Feasibility of thin seam coal mining at Dorstfontein Coal Mine

by

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treatise submitted in fulfillment of a part of the requirements for the

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ABSTRACT

Dorstfontein Coal Mine is situated in the northern limb of the Highveld Coalfield. The mine is currently owned by Total Coal South Africa Ltd (Pty). Mining to date has taken place where the seam heights are in the excess of 1.5 m with an average height of 1.9m. Some areas have been identified where the seam heights ranges between 1.2 and 1.4m with an average height of 1.32m. The in situ tonnage of the thin seam areas is 7.06mil. tons.

The thin seam coal quality is very good and product yield at an ash content of 13.5% is 95.7% and at a cut density of 1.6 the yield is 89.2% (Air dry basis).

The largest thin seam coal producer in the world is the U.S.A. followed by the former U.S.S.R. Other countries that produced coal from thin seams are mainly from Europe.

In the Republic of South Africa most of the thin seam coal mining was concentrated in the KwaZulu-Natal province. Most of the larger mines are now defunct but some small mines are still operating.

The risks involved in thin seam coal mining differ from that of thicker seam mining. There are occupational diseases associated specifically with thin seam coal mining. The most pronounced geological risks are changes in seam heights, changes in coal quality, in-seam partings and unpredicted dolerite intrusions.

At Dorstfontein Mine a newly developed German Wirth Paurat thin seam continuous miner is been tested. Some Stamler BH10 thin seam battery haulers were introduced to the section to haul the coal from the face to the tip.

There are some advantages in mining the thin seam coal. The increase in yields, savings in belt replacement, less handling of stone and the extension of the life of mine are some of the major benefits.

For the financial evaluation it was assumed that 30% of the run of mine tons will come from the thin seam resource. All Capex and Opex costs were allocated pro rata at a 30% basis. The production rate was based on current experience and the assumption that this section will reach its completion at the same time as the mine closes. The run of mine tons (R.O.M.) are 3.53mil. tons which is 50% (70% extraction, 10% mining loss, 10% geological loss) of the in situ resource of 7.06mil. tons. For 10 years at an average daily production of 1400 tons per day, a total R.O.M. of 3,514mil. tons could be achieved, which relates to 99.55% extraction of the in situ R.O.M. tons.

Capital expenditure is minimal and many sunk costs are excluded from the model. The main Capex item is the Wirth Paurat. The N.P.V. for the project is R 27,206 mil. at a discount rate of 15% and the corresponding I.R.R. is 305.2%. The distorted I.R.R. is related to the small but realistic capital input.

Sensitivity analyses were performed for Operating Costs, Selling Prices (Export and Domestic), Yield and Production. The project is the most sensitive for selling prices, and operating costs.

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CHAPTER 1: INTRODUCTION

Dorstfontein Coal Mine is situated at the northern limb of the Highveld Coalfield (Snyman, 1998). The close proximity of the Nebo Granite Suite (S.A.C.S., 1980), which outcrops near the box-cut, to the No. 2 Seam makes it a very difficult mine to operate. The coal seam mimics the granite paleo-topography and causes the seam conditions to vary extremely rapidly. Some of the related problems are floor rolls and the sudden change in the coal seam thickness. The mine has been in operation for four years during which time the best parts of the ore body were exploited. The seam heights were in the excess of 1.5 meters. In the north-western part of the mine the excessive rolling floor prohibited production. In some areas of the mine the seam is split into a thin (0.01 – 0.15m) upper and a thicker lower (1.2- 1.75m) seam by an upwards coarsening sandstone parting. Currently some mining is taking place below this seam-splitting parting where the seam height ranges between 1.5 and 1.75m. In other parts of the deposit very thin seam conditions prevail below the parting with heights ranging between 1.2 and 1.4 meters. Hopefully these very thin seam areas will be mined in the near future. In many countries these heights are not be regarded as thin as the definition for thin seams is any thickness between 0.6 and 1.0m (Clarke et al., 1982). In this treatise a thin seam will be regarded as a seam between 1.2 and 1.4m thick.

These thin seam areas were previously regarded as not mineable and omitted from reserves. These areas contain very high-grade coal and have the potential of adding another 6 years to the life of the mine. The aim of the study is to determine whether these areas can be mined economically and profitable.

