good	A	A	A	A	A	A	A	B
	Fair	B	B	B	A	A	A	A	B
	Poor	C	B	B	B	B	B	B	C
Production requirements	High	1	1	3	1	1	1	1	2
	Medium	1	1	1	2	1	1	1	1
	Low	1	1	1	3	1	1	1	1

6.4.1 Shovel Loading Method

The shovel loading method consists of the following mining sequences which is illustrated in Figure 6.7

(a) The entire width of the top coal seam is drilled and blasted and loaded with the shovel into coal haulers.

(b) The exposed parting horizon is then drilled and blasted. The first 14m strip of the parting is side casted into the spoil pile void. Based on the boom cinematic of the standard size rope shovel, it is only possible to remove a 14m wide parting strip without repositioning the shovel.

(c) After completion of this 14m wide cut, the bottom coal seam is cleaned by means of dozers, drilled, blasted and loaded with the rope shovel. To accommodate the haul trucks, levelling of the broken parting in the adjacent strip is required. As result of the very thin bottom seam (1,2 metres) stockpiling with dozers will be required to give better bucket fill factors.

(d) This sequence is repeated for the remaining two 14m wide strips.

The disadvantages of this mining method are summarized as follows:

x The mining width of 42 metres is determined from the geometry of the dragline. Reducing the stripping width is not possible due to the space required by the rope shovel and dumpers for efficient operation.

x Loading of the coal seam next to the broken parting results in excessive contamination due to the loading action of a rope shovel.

x Due to the high ground bearing pressure of the shovel, its repositioning during the mining of the parting would

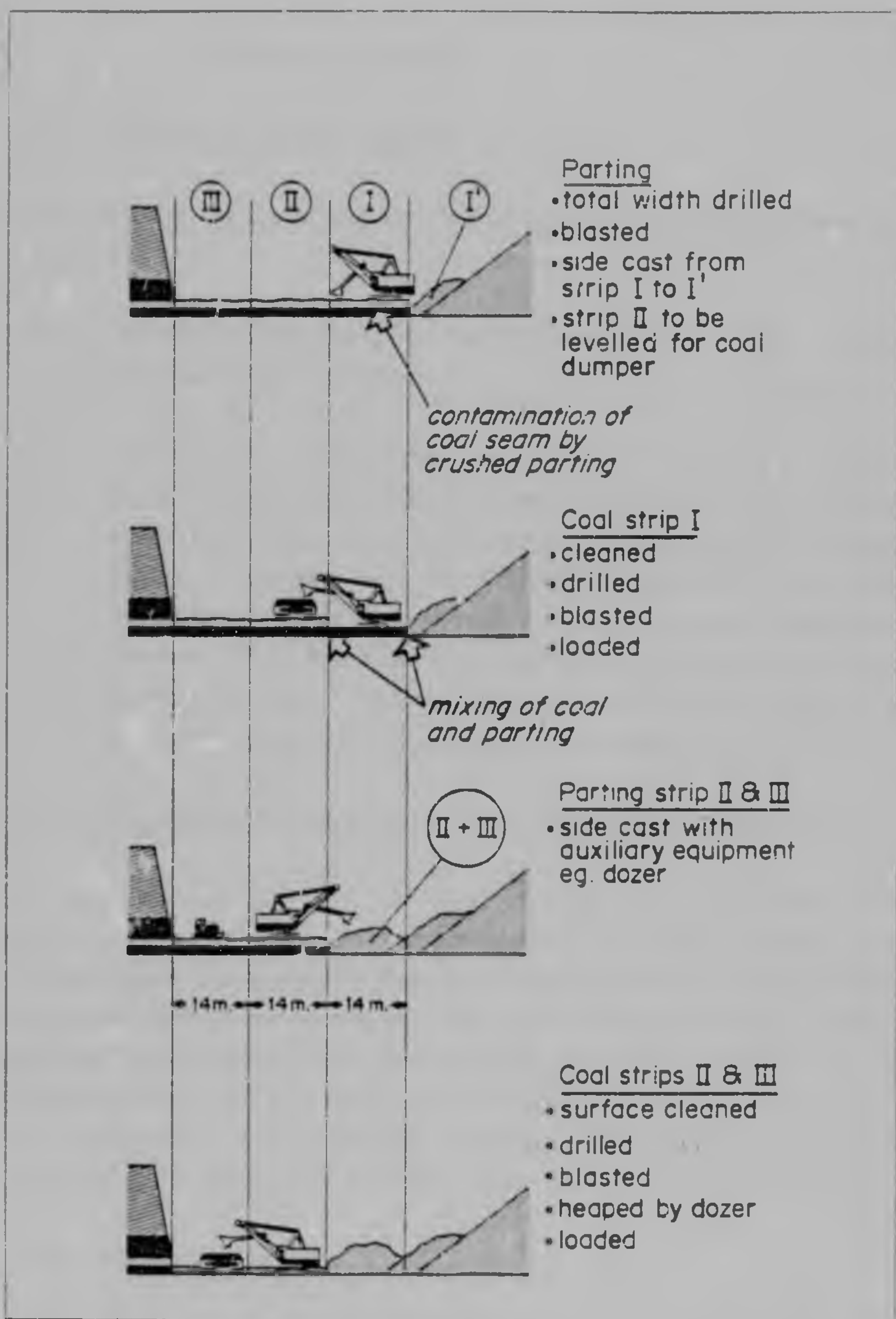

FIGURE 6.7

COAL LOADING USING ROPE SHOVELS

disturb the surface of the underlying coal, creating additional fine coal.

6.4.2 Hydraulic Excavator Method

The hydraulic excavator mining method can be illustrated as shown in Figure 6.8.

(a) As for the rope shovel, the top coal seam is drilled, blasted and loaded into coal haulers.

(b) The 42 metre wide parting cut can be handled in only 2 strips as the reach of the backactor type hydraulic excavator is much more than that of the rope shovel. The tear out and prying forces of this excavator are 850 kN (Brueggemann et al, 1983) which enable it to dig and load unblasted or lightly fragmented material. The backactor also operates from the top of the parting horizon, therefore reducing the possibility of fine coal generation caused by the tracks of the loader.

6.5 Advantages of Using the Hydraulic Excavator Loading Method

Brueggemann et al (1983) introduced the system of hydraulic excavators on a large South African opencast coal mine located in the Witbank Area and concluded that "the subsequent use of the hydraulic excavator resulted in considerable savings in production costs. With a backhoe bucket capacity of 7,5m^3, a tear out force of 460 kN and a prying force of 500 kN, the pattern of blasting could be extended to a 2 x 10m grid at 0,8m thickness of parting. This alone resulted in a 70% saving of drilling and blasting costs - or about R350 000 a year.

Additional savings included:

- coal contamination reduced by 35%
- higher productivity,

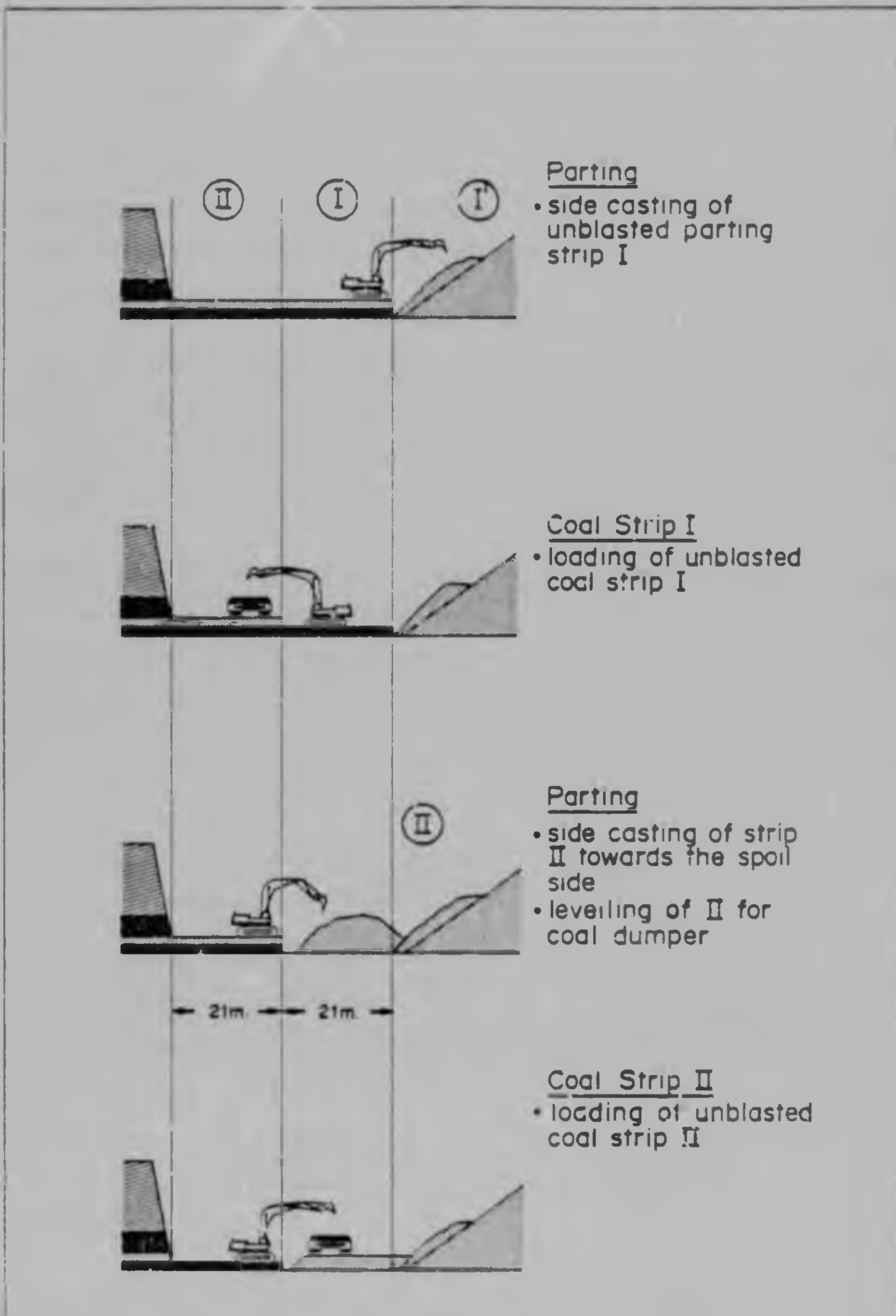

FIGURE 6.8

COAL LOADING USING HYDRAULIC EXCAVATORS

- auxilliary equipment reduced,

- improved usage of spoil area due to bigger digging reach"

Singhal (1983) compares equally sized bucket rope shovels and hydraulic excavators as illustrated in Figure 6.9 and states that "for a similar size of bucket the weight of a power shovel is more than twice that of a hydraulic shovel.

Because of its digging pattern a power shovel must lift its combined weight and payload a much greater distance after cutting free. This generates a momentum requiring a heavy counterweight on the power shovel.

Unlike the shovels, hydraulic cylinders of a hydraulic shovel can apply forces in two directions giving hydraulic machines their flexibility; the 'wrist action' of this type of machine is very useful in selective mining contributing to a reduction in downstream cost of screening and processing."

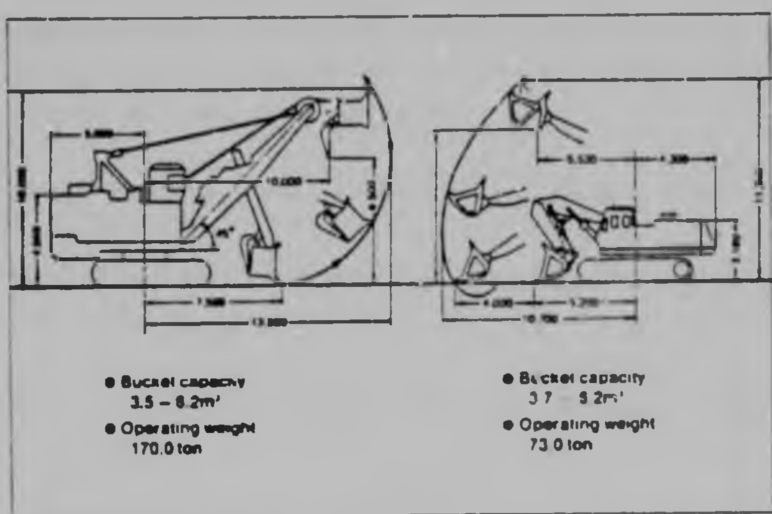

FIGURE 6.9

COMPARISON OF POWER SHOVEL AND HYDRAULIC EXCAVATOR WEIGHTS AND LOADING ACTIONS

6.6 Coal Transportation

The mining engineer will be faced with the necessity of having to make a haulage study to determine not only the most suitable method of hauling coal from the working face to the discharge point, but also the most effective and economical type of equipment to use for this particular operation.

This chapter outlines a general description of various types of haulage units and their components, guidelines on the selection of equipment, material both operational and cost wise to be accumulated, and the necessary reference material.

6.6.1 Trucks

The following types of off-highway haulage equipment are available.

- Rear dump trucks : these units having a body mounted on the truck chassis that is raised by means of an integrally mounted hydraulic hoist system is the most popular type of rubber-tired haulage vehicle. It can haul most any type of material but is generally used to handle heavy and bulky material such as blasted rock, ore, shale and other.

  Within this type there are two different classifications:

  (i) Two-axle, wherein one or both axles could be of the driving type.

  (ii) Three-axle, wherein only the rear two or bogie axles drive.

- Bottom Dump Trucks : These units consist of a prime mover or tractor coupled with a semitrailer having drop-bottom or clam-shell doors. The trailer hopper is generally wider and deeper at the front to place more weight on the drive axle and is wider at the top than at the bottom so that the side walls of the trailer slope inwards.