1.1 Definitions and terms

Box-cut:	A decline ramp intersecting the strata at an angle of ± 7° and ending in the mineable coal seam.
Thin seam:	A seam with a thickness between 1.2 and 1.4m.

Parting: A competent layer of sandstone or siltstone in the coal seam and sometimes separating different seams.

Pre-Karoo: All rocks older than Karoo age, that is older than ±320 Ma.

Proximate analysis: The most basic analysis for a coal sample and done on an air dried basis: Moisture content, Ash content, Volatile matter, Fixed Carbon content (Karr, 1978 and Meyers, 1981).

Raw coal: Not beneficiated, as mined.

R.O.M.: Run of mine, the material coming out of the mine. ± 50 -60% of in situ reserve.

Seam: The coal horizon.

Strong roof: The horizon above the coal that forms a roof with strength in the access of 60 MPa. It normally consists of a fine to medium grained sandstone.

Wash fraction: The relative density or densities (R.D.) at which coal is beneficiated. Listed in a washtable (Table 1). Can be any R.D. between 1.0 and 2.7.

Table 1. Float fractions and qualities used in a washtable

Float	Yield	CV	Ash	H_2O	Vol	FC	S	Phos
1.35								
1.37								
1.40								
1.45								
1.50								
1.60								
1.70								
1.80								
2.20								

Washtable: The quantitative values of each coal quality analyzed for, at a specific R.D., listed in table form (Table 1).

Weak roof: any horizons that will break up or part during normal mining activities.

Yield: The resultant tonnage when 1 ton of coal is washed at a specific R.D., expressed as a percentage. For this study all yields quoted are theoretical yields i.e. no plant efficiency or other losses were factored into the yield.

1.2 The problem and its settings

The areas of the thin seam coal resources are normally associated with the seam-split parting. This parting divides the coal into a very thin upper coal and a lower thicker coal. It is this lower coal that is of economic importance and needs to be extracted. The following problems exist:

a.) The parting left to form the roof creates dangerous roof conditions and reduces the mining heights to between 1,2 and 1,4 m. If the parting is extracted, the heights increase but the yields of the thin seam coal drop to uneconomical proportions. Stowing the parting underground is an option but stone handling is costly and can cause injury.

b.) It is clear that the continuous miner (CM) mining method is the most efficient to extract thin seam coal. Drill and blast methods need reasonable heights and space and currently the equipment on the mine is too high for the thin seam areas. Drill and blasting below the parting causes it to break and separate which defeats the whole object of excluding the sandstone from the R.O.M. The CM operation would probably be more effective but the CM can not cut hard stone.

c.) Production rate. There is a production cutoff where the cost of the tonnage mined exceeds the revenue received for the product. What is the minimum tonnage that can be produced economically from thin seam areas?

d.) Yield cutoff. Hand-in-hand with production rates goes the yield of the extracted material. If the yield is to low, the production must be increased to make up for the lost product coal. The parting must remain up to increase the yield. What is the cutoff yield and how is it affected by inclusion of the parting?

e.) Health and Safety. What are the safety implications if the parting is kept up? How will personnel and machinery be able to work safely in the thin seam area? What are the new health and safety risks when mining thin seam coal?

f.) Costs. How much will it cost to undertake thin seam mining? New thin seam equipment will be introduced and tested below the parting. What is the break-even point in production rate and costs?

1.3 Hypothesis

Current thick seam mining operations in similar conditions as thin seam areas indicate that the theoretical yield falls from 85% to 65% when the parting is included. This means that for every hundred tons mined, only 65 tons can be sold but the company still has to pay for hundred tons mined. It is more economical to mine as much "clean" coal as possible. The feeling is that in the thin seam areas the parting will have to stay up and form the roof to increase the yields and to make this an economical area. This mining method creates numerous problems regarding health and safety and will lead to a decline in the production rate. The risks have to be quantified and weighed up against the necessity to mine these thin seam areas. In the end the decision to go ahead with thin seam mining will be based on economical as well as health and safety issues. It is postulated that mining the thin seam coal will be expensive but profitable. The working conditions will change and workers will have to become comfortable with their new working environment.

1.4 Delimitations

1.4.1 Only thin seam areas have been assessed and evaluated.

1.4.2 The mine will be in operation for at least the next ten years.

1.4.3 This is not a complete feasibility study and only focuses on one aspect of the geology namely the thin seam resource.

1.4.4 The current borehole spacing is 1 hole / 300m and all the geological conditions have been modeled based on this spacing.