It has been designed to handle free flowing material since large pieces of rock or other bulky material might jam in the smaller bottom opening when the load is dumped.

In selecting a coal hauling system, the first step is therefore to decide between a rear dump truck and a bottom dump truck. The following information is intended as a guide for the engineer in the selection of the most suitable type of haulage equipment for the proposed application:

(a) Truck body sizes

All truck manufacturers furnish a standard body size and configuration for each of their specific models. These bodies have an SAE body capacity rating in cubic metres struck and heaped at a 2:1 slope.

In Table 6.4 a comparison of standard body sizes for various truck types and sizes are illustrated.

TABLE 6.4
TRUCK BODY SIZES (EUCLID, 1983; CATERPILLAR, 1983)

	TRUCK BODY CAPACITIES (m^3)	
PAYLOAD (tons)	REAR DUMP TRUCK	BOTTOM DUMP TRUCK
35	23,5	-
45,4	34,1	-
77,1	51,3	-
91	54,7	100,0
109	65,4	121,6
154,2	98,0	167,7

The relative density of broken coal ranges rom 0,9 to 1,1 ton per cubic metre (Geyser, 1983), resulting in under utilization of the rated payload capacity if rear dump trucks are used.

If rear dump trucks are however required to haul coal, for some specific reasons, in order to permit the haulage unit to handle its rated load, it will be necessary to increase the body volume by the addition of top extensions or sideboards. In many cases these extensions are not offered by the manufacturer as options and thus it becomes the responsibility of the distributor or owner to provide and install them at additional costs.

(b) Load Discharge Height

As mentioned, the generation of fine coal should be limited in all aspects of the mining operation, and this factor must therefore also be considered when selecting hauling equipment.

The reduced height which the coal load is dropped by using bottom dump trucks is illustrated in Table 6.5

TABLE 6.5

DISCHARGE HEIGHT FOR 154 TON TRUCKS

	REAR DUMP TRUCK	BOTTOM DUMP TRUCK
Top of truck body (m)	8,25m	3,96m
Discharge height (m)	1,30m	0,81m

Taking into account the abovementioned factors of truck body capacities and load discharge heights, bottom dump trucks should be considered as prime coal hauler for opencast coal mines in the Witbank Area.

(c) Truck size and numbers

In selecting the appropriate fleet of bottom dump trucks, the same criteria as described in section 5.6 of this project report apply.

6.6.2 Belt Conveyors

The demands of industry for handling materials at higher rates and at reduced costs have had a very significant effect on belt conveyor technology. Each component comprising the conveyor system has been continuously improved through design and materials technology to enable present conveyor systems to operate at ever increasing rates and reliability. These technological advances have therefore enabled the belt conveyor to not only attain acceptance as a method for in-plant movement of bulk materials, but also to be used for long distance transportation of bulk materials, particularly in areas of adverse terrain.

The information contained in this section of the Project Report is intended to assist the Mining Engineer to make basic decisions regarding the use and application of conveyor systems, because the design of an overland conveyor transportation system is site-specific and requires more detailed engineering design.

(a) Conveyor Capacities

One of the most important factors in determining the proper size of a conveyor system is the rate at which material must be handled. Although tables and charts are available to the conveyor designer to assist in selection of conveyor components, only the designer's knowledge of the present and future operating requirements of the conveyor system and the interrelation of other process machinery in the flow will enable him to establish a peak rate for handling material which will result in optimum economy and performance (Van Kleunen, 1968). For example, the required annual production rate divided by the scheduled annual operating hours would not produce an hourly capacity on which the conveyor system should be based. The prudent designer will allow for reduced operating days due to inclement weather, equipment downtime, routine maintenance, and other local factors which may affect the scheduled plant operation. The means by which material is brought to the conveyor system should also be considered when establishing the peak conveyor capacity.

(b) Characteristics of material which affect the selection of a conveyor System

(i) Density: Since the conveyor is a volumetric transporting medium, the density of the material being handled governs the resulting tonnage rating of the conveyor system.

(ii) Maximum lump size: Material lump size may dictate the selection of a belt width greater than that needed to convey the required tonnage. Material is ultimately loaded onto conveyors by means of a transfer chute and skirtboards which direct the flow of the material in the direction of belt travel. A guide to the relationship between lump size and belt width is shown in Table 6.6 (Van Kleunen, 1968).

(iii) Size Consist: Size consist affects belt width, belt speeds and the slope on which the material can be loaded and conveyed. The relative absence of fines reduces the predictability of material negotiating the steeper conveyor inclines, particularly when the material "tails off" at the end of an operating run. Coarse, jagged material without fines should be handled at moderate speeds to reduce belt cover wear at conveyor transfers.

As shown in Table 6.7, the general rule is to reduce belt speeds when the amount of fines in the material is reduced.

TABLE 6.6
SUGGESTED MINIMUM BELT WIDTH IN RELATION TO LUMP SIZE
DISCHARGE BETWEEN VERTICAL SKIRTBOARDS

TYPE OF MATERIAL	RATIO BELT WIDTH TO LUMP SIZE	BELT WIDTH (cm) 36	46	61	76	91	107	122	137	152	183
		MAXIMUM LUMP SIZE (cm) LONGEST LUMP DIMENSION									
		Idler Troughing Angle 20° - 30°									
ROM Material not more than 1% maximum lump	2 1/4	15	20	28	33	41	48	53	61	68	51
Balance 1/2 max. lump, or less	2 1/4	13	15	20	25	30	35	41	46	51	61
Occasional Lump		Any Troughing Angle									
Not more than 6% max. lump; balance 1/2 max. lump, or less	3	13	15	20	25	30	36	41	46	51	61
Crusher Product, Longest dimension	3 1/2	10	13	18	23	25	30	36	38	43	53
Crusher Product, Fines Removed	4	10	13	15	20	23	28	30	33	38	46
Sized Material from Screening Operation	4 1/2	8	10	13	18	20	23	28	30	33	41

TABLE 6.7

MAXIMUM RECOMMENDED BELT SPEEDS (METRES PER MINUTE)

FOR STANDARD SERVICE

(VAN KLEUNEN, 1968)

	MAXIMUM SPEED (m/min)	BELT WIDTH (cm) 36	46	61	76	91	107	122	137	152	183
Granular Fines											
Minus 25mm lump	305	122	152	183	213	243	274	305	305	305	305
Occasional lump 10% belt width	274	122	152	183	213	229	243	274	274	274	274
Half Maximum Sized Lump											
Rounded Pieces	244	91	122	168	198	198	213	244	244	244	244
Abrasive, Sharp	213	91	122	152	183	198	198	213	213	213	213
Maximum Sized Lump											
Rounded Pieces	200	91	122	137	152	168	183	198	198	198	198
Abrasive, not sharp	183	91	122	137	152	168	168	183	183	183	183
Abrasive, sharp	168	76	91	107	122	137	152	168	168	168	168
To Reduce Breakage											
Friable ores	152	76	91	107	107	122	137	152	152	152	152
Coal	122	76	76	91	91	107	122	122	122	122	122
Coke	91	76	76	76	76	76	91	91	91	91	91
To Reduce Dusting											
Heavy fines	91										
Light, bone dry fines	76										

(iv) Degradation and Dusting: Friable material, like the coal material from the Witbank Area, should be handled at reduced speeds if degradation of the material is a factor.

(c) Types of Belt Conveyors

Several types of belt conveyor constructions have been developed to meet specific applications. The proper selection of a conveyor construction will result in maximum economy and operating efficiency. The following types of conveyor systems are available:

- Pre-engineered conveyors
- Conventional stringer and deck conveyors
- Wire rope conveyors
- Shiftable conveyors
- Regenerative conveyors
- Compound conveyors
- Cable conveyors

As result of the dynamic nature of the working faces of an opencast mining operation, shiftable conveyors are the only type of conveyors that need to be considered in the Witbank Area for coal transportation. The design of the shiftable conveyor enables the entire conveyor to be moved laterally without dismantling any component. It is mounted on steel or timber skids set transversely to the conveyor and inter connected by a shifting rail running the length of the conveyor. The conveyor is shifted in a series of lateral steps ranging from 1,5 to 2,0 metres per step, depending upon terrain. Lateral shifting is usually done by using a track-type tractor fitted with a special roller equipped shifting mechanism which locks onto the head of a shifting rail. (Dippenaar et al 1983).

To prevent operational interference from other mining activities such as overburden stripping, and parting removal, the shiftable conveyor system can only be positioned at surface elevation either on the highwall or spoil pile side of the dragline cut. If it is positioned on the spoil side, levelling of these spoil piles must take place as soon as possible after stripping. If the conveyor is position at the highwall side, access ramps from the highwall will be required to link the conveyor belt and the coal producing faces.

Apart from the conveyor structure itself the following additional equipment will be required:

(i) Haul trucks to transport coal from the working face to a discharge point.

(ii) A coal discharge facility such as a feeder-breaker or mobile crusher that will crush the coal to an acceptable size for transporatation by belt.

(iii) Transfer points. As it will be unlikely that a straight line conveyor system and lay-out can be maintained, transfer points must be provided where the conveyor belt changes direction.

(iv) Each conveyor belt section will require seperate drive systems.

6.7 Conclusions and Recommendations

Standardization of mining methods and equipment on any operation is of prime importance in an effort to reduce operating costs. It is therefore recommended that the hydraulic excavator be considered for soft overburden removal, coal mining and parting removal. This unit has proved to be extremely successful in thin layered material such as the coal seams and parting horizons in the Witbank Area. The break-out force developed by the bucket arrangement and hydraulic cylinders allows lower explosive powder factors to be used resulting in cost savings on drilling and blasting and it also has the advantage of reducing the possibility of creating excessive fine coal which is unacceptable as a final product.

Although various coal transportation systems are available, bottom dump coal haulers still provides a practical solution at comparable costs and should be considered as basis of design for all opencast projects in the Witbank Area.

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CHAPTER 7

GENERAL CONCLUSIONS

The bulk of South Africa's coal, both for the inland market and for various overseas users, is produced from the well-known Witbank coalfield. During recent years some very large opencast mines have been established in this area and a number of projects are at present being evaluated and planned by various mining companies. The author therefore felt the need to publish this project report, not only to identify the various mining processes, but also to highlight the need to investigate alternative solutions to the problem in an effort to optimize return on capital investments.

South African mining regulations require the removal of topsoil and soft overburden to a minimum depth of one metre as a separate operation and the use of this material in rehabilitation of the disturbed land. Various classification systems to determine the excavatability of material have been used in the past. Probably the most common being the relationship between seismic velocity and rippability as derived by the Caterpillar Tractor Company (1978) for the use in track dozer and scraper operations. Due to the fact that other mining methods are competitive to the traditionally used dozer-scraper method, a different excavatability classification system was required. The system proposed by Kirsten (1982) proved to cater for all rock types, from the weakest soils to the hardest rock. By using this classification system, it is evident that the topsoil material of the Witbank Area can be classified as very soft and that it can be excavated succesfully with front-end loaders, hydraulic excavators or rope shovels.

The sandstones, shales and siltstones overlying the various coal seams are relatively hard, therefore requiring drilling and blasting prior to stripping. To select a suitable drilling unit that will give optimum results, one of the basic parameters to establish is the rock properties to be encountered during drilling operations. Test work by Praillet (1983) indicated that penetration rate is a function of pulldown force on the drill bit, bit diameter, bit rotation speed and rock compressive strengths. The latter being fixed for a given mining area, drill selection is therefore dependent on the rock hardness and the required production rates. The 250 mm diameter rotary drill, which is in general use in the Witbank Coalfield, proved not only to be cost effective, but also provides the mining operation with greater flexibility even for very high annual production rates.

As a result of ever increasing coal consumption throughout the world and diminishing shallow coal reserves, surface coal mines are forced to mine to greater depths; consequently, stripping operations are accounting for a larger proportion of the total operating cost and are therefore ever more critical. Historically, larger equipment was automatically selected to combat increasing costs and greater depths. However, a more reliable and cost effective mining system may consist of smaller equipment that has greater flexibility and proven capabilities. This project report describes in detail the selection criteria to be followed when selecting draglines, electric rope shovels, front-end loaders, hydraulic excavators and haultrucks for overburden stripping. By comparing these alternatives for different annual stripping rates, it has been proved in this report that, in spite of the fact that the initial capital outlay required for a dragline operation is higher than that of the other methods considered, draglines still provide a lower unit cost per cubic metre over the project life. It is also significant that the savings in a dragline operation become more significant with increasing annual production rates. This can be attributed to the fact that the other methods investigated are dependent on haultrucks for the transportation of overburden from the working face to the spoil pile area. These haultrucks consume large quantities of fuel which cause operating costs to increase.

The coal seams of the Witbank Area are inconsistent in thickness and the individual seams are often split by hard sandstone, siltstone or shale partings. As a result of this inter-relationship between coal seams and interbedded parting it is important to select a mining method and the associated equipment that will excavate both the coal seams and the parting horizons successfully at an economic cost without creating excessive fines. An investigation and simulation of the rope shovel and hydraulic excavator loading methods indicate that the hydraulic excavator has the potential to excavate thin layers of material. It also has the capability of a higher breakout force than the rope shovel and provides the mine operator with a similar capacity bucket to the rope shovel but at a much smaller machine weight. These factors combined, make the hydraulic excavator an ideal loading unit for both coal and parting horizons. An additional benefit is the fact that it can be used economically to excavate topsoil and soft overburden. Standardization of methods and equipment on any mining operation is of prime importance in an effort to reduce operating costs.

Although various coal transportation systems are available for delivering coal from the working face to the discharge point at the primary crusher, bottom dump coal haulers still provides a practical solution at comparable costs and should be considered as basis of design for all opencast projects in the Witbank Area.