1.4.5 Very little information exists about other thin seam operations.

1.5 Assumptions

1.5.1 It is assumed that the entire infrastructure exists on the mine surface and underground. This will just be an additional section at the mine.

1.5.2 This study assumes that the geology has been well defined and this is no attempt to revise the geological section of the feasibility report of Dorstfontein Mine. The geological insert merely acts as background for the reader with additional information about the thin seam added, as gathered through the lifetime of the mine.

1.5.3 The study intends to change the long-term planning and scheduling of the mine as it adds additional information and creates the possibility of extending the life of the mine.

1.5.4 This study assumes that the current policy of T.C.S.A., to use contractors for mining and to outsource all activities, will not change in the future.

1.6 Research methodology

1.6.1 Current history of Dorstfontein mine. The past and current mining problems and geological conditions will be reviewed.

1.6.2 Use of borehole information. Borehole core was used to study and analyze the parting strengths and properties. The information gathered from these reports and the analyses from coal sampling were used in this study.

1.6.3 Geological model simulations. Use was made of the geological data supplied by T.C.S.A. head office. The Minescape/Stratmodel software was used to model coal qualities and seam heights.

1.6.4 The same software was used to determine the in-situ thin seam coal resource.

1.6.5 The data gathered and analyzed was used to come to a conclusion regarding the feasibility of extracting coal from thin seam areas.

CHAPTER 2: REVIEW OF RELATED MATERIAL.

Very little information exists about thin seam coal mining. Contrary to this there exist great volumes regarding coal mining and coal as a rock. These publications are not relevant to the problem of thin seam mining, its methodology, products and cost. The only relevant publication found is that of Clarke et al, (1982): Thin Seam Coal Mining Technology. Another very interesting but old book by Smyth: Coal & Coal Mining was published in 1886. This book makes very interesting reading about the mining methods, problems and history of the old British collieries.

In the book of W. W. Smyth he refers to the startling observation made in 1860 that the British coal output had doubled in 20 years, from 65 million tons to 134.6 million tons per annum. The big concern of the day was the new technology of using explosives to liberate coal at the face, which led to many fatalities and injuries due to "blow-out" shots. One of the biggest concerns of the time was underground explosions caused by gases and poor ventilation. It seems that the greatest danger was the extinction of the miner's cap lamp flame during an explosion leaving the underground workers without light. This resulted in many miners being lost underground in the dark, as they could not find their way out. This seems to be one of the earliest health and safety problems due to bad lighting or no lights at all.

The relevant issues at the time (1885), which still hold for today's drill and blast mining and of which some can be applied to continuous miner operations, are the following: i.) adopting such methods that will produce the least dust, ii.) the removal of such dust and prevention of it being carried down the downcast ventilation system, iii.) watering where practical the places in which dust accumulates and the sprinkling of common salt or other deliquescent material, iv.) the avoidance of common concussions accompanied by much flame as caused by "blown-out" shots and the careful examination for gas and clearing of dust from the place where a shot is to be fired.

Smyth (1886) also describes the very primitive ways that were employed in the 1800s to liberate coal. The first procedure was to "hole" the coal by cutting a groove two to three feet deep in the lowest part of the coal with a pick. For this holing at the bottom of the seam the collier laid on his side and in an apparently constrained attitude swung the pick almost horizontally. Some coal seams had the advantage of being able to be holed in the middle, depending on the position of the in-seam partings. The sides were cut vertically, called shearing, to form a short block of coal that needed to be collapsed. The final breaking down or "collapsing" of the seam was done by applying taper wedges a few feet apart and driving them with heavy hammers. In some cases where the coal was more resistant to collapsing, use was made of gunpowder. Later developments made use of hand drills to drill holes into the coal seam and charged with gunpowder. This method led to many injuries as proper tamping of holes did not exist and gunpowder easily pre-ignites. It is also rendered useless when wet and waterproof packaging did not exist at the time.