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APPENDIX I

DETERMINATION OF VARIOUS INDIVIDUAL ROCK RATINGS TO DETERMINE RIPPABILITY AS PROPOSED BY KIRSTEN (1982).

The excavatability index N is defined by Kirsten (1982) as

$$N = Ms\ \frac{RQD}{Jn}.\ Js.\ \left(\frac{Jr}{Ja}\right),\ \text{where}$$

Ms	=	Mass strength or consistency number
$\frac{RQD}{Jn}$	=	Block size number
Js	=	Relative ground structure number.
$\frac{Jr}{Ja}$	=	Joint strength number

This section details the ratings for the basic characteristic parameters in the expression above to determine the excavatability index.

1. MASS STRENGTH OR CONSISTENCY NUMBER, Ms:

The five point scales for the consistency of soils and rocks are shown in Tables A1.1, A1.2 and A1.3. A five point scale similar to that for granular soils was developed by Kirsten (1982) for detritus as shown in Table A1.4.

The five point scales for the different material types are summarized together in Table A1.5

The average mass strength numbers given in Table A1.5 represent rounded off products of the respective uniaxial strengths and the coefficients of relative density.

TABLE A1.1
MASS STRENGTH NUMBER FOR GRANULAR SOILS (Ms)

CONSISTENCY	IDENTIFICATION IN PROFILE	SPT* BLOW COUNT	MASS STRENGTH NUMBER (Ms)
Very loose	Crumbles very easily when scraped with geological pick	0 - 4	0,02
Loose	Small resistance to penetration by sharp end of geological pick	4 - 10	0,04
Medium dense	Considerable resistance to penetration by sharp end of geological pick	30 - 50	0,09
Dense	Very high resistance to penetration of sharp end of geological pick - requires many blows of pick for excavation	50 - 80	0,19
Very dense	High resistance to repeated blows of geological pick - requires power tools for excavation		0,41

* Standard Penetrometer Test.

TABLE A1.2
MASS STRENGTH NUMBER FOR GRANULAR SOILS (Ms)

CONSISTENCY	IDENTIFICATION IN PROFILE	VANE SHEAR STRENGTH (kPa)	MASS STRENGTH NUMBER (Ms)
Very soft	Pick head can easily be pushed into the shaft of handle. Easily moulded by fingers	0 - 80	0,02
Soft	Easily penetrated by thumb; sharp end of pick can be pushed in 30mm-40mm; moulded by fingers with some pressure	80 - 140	0,04
Firm	Indented by thumb with effort; sharp end of pick can be pushed in up to 10mm; very difficult to mould with fingers. Can just be penetrated with an ordinary hand spade	140 - 210	0,09
Stiff	Penetrated by thumbnail; slight indentation produced by pushing pick point into soil; cannot be moulded by fingers. Requires hand pick for excavation	210 - 350	0,19
Very stiff	Indented by thumbnail with difficulty; slight indentation produced by blow of pick point. Requires power tools for excavation	350 - 750	0,41

NOTE: A cohesive material of which the vane shear strength is larger than 750 kPa shall be taken as a rock, for which the hardness can be obtained from Table A1.3.

TABLE A1.3
MASS STRENGTH NUMBER FOR ROCKS (Ms)

HARDNESS	IDENTIFICATION IN PROFILE	UNCONFINED COMPRESSIVE STRENGTH (MPa)	MASS STRENGTH NUMBER (Ms)
Very soft rock	Material crumbles under firm (moderate) blows with sharp end of geological pick and can be peeled off with a knife; it is too hard to cut a triaxial sample by hand	1,7 1,7- 3,3	0,87 1,86
Soft rock	Can just be scraped and peeled with a knife; identations 1mm to 3mm show in the specimen with firm (moderate) blows of the pick point	3,3- 6,6 6,6- 13,2	3,95 8,39
Hard rock	Cannot be scraped or peeled with a knife; hand-held specimen can be broken with hammer end of a geological pick with a single firm (moderate) blow	13,2- 26,4	17,70
Very hard rock	Hand-held specimen breaks with hammer end of pick under more than one blow	26,4- 53 53,0-106,[illegible]	35,0 70,0
Extremely hard rock (very, very hard rock)	Specimen requires many blows with geological pick to break through intact material	106,0-212,0 212,0	[illegible],0 280,0

TABLE A1.4
MASS STRENGTH NUMBER FOR DETRITUS (Ms)

CONSISTENCY	IDENTIFICATION IN PROFILE	IN SITU DEFORMATION MODULUS' (MPa)	MASS STRENGTH NUMBER (Ms)
Very loose	Detritus very loosely packed. High percentage voids and very easily dislodged by hand. Matrix crumbles very easily when scraped with geological pick. Ravelling often occurs in excavated faces	0 - 4	0,02
Loose	Detritus loosely packed. Some resistance to being dislodged by hand. Large number of voids. Matrix shows small resistance to penetration by sharp end of geological pick	4 - 10	0,05
Medium dense	Detritus closely packed. Difficult to dislodge individual particles by hand. Voids less apparent. Matrix has considerable resistance to penetration by sharp end of geological pick	30 - 80	0,10
Dense	Detritus very closely packed and occasionally very weakly cemented. Cannot dislodge individual particles by hand. The mass has a very high resistance to penetration by sharp end of geological pick - requires many blows to dislodge particles	10 - 30	0,21
Very dense	Detritus very densely packed and usually cemented together. The mass has a high resistance to repeated blows of geological pick - requires power tools for excavation	80 - 200	0,44

NOTE
1. Determined by plate bearing test of diameter 760mm
2. A detritus of which the in situ deformation modulus exceeds 200 MPa shall be taken as the lowest boulder formation

DERIVIATION OF MASS STRENGTH NUMBERS FROM UNIAXIAL STRENGTH AND DRY DENSITY

MATERIAL TYPE	CONSISTENCY		AVERAGE UNIAXIAL STRENGTH (MPa)	TYPICAL DRY UNIT DENSITY (Kg/m^3)	COEFFICIENT OF RELATIVE DENSITY	AVERAGE MASS STRENGTH NUMBER Ms
Granular/cohesive soil	Very loose/very soft		0,035	1429	0,52	0,02
	Loose/soft		0,069	1571	0,57	0,04
	Medium dense/firm		0,14	1735	0,63	0,09
	Dense/stiff		0,28	1378	0,68	0,19
	Very dense/very stiff		0,55	2010	0,73	0,41
Detritus	Very loose		0,035	1684	0,61	0,02
	Loose		0,069	1816	0,66	0,05
	Medium dense		0,14	1959	0,71	0,10
	Dense		0,28	2071	0,75	0,21
	Very dense		0,55	2173	0,79	0,44
Rock	Very soft rock	(a)	1,1	2173	0,79	0,87
		(b)	2,2	2316	0,84	1,9
	Soft rock	(a)	4,4	2450	0,89	4,0
		(b)	8,8	2622	0,95	8,4
	Hard rock		17,7	2755	1,00	17,7
	Very hard rock	(a)	35,4	2755	1,00	35,0
		(b)	70,7	2755	1,00	70,0
	Extremely hard rock	(a)	141,3	2755	1,00	140,0
		(b)	282,6	2755	1,00	280,0

2. BLOCK SIZE NUMBER, RQD/JN

This number is a function of the rock quality designation (RQD) and the number of different joint sets, Jn. RQD is conventionally related to rock mass quality. It represents the percentage of the total length of NX core which comprises separate lengths of core 100mm and longer. Following from this definition RQD can be said to provide a measure of the number of joints per unit volume of the material. Barton et al (1974) has shown that RQD is empirically related to the number of joints per cubic metre, J_c, by the following expression:

$$RQD = 115 - 3{,}3\ J_c$$

The formula allows for an equivalent RQD to be determined for soils. The above expression is given in tabular form in Table A1.6. For jointed soils the actual number of joints per cubic metre should be counted. In the case of intact granular soils, gravels or detritus the equivalent RQD should be taken as 5.

In cemented materials the equivalent RQD is equal to 100.

For jointed rock the RQD may be determined either in the conventional way from core samples or from an actual count of the total number of joints per cubic metre. The minimum value for RQD in all cases should not be less than 5.

Only joints which affect the excavation process should be taken into account.

The joint set number, Jn, is tabulated in Table A1.7

TABLE A1.6
JOINT COUNT NUMBER (J_c)

NUMBER OF JOINTS PER CUBIC METRE (J_c)	GROUND QUALITY DESIGNATION (RQD)	NUMBER OF JOINTS PER CUBIC METRE (J_c)	GROUND QUALITY DESIGNATION (RQD)
33	5	18	55
32	10	17	60
30	15	15	65
29	20	14	70
27	25	12	75
26	30	11	80
24	35	9	85
23	40	8	90
21	45	6	95
20	50	5	100

TABLE A1.7
JOINT SET NUMBER (J_n)

NUMBER OF JOINT SETS	JOINT SET NUMBER (J_n)
Intact, no or few joint/fissures	1,00
One joint/fissure set	1,22
One joint/fissure set plus random	1,50
Two joints/fissure set	1,83
Two joints/fissure set plus random	2,24
Three joints/fissure set	2,73
Three joints/fissure set plus random	3,34
Four joints/fissure set	4,09
Multiple joints/fissure set	5,00

NOTE: For intact granular materials take $J_n = 5{,}00$

3. RELATIVE GROUND STRUCTURE NUMBER

The relative orientation of the ground structure and the spacing of the structural features affect the effort required to penetrate the ground as well as the effort required to dislodge individual blocks. It is easier to rip the ground in the direction in which the joints dip than in the opposite direction.

The kinematic possibility of penetration along a structural feature is directly related to the gradient of the feature. Since more than one set of joints may divide the ground, the overall kinematic possibility of penetration may be taken as the average of the respective gradients weighted by the number of features per unit length. In practical situations the blocks or slabs into which the ground is divided may be considered to be demarcated by two sets of joints. Let θ and ψ denote the respective gradients of the two sets relative to the direction of ripping and S_θ and S_ψ the respective spacings as shown in Figure A1.1. The kinematic possibility of penetration, K_p, may then be expressed as follows:

$$K_p = \frac{S_\psi \tan\theta + S_\theta \tan\psi}{a(S_\psi + S_\theta)}$$

The difference $(1 - K_p)$ represents the effort $J^P{}_S$ required to rip the ground with regard to the possibility of gaining penetration. It follows that:

$$J^P{}_S = 1 - \frac{S_\psi \tan\theta + S_\theta \tan\psi}{a(S_\psi + S_\theta)}$$

Consider next the effort, $J^d{}_d$ required to rip the ground with regard to the possibility of dislodging individual blocks once penetration has been gained. Undisturbed blocks of material are free to be displaced in a direction perpendicular to the ground surface. This principal degree of freedom can be resolved into components parallel to the sides of the block as shown in Figure A1.1(b). The dislodging action can be represented by a unit horizontal force behind the block.

The respective products of the coaxial components of this force and the degrees of freedom parallel to the sides of the block may be defined as the kinematic possibility of dislodgement, K_d. It will be noted from Figure A1.1(b) that only one of these products is non-zero, the other one comprising components of force and freedom in opposite directions. Thus

$$K_d = \left| \frac{\cos\psi \sin\theta}{b \sin^2(\psi - \theta)} \right|$$

The difference $(1 - K_d)$ represents the parameter $J^d{}_s$. It therefore follows that

$$J^d_s \quad = \quad 1 - \left| \frac{\cos\theta \sin\theta}{b \sin^2(\psi - \theta)} \right|$$

The expressions for J^p_s and J^d_s may be combined as a product representing the effect of the relative block shape and orientation on the effort required to rip. The resulting parameter may be defined as the relative ground structure number, $J_s = J^p_s \times J^d_s$. The relative block shape is defined by the ratio $r = s \quad s_0$. Thus

$$J_s \quad = \quad \left[1 - \frac{r \tan\theta + \tan\psi}{a(r + 1)} \right] \left[1 - \left| \frac{\cos\theta \sin\theta}{b \sin^2(\psi - \theta)} \right| \right]$$

For practical purposes the dip angle, DA, and the dip direction, DD, of the joints are used rather than the gradients 0 and defined above. In this notation DA lies between 0° and 90° and DD is either 0° or 180° relative to the direction of ripping. The values of J_s for these ranges of DA and DD are given in Table A1.8 for a range of values of r from 1 to 1/8. It should be observed that the DA and DD given in Table A1.8 refer in all cases to the closer spaced joint set as illustrated in Figure A1.1. Beyond a ratio of r = 1/8, J_s can be shown not to change significantly. The values for J_s in Table A1.8, close to DA's of 0° and 90° have been determined by appropriate interpretation. This table is further based on values for constants a and b of 5 and 1 respectively. These have been based on empirical assessments of the effects of the direction of ripping on the efficiency of ripping.

The figures in Table A1.8 are based on the assumption that the joint sets are orthogonal. Problems of interpretation in those instances where the joint sets are not orthogonal are avoided by referring only to the DA and DD of the closer spaced joint set. The approach of adopting the orthogonal case as the norm is assumed for the sake of simplicity but mainly because the jointing may be taken to be at right angles without significant error in the large majority of practical situations.

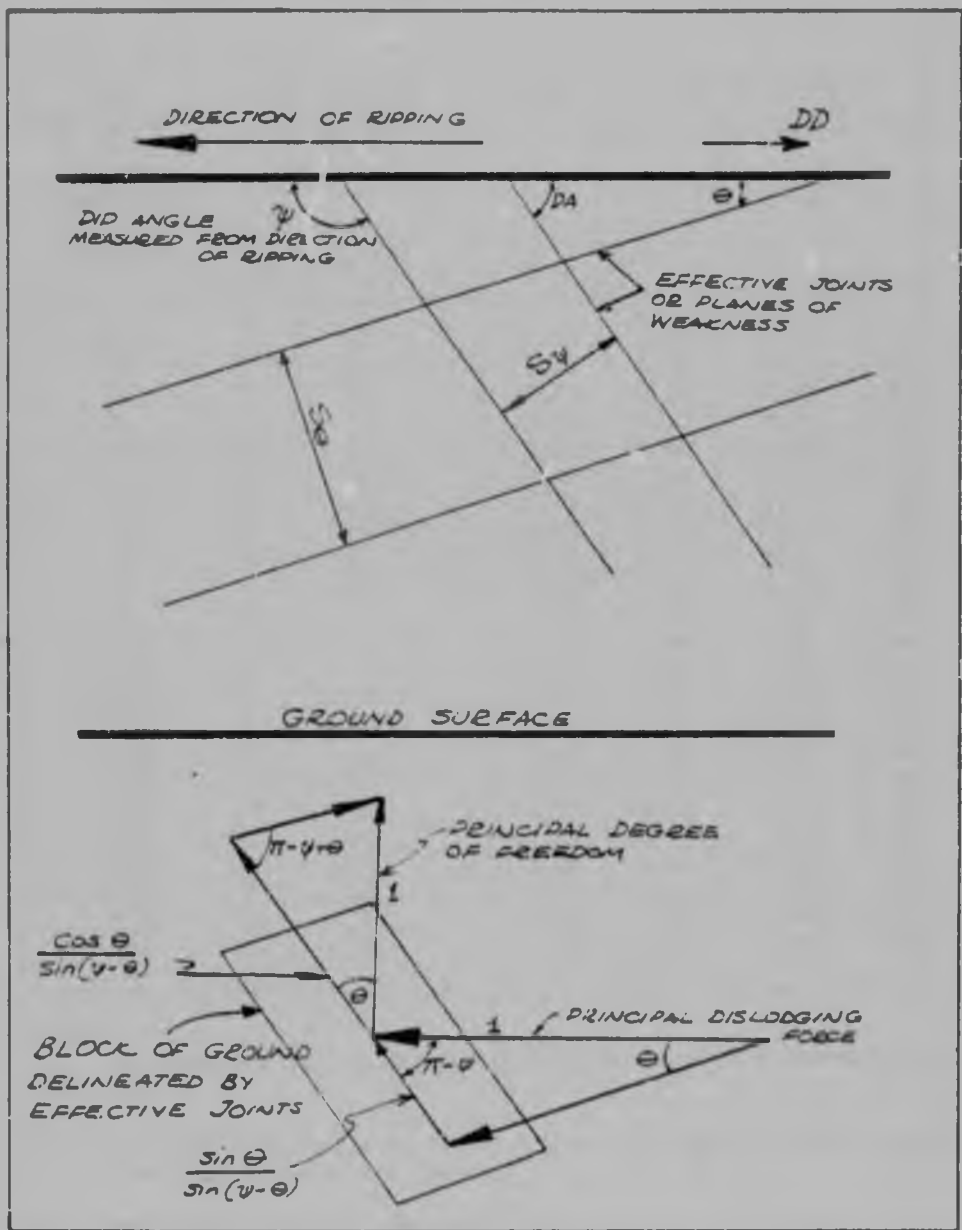

FIGURE A1.1

PRINCIPLE DEGREE OF FREEDOM AND DISLODGING FORCE RELATIVE TO BLOCK DELINEATED BY EFFECTIVE JOINTS

TABLE A1.8
RELATIVE GROUND STRUCTURE NUMBER (J_s)

DIP DIRECTION[1] OF CLOSER SPACED JOINT SET (DEGREES)	DIP ANGLE[2] OF CLOSER SPACED JOINT SET (DEGREES)	RATIO OF JOINT SPACING, r			
		1:1	1:2	1:4	1:8
180/0	90	1,00	1,00	1,00	1,00
0	85	0,72	0,67	0,62	0,56
0	80	0,63	0,57	0,50	0,45
0	70	0,52	0,45	0,41	0,38
0	60	0,49	0,44	0,41	0,37
0	50	0,49	0,46	0,43	0,40
0	40	0,53	0,49	0,46	0,44
0	30	0,63	0,59	0,55	0,53
0	20	0,84	0,77	0,71	0,68
0	10	1,22	1,10	0,99	0,93
0	5	1,33	1,20	1,09	1,03
0/180	0	1,00	1,00	1,00	1,00
/180	5	0,72	0,81	0,86	0,90
/180	10	0,63	0,70	0,76	0,81
/180	20	0,52	0,57	0,63	0,67
/180	30	0,49	0,53	0,57	0,59
/180	40	0,49	0,52	0,54	0,56
/180	50	0,53	0,56	0,58	0,60
/180	60	0,63	0,67	0,71	0,73
/180	70	0,84	0,91	0,97	1,01
/180	80	1,22	1,32	1,40	1,46
/180	85	1,33	1,39	1,45	1,50
/180	90	1,00	1,00	1,00	1,00

NOTES:

1. Dip direction of closer spaced joint set relative to direction of rip.
2. Apparent dip angle of closer spaced joint set in vertical plane containing direction of ripping.
3. For intact material take $J_s = 1,0$
4. For values of r less than 0,125 take J_s as for $r = 0,125$

4. JOINT STRENGTH NUMBER, J_r/J_a:

The parameters J_r and J_a represent the roughness and the degree of alteration of the joint walls or filling materials. Since it was found by Barton et al (1974) that the function arctan (J_r/J_a) is a fair approximation to the actual shear strength that might be expected of various combinations of wall roughness and alteration products, the actual values for these parameters have been taken over by Kirsten (1982) without change from Barton (1974) for the purpose of the classification system proposed herein.

The joint roughness number is tabulated for various joint conditions in Table A1.9

TABLE A1.9
JOINT ROUGHNESS NUMBER J_r

JOINT SEPARATION	CONDITION OF JOINT	JOINT ROUGHNESS NUMBER (J_r)
Joints/fissures tight or closing during excavation	Discontinuous joint/fissures	4,0
	Rough or irregular, undulating	3,0
	Smooth undulating	2,0
	Slickensided undulating	1,5
	Rough or irregular, planar	
	Smooth planar	
	Slickensided planar	0,5
Joints/fissures open and remain open during excavation	Joints/fissures either open or containing relatively soft gouge of sufficient thickness to prevent joint/fissure wall contact upon excavation	1,0
	shattered or micro-shattered clays	1,0

NOTE: For intact granular material take J_r = 3,0

The joint alteration number is tabulated for various types of gouge and amounts of joint separation in Table A1.10 (Kirsten 1982). The values for parameter J_a are identical to those that Barton (1974) proposed, although laid out in a different format. The asterisked values in the table have been added for completeness. It is possible that agreement may not be reached on construction in respect of the particular values for J_r and J_a. No rigorous test can be proposed for the determination of these parameters on their own. As a last resort in the event of uncertainty or a lack of agreement betweeen contracting parties it can be suggested that the joint strength number be determined from the expression.

$$J_r/J_a = \arctan(\tau_p/\sigma_n),$$

representing the total friction angle.

For fissures in soils the various categories of joint separation can only be considered if the gouge is softer than the parent material. If the gouge is of the same consistency as the joint wall material, or if the fissure approaches zero separation, the joint alteration numbers for a separation larger than 5,0 mm only should be considered.

If several joint sets of different strengths are present in the material the values of (J_r/J_a) should be determined for each set. The equivalent joint strength for the mass may then be determined as the average of the different values of (J_r/J_a) weighted according to the number of joints in each set per cubic metre.

TABLE A1.10
JOINT ALTERATION NUMBER (J_a)
(KIRSTEN 1982)

DESCRIPTION OF GOUGE	JOINT ALTERATION NUMBER (J_a) FOR JOINT SEPARATION 8mm)		
	1,0 (1)	1,0-5,0 (2)	5,0 (3)
Tightly healed, hard, non-softening impermeable filling	0,75	-	-
Unaltered joint walls, surface staining only	1,0	-	-
Slightly altered, non-softening, non-cohesive rock mineral or crushed rock filling	2,0	4,0	6,0
Non-softening, slightly clayey non-cohesive filling	3,0	6,0	10,0
Non-softening strongly over-consolidated clay mineral filling, with or without crushed rock	3,0*[4]	6,0*	10,0*
Softening or low friction clay mineral coatings and small quantities of swelling clays	4,0	8,0	13,0
Softening moderately over-consolidated clay mineral filling, with or without crushed rock	4,0*	8,0*	13,0*
Shattered or micro-shattered (swelling) clay gouge, with or without crushed rock	5,0	10,0	18,0

NOTE:
1. Joint walls effectively in contact
2. Joint walls come into contact after approximately 100mm shear
3. Joint walls do not come into contact at all upon shear
4. Values asterisked added to Barton's data

APPENDIX 2

TOPSOIL REMOVAL : EQUIPMENT PRODUCTIVITIES AND COSTS

OWNING AND OPERATING COSTS.

PHOENIX PROJECT YR 2 O/BURDEN 7690/7838

PAYLOAD (KGS)	CAPACITY (KGS/CUMTR)	OWNING AND OPERATING COST (R/HR) HAULING	LOADING	SUPPORT	AVAILABILITY (%) HAULING	LOADING
31750.	2550.0	54.53	103.37	.00	100.00	100.00

NO OF LOADERS	NO OF COURSES	MATCH FACTOR CURVE	NO OF HAULERS								
1	4	15	1	2	3	4	5	6	7	8	9

COURSE DESCRIPTION	QUANTITY (CUMTR)	----HAULING UNIT---- CALC. PROD. MTR/HR	CORR. FACT	CORR. PROD. MTR/HR	LOADER MAX. LOADS /HR	1 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	2 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	3 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R
HARDS ONLY	2134400.	71.	.95	67.	20.0	67.	31805.	10220[illegible]	124.	16526.	3571674.	132.	11755.	[illegible]
HARDS & SOFTS	2668000.	71.	.95	67.	20.0	67.	39757.	6277601.	124.	20763.	4414342.	132.	14499.	[illegible]
TOTALS	4802400.						71562.	11299[illegible]		37404.	7945916.		26458.	7063159.

COURSE DESCRIPTION	QUANTITY (CUMTR)	----HAULING UNIT---- CALC. PROD. MTR/HR	CORR. FACT	CORR. PROD. MTR/HR	LOADER MAX. LOADS /HR	4 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	5 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	6 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R
HARDS ONLY	2134400.	71.	.95	67.	20.0	222.	9602.	[illegible]	243.	8801.	3309322.	250.	8530.	3672656.
HARDS & SOFTS	2668000.	71.	.95	67.	20.0	222.	12002.	3953542.	243.	11001.	4136553.	250.	10662.	4590370.
TOTALS	4802400.						21604.	6945375.		19802.	7445975.		19192.	8263026.

COURSE DESCRIPTION	QUANTITY (CUMTR)	----HAULING UNIT---- CALC. PROD. MTR/HR	CORR. FACT	CORR. PROD. MTR/HR	LOADER MAX. LOADS /HR	7 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	8 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	9 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R
HARDS ONLY	2134400.	71.	.95	67.	20.0	254.	8416.	4082285.	254.	8403.	4534425.	254.	8403.	4992648.
HARDS & SOFTS	2668000.	71.	.95	67.	20.0	254.	10520.	5102857.	254.	10504.	5668035.	254.	10504.	6240810.
TOTALS	4802400.						18935.	9185142.		18907.	10202460.		18907.	11233460.

PHOENIX PROJECT YR 2 O/BURDEN 7690/7838

OPTIMUM FLEET SUMMARY NO. OF LOADERS- 1

COURSE DESCRIPTION	TOTAL QUANTITY (CUMTR)	HAULERS NO.	MACHINE HRS	LOADERS MACHINE HRS	PROD. MTR/HR	SPREAD HOURS	COST R
HARDS ONLY	2134400.	4	38407.	9602.	222.	9602.	[illegible]
HARDS & SOFTS	2668000.	4	48008.	12002.	222.	12002.	3659542.
TOTALS	4802400.		86415.	21604.		21604.	6945375.

PHOENIX PROJECT VR 2 D/BURDEN 7735/992C

PAYLOAD (KGS)	DENSITY (KGS/CUMTR)	OWNING AND OPERATING COST (R/HR) HAULING	LOADING	SUPPORT	AVAILABILITY (%) HAULING	LOADING
45400.	2500.0	79.64	199.09	.00	100.00	100.00

NO OF LOADERS	NO OF COURSES	MATCH FACTOR CURVE	NO OF HAULERS								
1	2	95	1	2	3	4	5	6	7	8	9

COURSE DESCRIPTION	QUANTITY (CUMTR)	----HAULING UNIT---- CALC. PROD. MTR/HR	CORR. FACT	CORR. PROD. MTR/HR	LOADER MAX. LOADS /HR	1 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	2 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	3 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R
HARDS ONLY	2134400.	106.	.95	101.	24.0	101.	21170.	5300620.	195.	10922.	3914256.	279.	7667.	3349449.
HARDS & SOFTS	2668000.	106.	.95	101.	24.0	101.	26462.	7375775.	195.	13653.	4692923.	279.	9559.	4184810.
TOTALS	4802400.						47632.	13278190.		24575.	8607321 1.83		17226.	7534258. 1.57

COURSE DESCRIPTION	QUANTITY (CUMTR)	----HAULING UNIT---- CALC. PROD. MTR/HR	CORR. FACT	CORR. PROD. MTR/HR	LOADER MAX. LOADS /HR	4 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	5 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	6 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R
HARDS ONLY	2134400.	106.	.95	101.	24.0	350.	6093.	3153912.	393.	5362.	3202390.	420.	5077.	3436618.
HARDS & SOFTS	2668000.	106.	.95	101.	24.0	350.	7616.	3942266.	393.	6702.	4052989.	420.	6346.	4295774.
TOTALS	4802400.						13708.	7096078.		12063.	7255379.		11423.	7732392.