Bord and pillar mining layouts were the most common but longwall-mining did exist, leaving nothing but goaf or gob behind. Support was installed by means of timber props to uphold the overlying strata and in many cases where the heaviest roof pressure was expected they used nogs and chocks instead of props. Coal was removed from the face by dragging sledges, loaded with coal, along the floor. In some of the more primitive mines the coal was loaded into baskets and carried by woman bearers. The Germans were the first people to introduce underground rails. The problems encountered with underground rails were their frequent sinuosity and unevenness, confined space and the tendency to disturb roof and floor. Special designed wagons were used to transport the coal up an incline shaft. The various trolleys and tubs were either pulled by Shetland ponies or pushed by boys. It is mentioned that where very thin seams were worked the cost of carting the coal becomes very onerous (Smyth, 1886). In thin seams the tubs or wagons must necessarily be low and the wheels small so that the total weight is low in order for the onsetter and banksmen to easily pull or push the trolley up the mostly incline shafts.

During the 1800's the fatality in British coal mines were between nine hundred (900) and one thousand two hundred (1,200) people per year. The most common cause of deaths and accidents were falls of roof, methane explosions due to poor ventilation, shaft accidents and holing into old workings where methane and other gases have accumulated as well as inrushes of water which were lying under pressure in the old areas. The most feared substance and cause of fatalities in the mines was so called firedamp better known today as methane.

A very interesting book and one used very extensively in this study is one on thin seam coal mining technology and by Clarke et al. (1982). This is the only book dealing exclusively with thin seam mining methods as most other publications and books deal with coal and mining methods in general. It can also be concluded that thin seam coal mining has become unfavourable due to its low production rate and high cost and that the focus is more on high output (economy of scales) from thicker coal seams. Clarke et al. (1982) highlights the occurrences of accidents in thin seams, various extraction methods and equipment, health and safety issues, mine design and layout, costs and thin seam resources, from mainly U.S.A. based mines. This book was published in 1982 and covers mainly the mining in the 1960's and 1970s when coal prices were high and costs exuberant. The mines sold low ash coal (12-16%) for $28.0 but mined that coal at $34.0-$40.0 per ton. They were and still are heavily subsidized and many tax incentives were introduced to keep these mines open so that small communities could survive.

Many lessons can be learned from the American thin seam collieries regarding their mining methods, health and safety issues and mining costs. Real issues and factual data was used from operating collieries within the U.S.A. and compared to other collieries in the former U.S.S.R., Colombia, Great Britain, and other European countries. Many of the issues raised in this publication can be directly implemented and applied to the Dorstfontein scenario. The risks involved are pertinent to our current mining as well as to the proposed thin seam mining areas. As very few mines are currently mining thin seam coal in the R.S.A., lessons must be learned from the past and be applied at Dorstfontein.

In Chapter 3 (Clarke et al., 1982) a comparison is made between the accident analysis of thick seam and thin seam mining. The various kinds of accidents mentioned are relevant to the current mining a Dorstfontein and will be used as risks for the thin seam mining. Chapter 12 deals with productivity and the factors affecting productivity. Although many of the statistics and data goes back to the 1960s and 1970s, it can be assumed that because of the mining conditions and productivity with modern-day machines will not be dissimilar from those eras. Many U.S.A. thin seam mines produced 20 000 tons per month per section from 24 inch (0.6m) high seams. In the conclusions it is quoted that there is a correlation between seam thickness and labour productivity. There are also countries where thin seam mines are very productive due to good geological conditions such as competent and strong roofs and flat seams.

Chapter 13 (Clarke et al., 1982) deals with costs and although costs in the 1960s and 1970s cannot be compared to today's cost, one can come to a conclusion about the exorbitant costs of thin seam mining. It is interesting however that the selling price of high quality coal in dollar terms in 1977 is the same as today but decreased in terms of inflation adjusted figures. The main reason for this is that the highest quality coal occurs in thin seams and is well sought after because of the low sulphur and ash content. This is the same quality coal produced at Dorstfontein Mine. Chapter 14 covers the health and safety environment and gives a very good insight into conditions that could be expected when entering the thin seam areas. Up to now at Dorstfontein seam heights (all above 1.5m) comparable to that mined in the U.S.A (between 0.6 and 0.75m) have not been encountered. In Chapter 15 the authors deal with the various mining systems and methods and give one insight into the various methods employed in thin seam coal winning. Chapter 19 discusses the output and productivity of various mining methods. At Dorstfontein the mining methods are fixed, in the sense that bord and pillar layout applies, continuous miner machines are being used and that the necessary equipment for thin seam extraction has already been bought or current equipment adapted. Chapter 20 deals with the costs involved in thin seam mining. It appears that labour cost forms the greatest component in the U.S.A. but in the R.S.A. the possibility exists that the capital costs will form the greatest component due to the volatile exchange rate. The financial sensitivities involved in thin seam mining and their effect on production and cost are discussed in Chapter 22. Extracts from this publication have been used to design the financial model, assess the risks and the sensitivities. It provides a general background on the various thin seam mining methods used in the U.S.A. and other parts of the world.

Very little information exists in the R.S.A. about previous mining of thin seams in KwaZulu-Natal. Spurr et al. (1986) published a few papers on the general geology of the Vryheid and Utrecht coalfields, its qualities and tonnages. Most of the mining problems, production rates and costs are kept in in-house reports and are not available to the public.

Jacobs (1989) identified the relationship between geological conditions and mining problems at Ermelo Mines, but the problems of the thin seam areas here differ distinctly from Dorstfontein as they encountered bad roof conditions, which do not occur at Dorstfontein as frequently as they did at Ermelo Mines.

This document would therefore appear to be one of the few documenting the potential mining of thin seam coal resources in South Africa. This is a radical opinion since thin seam coal mining has become unfavourable due to its high costs and low production rates. It is the opinion of the author however, that this view will change as thick coal seam resources are being depleted and the need for additional coal resources will necessitates the reinvestigation of thin seam deposits. The findings relevant to the Dorstfontein deposit may have far reaching consequences in other mining areas as it may result in substantial increases in available resourcess.

CHAPTER 3: GEOLOGY OF THE NO. 2 THIN SEAM.

3.1 Introduction

Extracts from a 1999 AngloVaal Minerals geological report by Stewardson and Saunderson have been used for this chapter. A few amendments have been made based on additional information that has become available from recent drilling programmes. Underground mapping and recording of mining problems have added to this information, which has been reconciled with the borehole data.

The term "reserve" used in this study complies with the SAMREC code (SAMREC, 2000) as this thin seam area has been included in the approved Environmental Management Programme Report (EMPR) and the mining permission area. The necessary extraction rates are known, the market exists and all the other elements of the definition have been met. The thin seam was not regarded as mineable due to practical reasons like the non-existence of modern high productive equipment.

3.1.1 General

Dorstfontein Coal Mine falls within the Highveld Coalfield and is situated 4 km east of the town of Kriel and 25 km northwest of Bethal. (Fig. 3.1) Adjacent collieries include the defunct Ingwe operated Transvaal Navigation Colliery (TNC), the current Xstrata Mines of Arthur Taylor Colliery (ATC) and Arthur Taylor Colliery Open Cast Mine (ATCOM) which are 15 km to the north, the Anglo Coal operated Kriel Mine and Eyesizwe operated Matla Colliery, about 10 km west (Snyman, 1998 and Baker, 1999). Only Matla and Kriel Collieries are also in the Highveld Coalfield while the other neighbours fall inside the Witbank Coalfield. Other mines in the Highveld Coalfield (Fig. 3.2.) are the SASOL owned Secunda Collieries (Brandspruit, Twistdraai, Syferfontein and Bosjesspruit) at Secunda, the Anglo Coal owned New

Denmark Colliery near Standerton and the Total Exploration SA owned Forzando Colliery near Hendrina (Jordaan, 1986 and Barker, 1999).

Various studies were conducted to determine the local and regional stratigraphy as well as the depositional environment of the Highveld Coalfield (Winter et al., 1987). The area studied by Winter et al. in 1987 was seen as part of the Highveld Coalfield at the time but is currently viewed as the western part of the Witbank Coalfield (Snyman, 1998). The seam correlations and depositional environment are similar to the Highveld Coalfield and may still be used for research. Other researchers have done some work in various parts of the Highveld Coalfield since 1928 and include names like Wybergh, W.J. in 1928, Venter, F.A. in 1934, Stanistreet, I.G. et al. in 1980, Smith, D.A.M. in 1970, Cadle, A.B. and Hobday, D.R. in 1977 (Jordaan, 1986).

T.C.S.A. owns all the coal rights over the farms Dorstfontein 71 IS, Welstand 55IS, Fentonia 54IS and Boschkrans 53IS (Fig. 3.3) (Stewardson and Saunderson, 1999). Mining is currently taking place on the farm Dorstfontein 71IS where a high-grade coal, suitable for export and metallurgical applications, is extracted. The study only deals with the very thin seam coal area (heights between 1.2 and 1.4m) at Dorstfontein 71IS, which was until recently been regarded as un-mineable and thus excluded from reserves.