COURSE DESCRIPTION	QUANTITY (CUMTR)	----HAULING UNIT---- CALC. PROD. MTR/HR	CORR. FACT	CORR. PROD. MTR/HR	LOADER MAX. LOADS /HR	7 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	8 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	9 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R
HARDS ONLY	2134400.	106.	.95	101.	24.0	430.	4958.	3751424.	435.	4904.	4100961.	436.	4897.	4485112.
HARDS & SOFTS	2668000.	106.	.95	101.	24.0	430.	6198.	4689281.	435.	6130.	5126200.	436.	6122.	5606391.
TOTALS	4802400.						11157.	8440705.		11035.	9227161.		11019.	10091500.

PHOENIX PROJECT VR 2 D/BURDEN 7738/992C

OPTIMUM FLEET SUMMARY NO. OF LOADERS- 1

COURSE DESCRIPTION	TOTAL QUANTITY (CUMTR)	HAULERS NO.	MACHINE HRS	LOADERS MACHINE HRS	PROD. MTR/HR	SPREAD HOURS	COST R
HARDS ONLY	2134400.	4	24370.	6093.	350.	6093.	3153912.
HARDS & SOFTS	2668000.	4	30465.	7616.	350.	7616.	3942265.
TOTALS	4802400.		54835.	13708.		13709.	7096078.

PHOENIX PROJECT YR 2 O/BURDEN 777/992C

PAYLOAD (KGS)	DENSITY (KGS/CUMTR)	OWNING AND OPERATING COST (R/HR) HAULING	LOADING	SUPPORT	AVAILABILITY (%) HAULING	LOADING
77100.	2500.0	132.59	199.09	.00	1[illegible].00	100.00

NO OF LOADERS	NO OF COURSES	MATCH FACTOR CURVE	NO OF HAULERS
1	2	95	1 2 3 4 5 6 7 8 9

COURSE DESCRIPTION	QUANTITY (CUMTR)	HAULING UNIT CALC. PROD. MTR/HR	CORR. FACT	CORR. PROD. MTR/HR	LOADER MAX. LOADS /HR	1 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	2 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	3 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R
HARDS ONLY	2134400.	153.	.95	146.	15.4	14[illegible].	14[illegible].	4430557.	274.	7801.	[illegible]	380.	5620.	2848703.
HARDS & SOFTS	2668000.	153.	.95	146.	15.4	145.	1[illegible].	5533191.	274.	9751.	[illegible]	380.	7025.	3560878.
TOTALS	4802400.						33044.	9968744.		17553.	7095957. (14)		12646.	6409591. 1.33

COURSE DESCRIPTION	QUANTITY (CUMTR)	HAULING UNIT CALC. PROD. MTR/HR	CORR. FACT	CORR. PROD. MTR/HR	LOADER MAX. LOADS /HR	4 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	5 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	6 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R
HARDS ONLY	2134400.	153.	.95	146.	15.4	443.	4816.	2933845.	466.	4581.	3261514.	473.	4[illegible].	3672897.
HARDS & SOFTS	2668000.	153.	.95	146.	15.4	443.	6017.	3667306.	466.	5726.	4076895.	473.	5636.	4591125.
TOTALS	4802400.						10831.	6601151. 1.37		10306.	7338409. (1.53)		10145.	8264022.

COURSE DESCRIPTION	QUANTITY (CUMTR)	HAULING UNIT CALC. PROD. MTR/HR	CORR. FACT	CORR. PROD. MTR/HR	LOADER MAX. LOADS /HR	7 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	8 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	9 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R
HARDS ONLY	2134400.	153.	.95	146.	15.4	474.	4500.	4127420.	474.	4500.	4589068.	474.	4500.	5050715.
HARDS & SOFTS	2668000.	153.	.95	146.	15.4	474.	5625.	5159278.	474.	5625.	5736338.	474.	5625.	6313397.
TOTALS	4802400.						10125.	9286698.		10125.	10325410.		10125.	11364110.

PHOENIX PROJECT YR 2 O/BURDEN 777/992C

OPTIMUM FLEET SUMMARY NO. OF LOADERS- 1

COURSE DESCRIPTION	TOTAL QUANTITY (CUMTR)	HAULERS NO.	HAULERS MACHINE HRS	LOADERS MACHINE HRS	PROD. MTR/HR	SPREAD HOURS	COST R
HARDS ONLY	2134400.	3	16[illegible]1.	5620.	380.	5620.	2848703.
HARDS & SOFTS	2668000.	3	21[illegible]6.	7025.	380.	7025.	3560878.
TOTALS	4802400.		37[illegible]7.	12646.		12646.	6409581.

PHOENIX PROJECT YR 2 "SOFTS" 5218/D9L

PAYLOAD (KGS)	DENSITY (KGS/BCMTR)	OWNING AND OPERATING COST (R/HR) HAULING	LOADING	SUPPORT	AVAILABILITY (%) HAULING	LOADING
21770.	2365.0	72.01	75.39	.00	100.00	100.00

NO OF LOADERS	NO OF COURSES	MATCH FACTOR CURVE	NO OF HAULERS								
1	2	95	1	2	3	4	5	6	0	0	0

COURSE DESCRIPTION	QUANTITY (CUMTR)	----HAULING UNIT---- CALC. PROD. MTR/HR	CORR. FACT	CORR. PROD. MTR/HR	LOADER MAX. LOADS /HR	1 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	2 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	3 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R
TOPSOIL & SOFTS	2134400.	73.	.96	69.	52.2	69.	33577.	4593937.	139.	15489.	3412307.	202.	10578.	3092245.
TOPSOIL ONLY	249030.	73.	.94	69.	52.2	69.	3599.	533778.	139.	1900.	396483.	202.	1229.	359294.
TOTALS	2383430.						34577.	5127715. 2.15 0.61		17289.	3808769 1.60 0.61		11807.	3451538 1.45

COURSE DESCRIPTION	QUANTITY (CUMTR)	----HAULING UNIT---- CALC. PROD. MTR/HR	CORR. FACT	CORR. PROD. MTR/HR	LOADER MAX. LOADS /HR	4 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	5 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	6 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R
TOPSOIL & SOFTS	2134400.	73.	.96	69.	52.2	262.	8159.	2972582.	317.	6739.	2940063.	367.	5819.	2958306.
TOPSOIL ONLY	249030.	73.	.94	69.	52.2	262.	945.	345390.	317.	793.	341612.	367.	676.	343731.
TOTALS	2382430.							3317972. 1.39 0.60		7521.	3281676 1.38 0.59		6496.	3302037. 1.34 0.40

PHOENIX PROJECT YR 2 "SOFTS" 621B/CBL

OPTIMUM FLEET SUMMARY NO. OF LOADERS- 1

COURSE DESCRIPTION	TOTAL QUANTITY (CUMTR)	HAULERS NO.	HAULERS MACHINE HRS	LOADERS MACHINE HRS	PROD. MTD/HR	SPREAD HOURS	COST R
TOPSOIL & SOFTS	2134400.	5	33690.	6738.	317.	6738.	2940063
TOPSOIL ONLY	248000.	5	3915.	783.	317.	783.	341612.
TOTALS	2382400.		37605.	7521.		7521.	3281674.

PHOENIX PROJECT YR 2 "SOFTS" 6710/DRL

PAYLOAD (KGS)	DENSITY (KGS/CUMTR)	OWNING AND OPERATING COST (R/HR) HAULING	LOADING	SUPPORT	AVAILABILITY (%) HAULING	LOADING
34020.	2240.0	108.53	104.93	.00	100.00	100.00

NO OF LOADERS	NO OF COURSES	MATCH FACTOR CURVE	NO OF HAULERS								
1	2	95	1	2	3	4	5	6	0	0	0

COURSE DESCRIPTION	QUANTITY (CUMTR)	----HAULING UNIT---- CALC. PROD. MTR/HR	CORR. FACT	CORR. PROD. MTR/HR	LOADER MAX. LOADS /HR	1 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	2 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	3 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R
TOPSOIL & SOFTS	2134400.	113.	.94	106.	52.2	106.	20090.	4285260.	213.	10040.	3232765.	312.	6850.	2928966.
TOPSOIL ONLY	248000.	113.	.94	106.	52.2	106.	[illegible]	499029.	213.	1167.	375621.	312.	796.	362644.
TOTALS	2382400.						22413.	4784289. (2.0) 0.57		11207.	3608387. (1.51) 0.83		7646.	3291559. 1.38 0.94

COURSE DESCRIPTION	QUANTITY (CUMTR)	----HAULING UNIT---- CALC. PROD. MTR/HR	CORR. FACT	CORR. PROD. MTR/HR	LOADER MAX. LOADS /HR	4 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	5 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R	6 HAULERS/ 1 LOADERS PROD. MTR/HR	SPREAD HOURS	COST R
TOPSOIL & SOFTS	2134400.	113.	.94	106.	52.2	404.	5281.	2846789.	490.	4359.	2822[illegible]	[illegible]	3761.	2843494.
TOPSOIL ONLY	248000.	113.	.94	106.	52.2	404.	614.	330774.	490.	506.	327943.	[illegible]	437.	330391.
TOTALS	2382400.						5895.	3177562. (1.53) 0.78		4865.	3150365. (1.32) 0.87		4198.	3173885. (1.53) 0.78

PHOENIX PROJECT YR 2 "SOFTS" 6310/09L

OPTIMUM FLEET SUMMARY NO. OF LOADERS- 1

COURSE DESCRIPTION	TOTAL QUANTITY (CUMTRS)	HAULERS NO.	MACHINE HRS	LOADERS MACHINE HRS	PROD. MTRS/HR	SPREAD HOURS	COST $
TOPSOIL & SOFTS	2134[illegible].	5	21792.	4359.	49[illegible].	4359.	2822423.
TOPSOIL ONLY	248300.	5	2532.	506.	49[illegible].	506.	327943.
TOTALS	2382400.		24324.	4865.		4865.	3150165.

APPENDIX 3

FRONT END LOADER SELECTION: NOMOGRAPHS

A. <u>DETERMINATION OF REQUIRED BUCKET PAYLOAD AND BUCKET SIZE</u>

<u>METHOD</u>

(i) Enter the required hourly production rate on Scale B (Figure A3.1)

(ii) Enter the number of cycles per hour on Scale A.

(iii) Draw a line through A and B to C. This intersection point (C) shows the required payload per cycle.

(iv) Enter the estimated bucket fill factor on Scale D.

(v) Connect C through D to E for the required bucket size.

(vi) Transfer the cycles per hour (Scale A) and the required payload (Scale C) to Figure A3.2.

B. <u>DETERMINATION OF PAYLOAD WEIGHT AND TONS PER HOUR.</u>

(i) Enter the material density on Scale F (Figure A3.2).

(ii) Connect C through Scale F to Scale G. The intersection of this line on Scale G will give the payload weight per cycle.

(iii) Compare the Bucket Payload (Scale G) with the recommended operating loads for different front-end loader sizes.

(iv) To determine the hourly production rate (tons per hour), draw a straight line from Scale G through Scale A to Scale I.