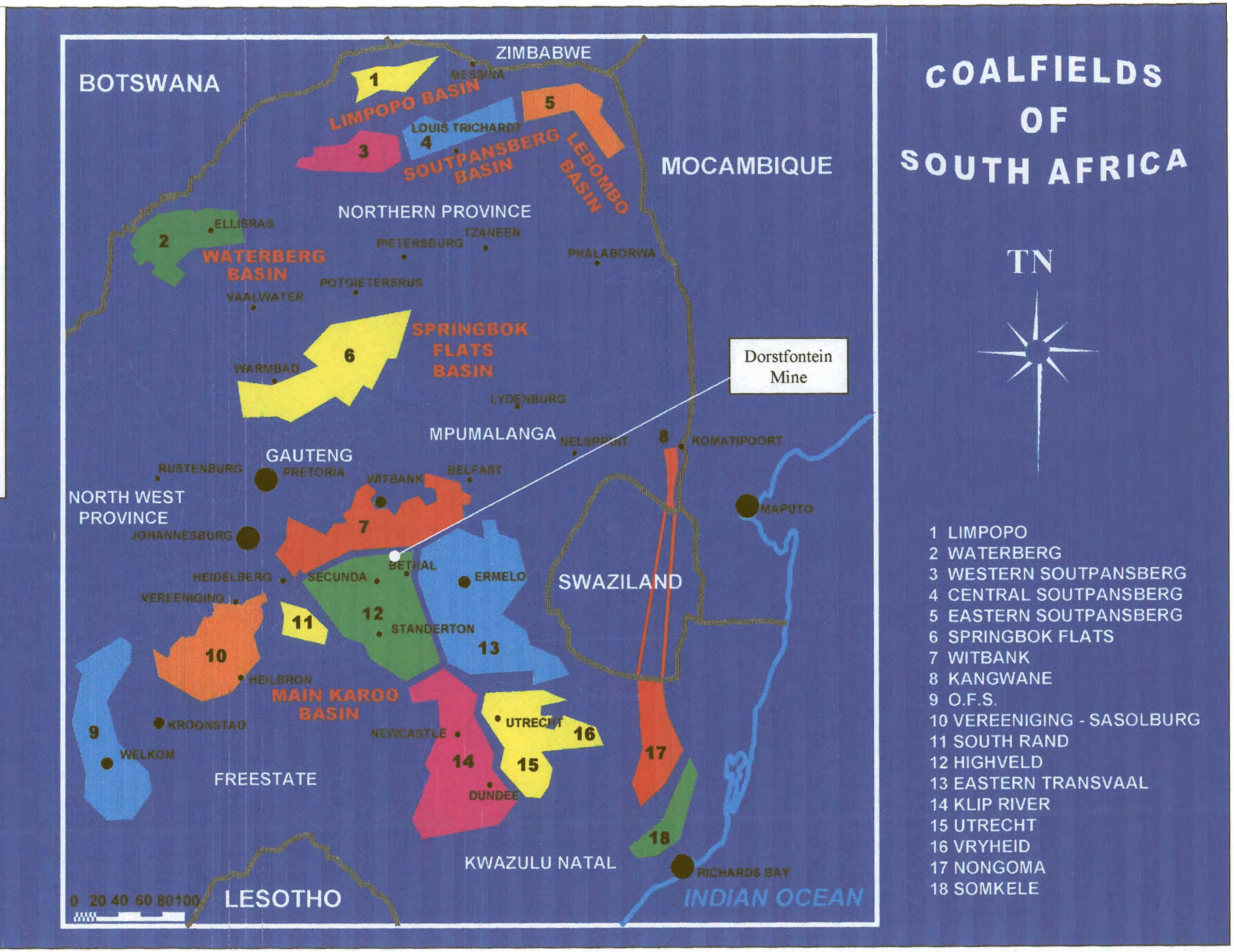


Fig. 3.1. Coalfields of South Africa. (Source unknown)