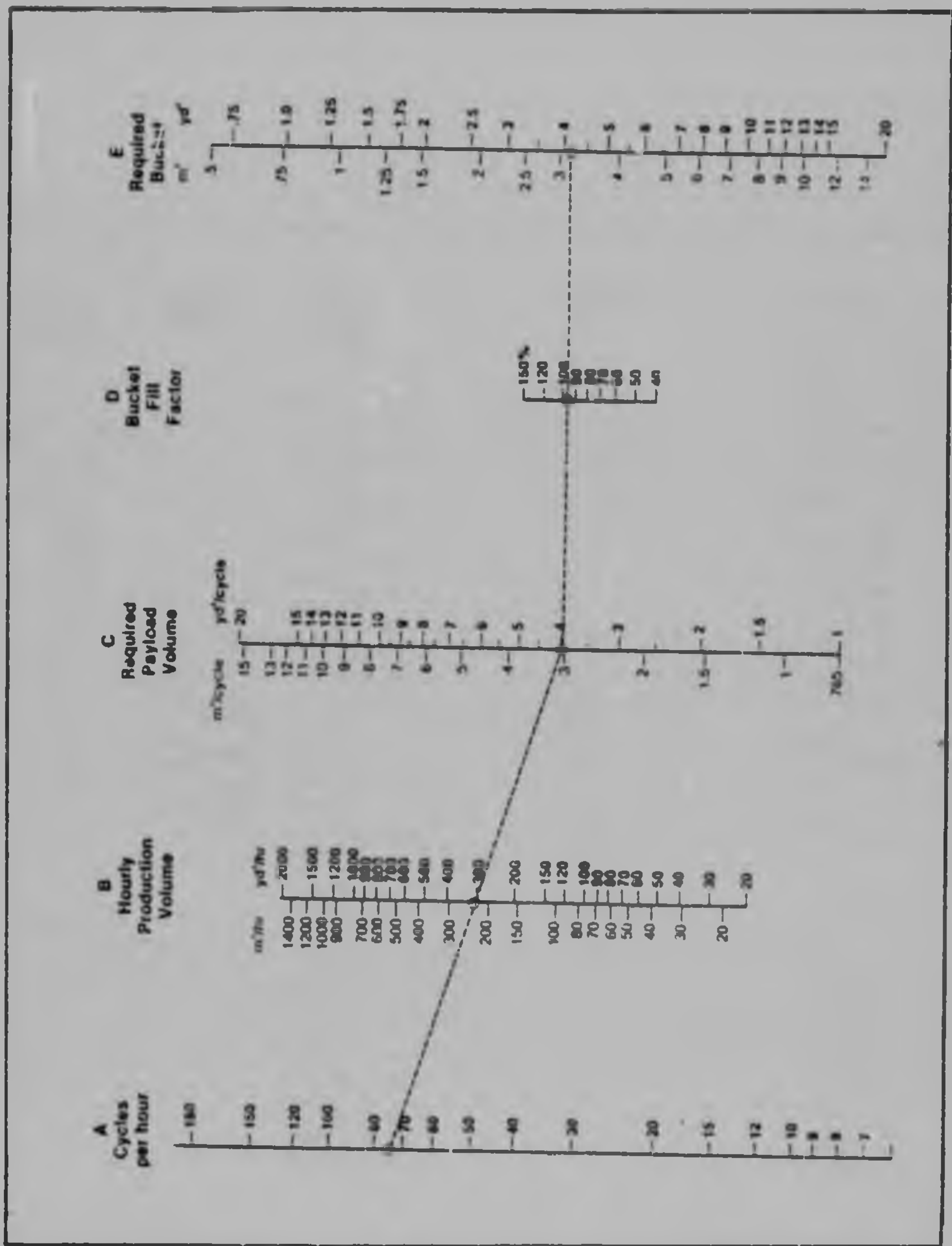

FIGURE A3.1

NOMOGRAPH TO DETERMINE PAYLOAD AND BUCKET SIZE

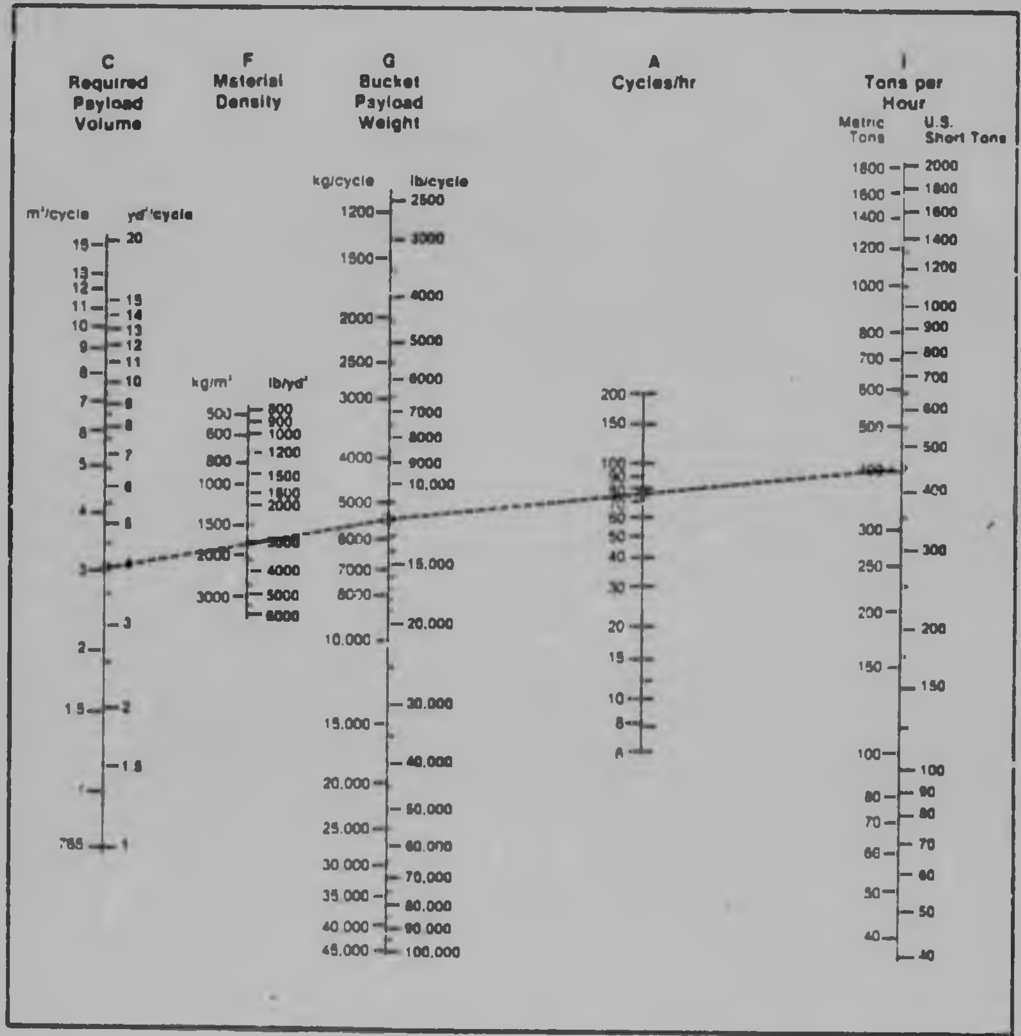

FIGURE A3.2

NOMOGRAPH TO DETERMINE PAYLOAD WEIGHT AND HOURLY PRODUCTION RATE (TONS)

APPENDIX 4

HAUL TRUCK SIMULATION STUDY

Field Application Study

Prepared For

G.A.FOURIE

Prepared By

FIELD APPLICATIONS ENGINEERING
WABCO CONSTRUCTION AND MINING EQUIPMENT
A DIVISION OF AMERICAN STANDARD INC.
PEORIA, ILLINOIS U.S.A.

Study No. 2270-0 November 12, 1982

WABCO

Prepared by W. H. Hoen

Approved by J— N B—y

PURPOSE OF THE STUDY

Determine the productivity and haulage cost for the WABCO 85D/150CT Bottom Dump Coal Hauler on haulage profiles typical of those at the Phoenix Project.

SUMMARY OF RESULTS

The 85D/150CT can move 370 metric tons per hour on the "J" profile at a cost of R0.25 per metric ton. On the "C" profile the productivity would be 343 t/hr. at a cost of R0.28/t.

ANALYSIS OF DATA

The productivity and haulage cost of the WABCO 85D/150CT HAULPAK Bottom Dump Coal Hauler was determined by using a computer simulation of the theoretical performance of the truck on the two haulage profiles. This represents WABCO's best estimate of the production capability of the truck based upon the operating conditions specified and upon the published performance characteristics of the power train components.

In this study the truck was allowed to operate at its maximum performance capability, except that the speed was limited on the curves and when returning empty down the 6% ramp grades.

The following table summarizes the production and estimated haulage cost on the two profiles. Complete details of the performance on each segment of the profiles are shown in the "Vehicle Simulation" section.

The operating costs are shown in Rands and are based upon typical operation of the 85D/150CT in U.S. mines. The repair cost has been estimated to represent that cost which the typical user should expect during the third year of operation of the truck on these profiles.

MODEL 85D/150CT HAULPAK B.D.

ENGINE	CUM KTA-2300-C	POWER 783.0 KW (1050 HP) TIRES	27.00-49,36PR,E3
TRANSMISSION	ALLISON DP-8961	FINAL REDUCTION 23.29 ROLLING RADIUS	1.314 METERS
EMPTY WEIGHT	93214		
MATERIAL	COAL		
LOADING UNIT	16 M3 SHOVEL		

UL DESIGNATION	J	C
AD TIME	4.000	4.000
UL TIME	9.052	9.965
RN AND DUMP TIME	0.500	0.500
TURN TIME	4.831	5.452
OT TIME	0.750	0.750
TAL CYCLE TIME	19.133	20.667
IPS PER 50.0 MIN. HR	2.613	2.419
NS PER TRIP	141.7	141.7
NS PER 50.0 MIN. HR	370.3	342.8
URLY OPERATING COST		
OR HAUL UNIT ONLY)		
FUEL AT R 0.470 PER LITER	R 41.33	R 43.65
PREVENTIVE MAINTENANCE	2.62	2.62
REPAIRS	26.28	27.75
TIRES	17.48	17.75
OPERATOR	4.00	4.00
TOTAL OPERATING COST	R 91.71	R 95.77
RATING COST/TON - HAUL UNIT -	R 0.2477	R 0.2794
ITICAL TIRE TKPH	255	259
T. TIRE WEAROUT LIFE (HOURS)	2541	2503

TE: ALL TIMES IN MINUTES AND HUNDREDTHS. ALL CALCULATIONS ARE BASED ON A 50.00 MINUTE HOUR.
ALL TKPH CALCULATIONS ARE BASED ON 7.00 WORKING HOURS PER 8.0 HOUR SHIFT.
PRODUCTION SHOWN IS IN METRIC TONS. LOADING UNIT ALWAYS WAITS.

CONDITIONS AND DATA

Material: Coal, well blasted
Bank density - 1.6 t/m^3
Loose density - 1.23 t/m^3

Loading:

Power shovel
16 m^3 bucket (21 $yd.^3$)
0.9 bucket fill factor
Cycle time - 30 seconds/pass

Passes	m^3	t	Load Time
1	14.4	17.712	0.5
7	100.8	123.98	3.5
8	115.2	141.70	4.0

Cost Data:

All costs in S.A. Rand
Exchange rate: R 1.10 = $US 1.00
Operator wage - R 4.00/hr.
Mechanic wage - R 9.00/hr.
P.M. personnel wage - R 8.00/hr.
Fuel - R 0.47/L
Engine and hydraulic oil - R 1.05/L
Grease - R 1.35/kg
Tires - 27 x 49, 42 PR (E-4) - R 5874 each
27 x 49, 36 PR (E-3) - R 4443 each (est.)

Production Data:

7 operating hours per 8 hour shift
3 shifts per day
6 days per week
52 weeks per year
12 holidays per year
50 minute efficiency hour
Altitude - 1500 metres
Ambient temperature - 20° C

J.C.I. - PHOENIX PROJECT HAUL PROFILES

HAUL DESIGNATION	GRADE (%)	R.R (%)	DISTANCE (METER)	SPEED LIMIT HAUL	SPEED LIMIT RETURN
J	0.00	2.00	420.0	50.0	50.0
	0.00	2.00	90.0	20.0	20.0
	6.00	2.00	720.0	50.0	35.0
	0.00	2.00	80.0	20.0	20.0
	1.00	2.00	1500.0	30.0	50.0
	0.00	2.00	160.0	50.0	50.0
	6.00	2.00	85.0	50.0	35.0
	0.00	1.00	40.0	20.0	20.0

TOTAL = 3105.0 METER (3.11 KILOMETER)

HAUL DESIGNATION	GRADE (%)	R.R. (%)	DISTANCE (METER)	SPEED LIMIT HAUL	SPEED LIMIT RETURN
C	0.00	2.00	420.0	50.0	50.0
	0.00	2.00	30.0	20.0	20.0
	6.00	2.00	920.0	50.0	35.0
	6.00	2.00	30.0	20.0	20.0
	0.00	2.00	1550.0	50.0	50.0
	0.00	2.00	50.0	26.0	26.0
	0.00	2.00	130.0	50.0	50.0
	6.00	2.00	35.0	50.0	35.0
	0.00	2.00	40.0	20.0	20.0

TOTAL = 3405.0 METER (3.41 KILOMETER)

SIMULATION OF EARTHMOVING EQUIPMENT OPERATION
FOR
J C. I -PHOENIX
HAUL NUMBER J

MODEL 85D/150CT HAULPAK B. D

ENGINE	CUM KTA-2300-C	POWER	783. 3 KW (1050 HP)
TRANSMISSION	ALLISON DP-8961	FINAL REDUCTION	23. 29
TIRES	27. 00-49, 36PR, E3	ROLLING RADIUS	1. 314 METER
LOADED WEIGHT	234914 KG	EMPTY WEIGHT	93214. KG
MATERIAL	COAL		
LOADING UNIT	16 M3 SHOVEL		

TRANS GEAR	VEL KPH	TIME MIN.	DIST METERS	GRADE (PCT)	ROL RES (PCT)	SEGMENT LENGTH (M)	SPEED LIMIT
				0. 00	2. 00	420. 0	50. 00
5th	34. 40	0. 950	401. 2				
5th	20. 00	0. 991	420. 0				
				0. 00	2. 00	80. 0	20. 00
5th	20. 00	1. 231	500. 0				
				6. 00	2. 00	720. 0	50. 00
1st	10. 62	4. 944	1220. 0				
				0. 00	2. 00	80. 0	20. 00
3rd	20. 00	5. 220	1300. 0				
				1. 00	2. 00	1500. 0	50. 00
4th	30. 45	8. 320	2800. 0				
				0. 00	2. 00	180. 0	50. 00
5th	35. 00	8. 706	2980. 0				
				6. 00	2. 00	85. 0	50. 00
2nd	18. 85	8. 900	3065. 0				
				0. 00	1. 00	40. 0	20. 00
3rd	19. 68	8. 990	3094. 7				
3rd	0. 00	9. 052	3105. 1				

HAUL TIME = 9. 052 MINUTES

TRANS GEAR	VEL KPH	TIME MIN.	DIST METERS	GRADE (PCT)	ROL RES (PCT)	SEGMENT LENGTH (M)	SPEED LIMIT
				0. 00	1. 00	40. 0	20. 00
3rd	20. 00	0. 160	40. 0				
				-6. 00	2. 00	85. 0	35. 00
5th	35. 00	0. 321	125. 0				
				0. 00	2. 00	180. 0	50. 00
6th	49. 34	0. 573	305. 0				
				-1. 00	2. 00	1500. 0	50. 00
6th	49. 69	2. 305	1748. 5				
6th	20. 00	2. 402	1905. 1				
				0. 00	2. 00	80. 0	20. 00
6th	20. 00	2. 642	1985. 0				
				-6. 00	2. 00	720. 0	35. 00
5th	35. 00	3. 837	2572. 7				
5th	20. 00	3. 908	2605. 1				
				0. 00	2. 00	80. 0	20. 00
5th	20. 00	4. 148	2685. 0				
				0. 00	2. 00	420. 0	50. 00

```
5th    49 32   4 577  3042 2
6th     0.00   4 831  3105.1
       RETURN TIME =   4 831 MINUTES

          LOAD TIME                    4 000 MINUTES
          HAUL TIME                    9.052 MINUTES
          TURN AND DUMP TIME           0 500 MINUTES
          RETURN TIME                  4.831 MINUTES
          SPOT TIME                    0.750 MINUTES
          TOTAL CYCLE TIME            19.133 MINUTES
          TRIPS PER 50 0 MIN. HR       2 613 TRIPS
          AVG. SPEED(50 0 MIN HR)     16 228 KPH
          PAYLOAD PER TRIP             141 7 METRIC TONS
          PROD  PER 50 0 MIN  HR.      370.3 METRIC TONS
          FUEL PER 50.0 MIN. HR.       87.94 LITERS
```