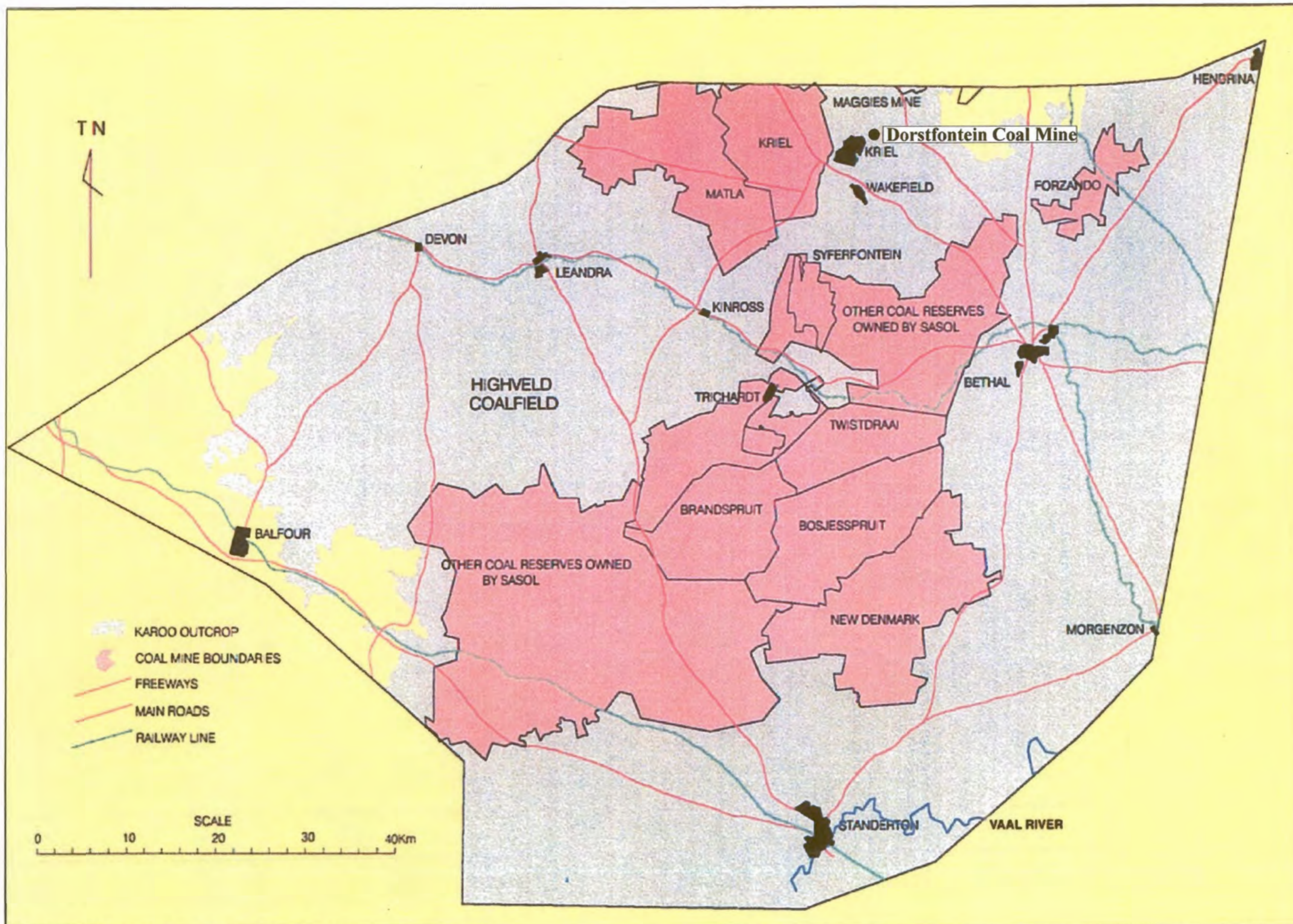


Fig. 3.2 Coalfield Boundaries and Coal Mines of the Highveld Coalfield (after Barker, 1999)