TIRE CALCULATIONS

LOADED FRONT TIRE LOAD	90.75	PCT. OF 30 MPH RATING
LOADED DRIVE TIRE LOAD	100.63	PCT. OF 30 MPH RATING
LOADED REAR TIRE LOAD	113.34	PCT. OF 30 MPH RATING
AVERAGE FRONT TIRE LOAD	17.94	METRIC TONS
AVERAGE DRIVE TIRE LOAD	15.63	METRIC TONS
AVERAGE REAR TIRE LOAD	16.41	METRIC TONS
FRONT TIRE TKPH	254.8	
DRIVE TIRE TKPH	222.0	
REAR TIRE TKPH	233.0	
ESTIMATED TIRE LIFE	2541.	HOURS

COST CALCULATIONS

HOURLY OPERATING COSTS

FUEL AT R 0.470 PER LITER	R 41.33
PREVENTIVE MAINTENANCE, LUBE	2.62
REPAIR	26.28
TIRES	17.48
OPERATOR	4.00
TOTAL HOURLY OPERATING COSTS	R 91.71

OPERATING COST PER METRIC TON   R 0.2476

NOTES
ALL TIRE LOAD RATINGS ARE BASED ON 70 PSI INFLATION PRESSURE
ALL TIRE TKPH CALCULATIONS ARE BASED ON 7.00 WORKING HOURS PER 8.0 HOUR SHIFT

M146FL   0.9360   0.0486   44430.00

TIRE CALCULATIONS

LOADED FRONT TIRE LOAD	90.78	PCT. OF 30 MPH RATING
LOADED DRIVE TIRE LOAD	100.63	PCT. OF 30 MPH RATING
LOADED REAR TIRE LOAD	113.34	PCT. OF 30 MPH RATING
AVERAGE FRONT TIRE LOAD	17.94	METRIC TONS
AVERAGE DRIVE TIRE LOAD	15.63	METRIC TONS
AVERAGE REAR TIRE LOAD	16.41	METRIC TONS
FRONT TIRE TKPH	254.3	
DRIVE TIRE TKPH	222.0	
REAR TIRE TKPH	233.0	
ESTIMATED TIRE LIFE	2541	HOURS

COST CALCULATIONS

HOURLY OPERATING COSTS

FUEL AT R 0.470 PER LITER	R 41.33
PREVENTIVE MAINTENANCE, LUBE	2.62
REPAIR	26.28
TIRES	17.48
OPERATOR	4.00
TOTAL HOURLY OPERATING COSTS	R 91.71

OPERATING COST PER METRIC TON R 0.2476

NOTES
ALL TIRE LOAD RATINGS ARE BASED ON 70 PSI INFLATION PRESSURE
ALL TIRE TKPH CALCULATIONS ARE BASED ON 7.00 WORKING HOURS PER 8.0 HOUR SHIFT

M146FL 0.9360 0.0498 44430.00

SIMULATION OF EARTHMOVING EQUIPMENT OPERATION
FOR
J. C. I. -PHOENIX
HAUL NUMBER C

MODEL 85D/150CT HAULPAK B.D

ENGINE	CUM KTA-2300-C	POWER	783.3 KW (1050 HP)
TRANSMISSION	ALLISON DP-8961	FINAL REDUCTION	23.29
TIRES	27.00-49,36PR,E3	ROLLING RADIUS	1.314 METER
LOADED WEIGHT	234914 KG	EMPTY WEIGHT	93214 KG
MATERIAL	COAL		
LOADING UNIT	16 M3 SHOVEL		

TRANS GEAR	VEL KPH	TIME MIN.	DIST METERS	GRADE (PCT)	ROL RES (PCT)	SEGMENT LENGTH (M)	SPEED LIMIT
				0.00	2.00	420.0	50.00
5th	34.40	0.950	401.2				
5th	20.00	0.991	420.0				
				0.00	2.00	80.0	20.00
5th	20.00	1.231	500.0				
				6.00	2.00	920.0	50.00
1st	10.62	6.074	1420.0				
				6.00	2.00	80.0	20.00
1st	10.62	6.526	1500.0				
				0.00	2.00	1550.0	50.00
5th	43.12	9.036	3021.6				
5th	26.00	9.135	3050.1				
				0.00	2.00	50.0	26.00
5th	26.00	9.251	3100.0				
				0.00	2.00	180.0	50.00
5th	32.05	9.619	3280.0				
				6.00	2.00	85.0	50.00
2nd	18.81	9.814	3365.0				
				0.00	2.00	40.0	20.00
3rd	19.37	9.907	3395.6				
3rd	0.00	9.965	3405.1				

HAUL TIME = 9.965 MINUTES

TRANS GEAR	VEL KPH	TIME MIN.	DIST METERS	GRADE (PCT)	ROL RES (PCT)	SEGMENT LENGTH (M)	SPEED LIMIT
				0.00	2.00	40.0	20.00
3rd	20.00	0.181	40.0				
				-6.00	2.00	85.0	35.00
5th	35.00	0.322	125.0				
				0.00	2.00	180.0	50.00
5th	46.05	0.532	267.6				
5th	26.00	0.594	305.0				
				0.00	2.00	50.0	26.00
5th	26.00	0.710	355.0				
				0.00	2.00	1550.0	50.00
5th	49.35	2.590	1852.5				
5th	20.00	2.[illegible]50	1905.1				
				-6.00	2.00	80.0	20.00
5th	20.00	2.920	1985.0				

				-6.00	2.00	920.0	35.00
5th	35.00	4.458	2872.7				
5th	20.00	4.529	2903.1				
				0.00	2.00	80.0	-20.00
5th	20.00	4.769	2985.0				
				0.00	2.00	420.0	50.00
6th	49.38	5.300	3342.2				
6th	0.00	5.452	3405.1				

RETURN TIME = 5.452 MINUTES

LOAD TIME	4.000 MINUTES
HAUL TIME	9.965 MINUTES
TURN AND DUMP TIME	0.500 MINUTES
RETURN TIME	5.452 MINUTES
SPOT TIME	0.750 MINUTES
TOTAL CYCLE TIME	20.667 MINUTES
TRIPS PER 50.0 MIN. HR.	2.419 TRIPS
AVG. SPEED(50.0 MIN HR)	16.475 KPH
PAYLOAD PER TRIP	141.7 METRIC TONS
PROD. PER 50.0 MIN. HR.	342.8 METRIC TONS
FUEL PER 50.0 MIN. HR	92.85 LITERS

HAUL NUMBER 0 175.

TIRE CALCULATIONS

LOADED FRONT TIRE LOAD	90.78 PCT. OF 30 MPH RATING
LOADED DRIVE TIRE LOAD	100.63 PCT. OF 30 MPH RATING
LOADED REAR TIRE LOAD	113.34 PCT. OF 30 MPH RATING
AVERAGE FRONT TIRE LOAD	17.94 METRIC TONS
AVERAGE DRIVE TIRE LOAD	15.63 METRIC TONS
AVERAGE REAR TIRE LOAD	16.41 METRIC TONS
FRONT TIRE TKPH	258.7
DRIVE TIRE TKPH	225.4
REAR TIRE TKPH	236.6
ESTIMATED TIRE LIFE	2503. HOURS

COST CALCULATIONS

HOURLY OPERATING COSTS

FUEL AT R 0.470 PER LITER	R 43.65
PREVENTIVE MAINTENANCE, LUBE	2.62
REPAIR	27.75
TIRES	17.75
OPERATOR	4.00
TOTAL HOURLY OPERATING COSTS	R 95.77

OPERATING COST PER METRIC TON R 0.2793

NOTES
ALL TIRE LOAD RATINGS ARE BASED ON 70 PSI INFLATION PRESSURE
ALL TIRE TKPH CALCULATIONS ARE BASED ON 7.00 WORKING HOURS
PER 8.0 HOUR SHIFT

M146FL 0.9360 0.0488 44430.00

HAUL NUMBER 0 175.

TIRE CALCULATIONS

LOADED FRONT TIRE LOAD	90.78 PCT. OF 30 MPH RATING
LOADED DRIVE TIRE LOAD	100.63 PCT. OF 30 MPH RATING
LOADED REAR TIRE LOAD	113.34 PCT. OF 30 MPH RATING
AVERAGE FRONT TIRE LOAD	17.94 METRIC TONS
AVERAGE DRIVE TIRE LOAD	15.63 METRIC TONS
AVERAGE REAR TIRE LOAD	16.41 METRIC TONS
FRONT TIRE TKPH	258.7
DRIVE TIRE TKPH	225.4
REAR TIRE TKPH	236.6
ESTIMATED TIRE LIFE	2503. HOURS

COST CALCULATIONS

HOURLY OPERATING COSTS

FUEL AT R 0.470 PER LITER	R 43.65
PREVENTIVE MAINTENANCE, LUBE	2.62
REPAIR	27.75
TIRES	17.75
OPERATOR	4.00
TOTAL HOURLY OPERATING COSTS	R 95.77

OPERATING COST PER METRIC TON R 0.2793

NOTES
ALL TIRE LOAD RATINGS ARE BASED ON 70 PSI INFLATION PRESSURE
ALL TIRE TKPH CALCULATIONS ARE BASED ON 7.00 WORKING HOURS PER 8.0 HOUR SHIFT

M146FL 0.9360 0.0488 44430.00

APPENDIX 5

OVERBURDEN STRIPPING - FINANCIAL EVALUATION OF ALTERNATIVE METHODS

OVERBURDEN STRIPPING

To illustrate the importance of evaluating alternative mining methods and equipment, a typical opencast operation will be analized, consisting of a single coal seam of 4 metres, overlaid by 30 metres of sandstone overburden. The basic assumptions and parameters used in the evaluation are as follows:

(a) All overburden material was assumed to have a swell of 130% for construction of dragline range diagrams and to determine bucket loads. Where reference is made to overburden depths, it is assumed that topsoil has been removed by other methods. No allowance was made for dragline spoil stability as it relates to spoil height.

(b) This calculation assumes a pit of sufficient length to eliminate any operating constraints. The pit width was assumed to be 40 metres in all examples. Highwall slopes of 70° were used along with spoil slopes of 38°.

(c) Working Schedule: The equipment used in these comparisons was scheduled based on 7416 hours per year, which is equivalent to 309 working days at 24 hours per day. For dragline operations a total of 8400 scheduled hours per year is used, as exemptions were granted by the office of the Government Mining Engineer to operate these units on a 7 day per week basis. Operating efficiencies of 85% for dragline excavators and 70% for shovels, hydraulic excavators and front end loaders were used.

(d) For draglines an availability of 85% was used and for all other equipment a factor of 80% was assumed.

(e) The economic comparison is based on the present value of operating costs and original investment, inflation of 10% and escalation at 10% per annum, and a 20 year investment period.

In all instances, it was assumed that the equipment could be erected in two years. Forty percent of the investment was arbitrarily assigned to the first year and 60 percent to the second. The economic rating illustration compares the most desirable alternative, expressed as 100, with the other alternatives.

Operating costs include labour and electric power. No differentiation was made for the various classes of labour because it was felt the accuracy would not materially effect the results.

An allowance of R5 000 per hectare covers those cases where dragline spoil piles are created and must be levelled. This is not intended to be the total reclamation cost, but only the cost of levelling the peaks to a gentle roll. Where spoil dumping is carried out by trucks, a cost of R2 500 per hectare was used.

This analysis covers the following mining methods:

- Draglines
- Shovels and Trucks
- Hydraulic Excavators and Trucks

for the following annual production rates: 5; 10 and 15 million cubic metres of overburden per year.

1. <u>Dragline Selection</u>

   The first step in selecting a suitable dragline size is to determine the required dragline reach.

   By using equation 5.1 (page 62) the reach factor is given as:

$$\frac{D(1+S)}{\tan\theta} + \frac{W}{4} + \frac{D}{\tan\beta} - \frac{T}{\tan\theta}$$

Where D = Overburden depth

S = Spoil Pile Swell factor

θ = Spoil Pile Slope with the horizontal (degrees)

W = Pit width

β = Highwall slope with the horizontal (degrees)

T = Coal thickness

By substituting the given parameters into this equation, then:

$$\text{Reach factor} = \frac{30(1,30)}{\text{Tan } 38} + \frac{40}{4} + \frac{30}{\text{Tan } 70} - \frac{4}{\text{Tan } 38}$$

$$= \underline{65,7 \text{ metres}}$$

The next important factor to calculate is the stacking height.

From equation 5.2 the stacking height is given as:

$$= \frac{Rf - D \text{ Cot}\beta}{\text{Cot } \theta} - T - D$$

$$= \frac{65,7 - 40 \text{ Cot } 70}{\text{Cot } 38} - 4 - 30$$

$$= \underline{6 \text{ metres}}$$

To determine the bucket capacity, equation 5.3 is used. Therefore, to handle 5,0 million cubic metres of overburden per annum, the bucket capacity must be as follows:

$$\text{Bucket capacity} = \frac{Y \times C \times (1 + S)}{O \times A \times U \times B} \qquad (5.3)$$

$$= \frac{5 \times 10^6 \times 60 \times 1{,}30}{(8400 \times 60 \times 60) \times 0{,}85 \times 0{,}85 \times 0{,}90}$$

$$= 19{,}8m^3$$

To allow for 10% rehandling of spoil due to ramps and other unplanned activities, the required bucket capacity must therefore be increased to 19,8 x 1,10 = 21,78 m^3.

The maximum required suspended load equals 21,78 x 2272 kg.

= <u>49 484 kg</u>

These parameters are within the design limits of the Bucyrus Erie 1260-W model 15 dragline as detailed in Table 5.3

By applying the same calculations as above, the following size dragline will be required to strip 10 million cubic metres per annum.

Reach Factor	=	65,7 metres
Bucket Capacity	=	43,6 m^3
Maximum Suspended Load	=	99059 kg

Model : Bucyrus Erie 1370W.

Similarly the following parameters will apply for the required dragline to strip 15 million cubic metres per annum.

Reach Factor	=	65.7m
Bucket Capacity	=	65m^3
Maximum Suspended Load	=	1477[illegible] kg

Model : Bucyrus Erie 1570W

From a detailed study by the author (Fourie, 1983) the capital and hourly operating costs for these draglines are as follows:

TABLE A5.1
DRAGLINE CAPITAL AND OPERATING COSTS

DRAGLINE	CAPITAL COST	HOURLY OPERATING COST
BE 1260 W	R10 500 000	R120
BE 1370 W	R22 031 000	R200
BE 1570 W	R26 361 000	R280

Annual rehabilitation costs at R5 000 per hectare for the various stripping rates are as follows:

TABLE A5.2
ESTIMATED ANNUAL REHABILITATION COST

STRIPPING RATE (BCM/ANNUM)	REHABILITATION COST PER ANNUM (R)
5×10^6	R 83 000
10×10^6	R167 000
15×10^6	R250 000

The cash flows for a project period of 20 years at different production rates are detailed in Table A5.3

TABLE A5.3

CASH FLOWS - DRAGLINES

YEAR	$5,0 \times 10^6$ BCM/annum	$10,0 \times 10^6$ BCM/annum	$15,0 \times 10^6$ BCM/annum
1	4 200 000	8 812 000	10 544 400
2	6 930 000	14 540 900	17 398 260
3	1 452 121	2 234 870	3 148 420
4	1 597 333	2 458 357	3 463 262
5	1 757 066	2 704 193	3 809 588
6	1 932 773	2 974 612	4 190 547
7	2 126 050	3 272 073	4 609 602
8	2 338 655	3 599 280	5 070 561
9	2 572 521	3 959 209	5 577 618
10	2 829 773	4 355 129	6 135 379
11	3 112 750	4 790 642	6 748 918
12	3 424 025	5 269 707	7 423 810
13	3 766 428	5 796 677	8 166 191
14	4 143 071	6 376 345	8 982 810
15	4 557 378	7 013 979	9 881 091
16	5 013 116	7 715 377	10 869 200
17	5 514 427	8 486 915	11 956 120
18	6 065 870	9 335 607	13 151 731
19	6 672 457	10 269 167	14 466 905
20	7 339 702	11 296 084	15 913 595
NPV @ 10%	29 181 000	50 248 000	66 539 000

2. <u>Shovel and Truck Selection</u>

Shovel cycle time in medium digging conditions = 29 seconds.

∴ Required hourly production rate to produce 5 million BCM per annum

$$= \frac{5\ 000\ 000}{7416}$$

$$= 674 \text{ BCM/hour}$$

∴ Dipper capacity required

$$= \frac{674 \times 1{,}30 \times 29}{60 \times 60 \times 0{,}80 \times 0{,}70 \times 0{,}9}$$

$$= 14{,}0 \text{ m}^3$$

Maximum suspended load

$$= 14{,}0 \times 2272 \text{ kg}$$

$$= 31\ 808 \text{ kg}$$

This is equivalent to a Bucyrus Erie model 195B rope shovel (See Table 5.5) with a capital cost of R3,2 million.

Total hourly truck production rate of 150 ton capacity rear dump truck = 151 BCM/hour (see Appendix 4 for truck simulation).

∴ Number of trucks required = 674 ÷ 151 = 5

Truck hourly operating cost = R126,59

∴ Replacement period for truck estimated at 5 years or 37 000 hours. Capital cost per truck = R900 000

Shovel operating cost per hour = R50,00

To produce 10 million BCM/annum and 15 million BCM/annum the same mining fleet as for the 5 million BCM/annum operation will be used and increased proportionally.

The annual cash flow for a shovel-truck operation is detailed in Table A5.4.

3. Hydraulic Excavators and Trucks

As for the electric rope shovel, the hourly loading rate for the hydraulic excavator must be 674 BCM/hour to produce 5 million BCM per hour, and the dipper capacity must be 14,0m^3. This is equivalent to the O & K RH120 hydraulic excavator, with a capital cost of R1 700 000 and an hourly operating cost of R236,00. The estimated life of this unit is 8 years. The same truck fleets as for the shovel/truck operation will be used.

The cash flows for the different production rates are detailed in Table A5.5.

TABLE A5.4
CASH FLOWS : SHOVEL AND TRUCK

YEAR	5,0 x 10^6 BCM/a	10,0 x 10^6 BCM/a	15,0 x 10^6 BCM/a
1	3 080 000	6 160 000	9 240 000
2	5 082 000	10 164 000	15 246 000
3	3 775 067	7 550 134	11 325 201
4	4 152 000	8 304 000	12 456 000
5	4 567 832	9 135 664	13 703 496
6	9 083 075	18 166 150	27 249 225
7	5 527 076	11 054 152	16 581 228
8	6 079 784	12 159 568	18 239 352
9	6 687 762	13 375 524	20 063 286
10	7 356 538	14 713 076	22 069 614
11	14 628 342	29 256 684	43 885 026
12	8 901 412	17 802 824	26 704 236
13	9 791 553	19 583 106	29 374 659
14	10 770 708	21 541 416	32 312 124
15	11 847 778	23 695 556	35 543 334
16	23 559 026	47 118 052	70 677 078
17	14 335 812	28 671 624	43 007 436
18	15 769 395	31 538 790	47 308 185
19	17 346 332	34 692 664	52 038 996
20	19 080 966	38 161 932	57 242 898
NPV @ 10%	64 930 000	129 860 000	194 790 000

TABLE A5.5
CASH FLOWS: HYDRAULIC EXCAVATORS AND TRUCKS

YEAR	$5,0 \times 10^6$ BCM/a	$10,0 \times 10^6$ BCM/a	$15,0 \times 10^6$ BCM/a
1	2 480 000	4 960 000	7 440 000
2	4 092 000	8 184 000	12 276 000
3	3 864 036	7 728 073	11 592 109
4	4 250 440	8 500 881	12 751 322
5	4 675 484	9 350 968	14 026 453
6	9 201 492	18 402 985	27 604 478
7	5 657 336	11 314 672	16 972 008
8	6 223 069	12 446 139	18 669 208
9	8 886 073	17 772 145	26 658 218
10	7 529 914	15 059 828	22 589 742
11	14 695 466	29 390 931	44 986 397
12	9 111 196	18 222 392	27 333 588
13	10 022 316	20 044 631	30 066 947
14	11 024 547	22 049 094	33 073 641
15	12 127 002	24 254 003	36 381 005
16	23 865 873	47 731 746	71 597 618
17	14 673 672	29 347 344	44 021 016
18	16 141 039	32 282 078	48 423 117
19	17 755 143	35 510 287	53 265 430
20	19 530 657	39 061 315	58 591 972
NPV @ 10%	65 630 000	131 260 000	196 890 000

APPENDIX 6

ECONOMICS OF IN-PIT
PARTING SEPARATION

The coal seams of the Witbank Coalfield are characterised by the presence of various partings, ranging in thickness from a few centimetres to more than three metres.

To obtain maximum results and to minimize operating costs it is therefore essential to determine the optimum thickness of parting material which can be tolerated in the run-of-mine feed to the coal washing plant before affecting either the mining cost or coal preparation plant operating costs.

The following typical cross-section of the 4 seam horizon is used for this calculation:

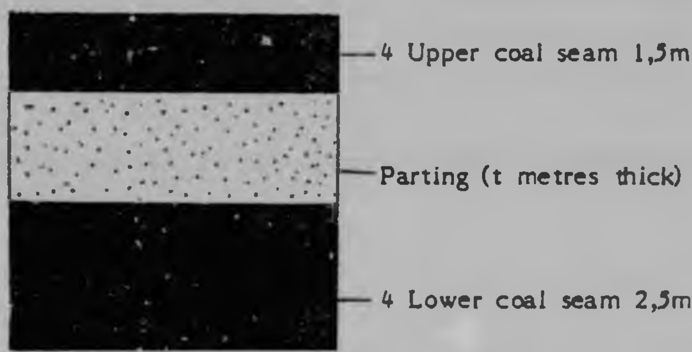

FIGURE A6.1
No 4 SEAM

Let the thickness of the parting separating the 4-upper and 4-lower seams be 't' metres thick.

The first option is to mine the three layers as individual mining horizons and the second option is to include the parting in the run-of-mine feed.

Option A
From Figure A6.1 the total in-situ coal thickness equals 1,5 + 2,5 = 4,0 metres.

Assume a geological loss of 5% which will be included as dilution.

∴ ROM coal = 1,425m + 2,375m

= <u>3,80m</u>

Assume a mining loss of 10cm at each coal contact surface due to mining activities.

The total mineable coal seam reduces to (1,425 - 0,20) + (2,375 - 0,20)m = 1,225 + 2,175m

= <u>3,40m</u>

The total ROM product to the plant is 1,225m ROM coal + 0,075m dilution + 2,175m ROM coal + 0,125m dilution

= 3,60m total of which 3,40 metre is coal.

At a relative density of 1,60 this is equal to 5,44 ROM tons of coal.

The following typical mining costs are used to estimate the total operating costs.

Drill and blast 4U seam	=	1,5 x 0,35 = 0,53
Drill and blast 4L seam (2,5 x 0,29)	=	0,73
Drill and blast parting t x 0,54	=	0,54t
Loading to plant	=	3,60 x 0,13 x 1,65 = 0,77
Hauling to plant	=	3,60 x 0,34 x 1,65 = 2,02
Rehandle of discard at plant	=	0,20 x 0,12 x 2,5 = 0,06

Dozing of in-situ parting over a distance of 60m to expose 4L Seam

= (0,20 + t) x 0,48

∴ Total cost = (4,20 + 1,02t)

∴ Cost per ROM ton coal = (4,20 + 1,02t) : 5,44

= <u>0,77 + 0,19t</u> (1)

Option B

This alternative investigates the total cost per ROM ton assuming that all the included parting will be hauled to the coal preparation plant.

From Figure A6.1

Geological in-situ coal	= 1,5m + 2,5m
	= 4,0m
Geological losses	= 5%
∴ ROM coal	= 1,425 + 2,375m
	= 3,80m

The total number of mining contacts reduce to two, and by assuming a mining loss of 10cm at each contact the total ROM

= (1,425-0,10)+(2,375-0,10)

= 3,60m

∴ ROM product to the plant	= 1,325m coal + 0,075m dilution
	+ 2,275 coal + 0,125m dilution
	+ tm parting
	= 3,60m coal +(0,20+t)m parting

At a RD of 1,60 the total coal in the plant feed is equal to 5,76 ton. The total operating cost for this operation is as follows:

Drilling and blasting	= (4,0 x t) x 0,21
	= 0,84 + 0,21t
Loading to the plant	= 0,13[3,60 x 1,6 + (0,20+t) 2,5]
	= 0,81 + 0,325t
Hauling to the plant	= 0,34[(3,60 x 1,6)+(0,20 +t)2,5]
	= 2,13 + 0,85t
Rehandle at plant	= 0,12 (0,20 + t) 2,5
	= 0,06 + 0,30 t

$\therefore$ Total operating cost = 3,85 + 1,69t and the operating cost per ton is equivalent to (0,67 + 0,29t) Rands (2)

For a break-even cost, equation 1 must equal equation 2

$\therefore$ 0,77 + 0,19t = 0,67 + 0,29t

t = 1,0 metres

Therefore, for a parting thickness of one metre or more, it is more economic to strip the parting in the pit as a seperate mining horizon rather than hauling it to the washing plant.

If the coal preparation plant costs or washing costs are included in the calculations, the calculations can be modified as follows:

Assume a rotary breaker operating cost to be R0,06/ton feed, a plant washing cost of R1,00/ton of feed and 50% of the parting material are discarded by the rotary breaker.

Option A

Separate mining of coal and partings

Total cost	= (4,20 + 1,02t) Rands
Add	
Rotary breaker cost (partings)	= 0,5 x 0,20 x 2,5 x 1,00
Washing cost of partings	= 0,05 x 0,20 x 2,5 x 1,00
$\therefore$ Total	= 0,28

Revised operating cost for option A is (4,48 + 1,02t) Rands

= (0,82 + 0,19t) Rands per ROM ton (3)

Option B

Mining the coal and parting as a single unit.

Total cost (from equation 2)	= (3,85 + 1,69t)Rands
Add	
Rotary breaker cost	= (0,20 + t) x 2,5 x 0,06
Washing of parting	= 0,50 x (0,20 + t) x 2,5 x 1,00

Total	=	(0,28 + 1,40t)	
∴ Revised cost	=	(4,13 + 3,09t) Rands	
	=	(0,72 + 0,54t) Rand per ROM ton of coal	(4)

∴ For a break-even cost and solving for 't' then

(0,82 x 0,19t)	=	(0,72 + 0,54t)
t	=	0,29m