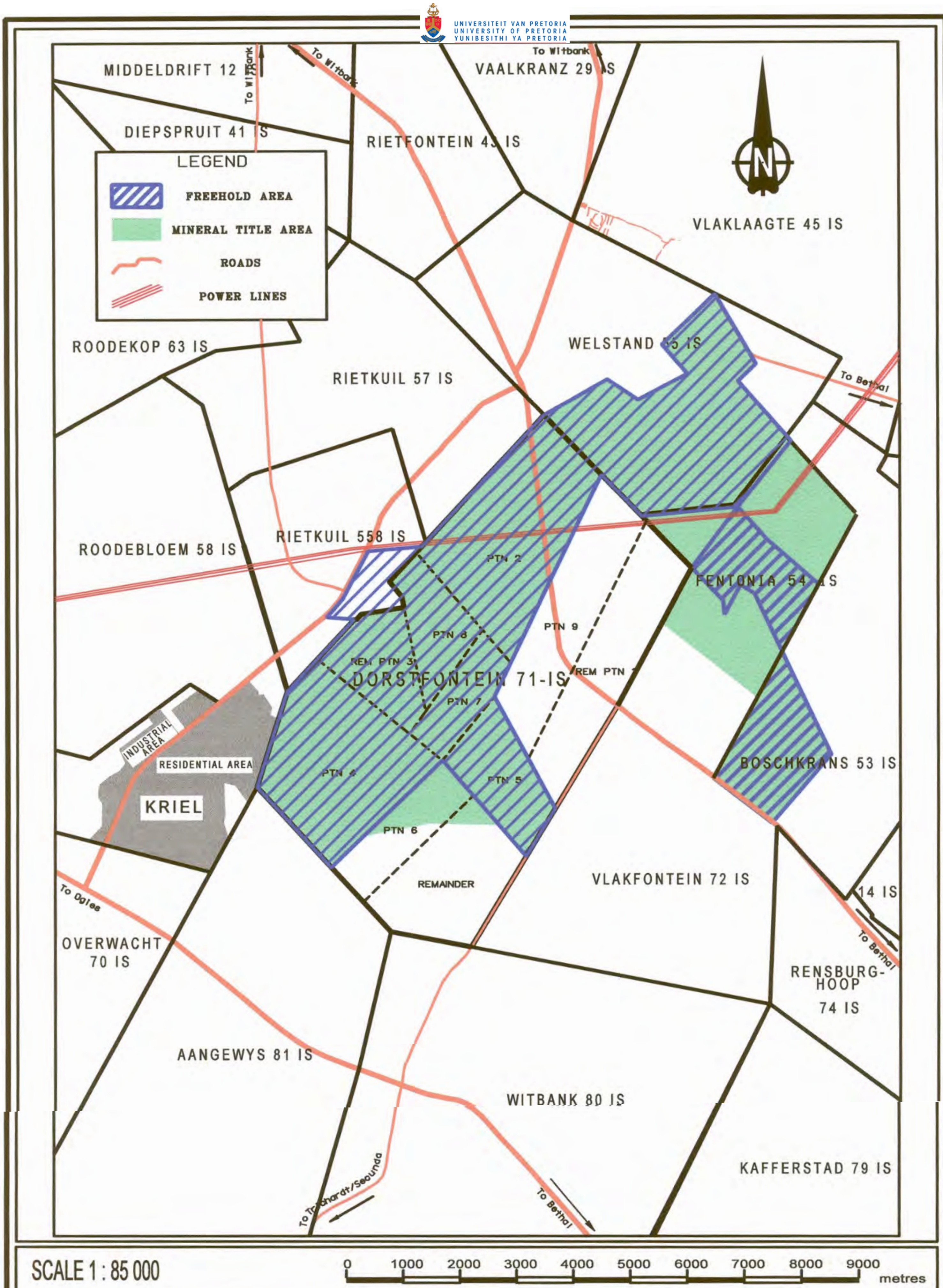


Fig. 3.3. Locality Plan and Mineral Rights. (after Stewardson and Saunderson, 1999)

3.1.2 Topography and land usage

The topography is gently undulating (Fig. 3.4) with a few small tributaries of the Steenkoolspruit draining the property. The previous farmer or owner constructed a few farm dams on the property. The T.C.S.A. owned surface is currently being rented out to farmers who use it for maize cultivation and grazing. The property is sparsely populated by a few farm workers staying in workers huts (Stewardson and Saunderson, 1999). The use of bord and pillar mining methods and the properly designed pillars, prevent surface subsidence. In terms of sustainable development objectives, the surface should be returned to its original use for agriculture as minimal negative impacts on the surface was done by mining.

3.1.3 Mineral Rights

T.C.S.A. owns all of the mineral rights in the mining lease area (Stewardson and Saunderson, 1999). These rights were acquired by AngloVaal Minerals in the 1980s and 1990s and transferred to T.C.S.A. with the selling of Dorstfontein in 1999. Adjacent mineral rights owners are:

- Anglo Coal Plc and
- Mr. N.E. Hirschowitz

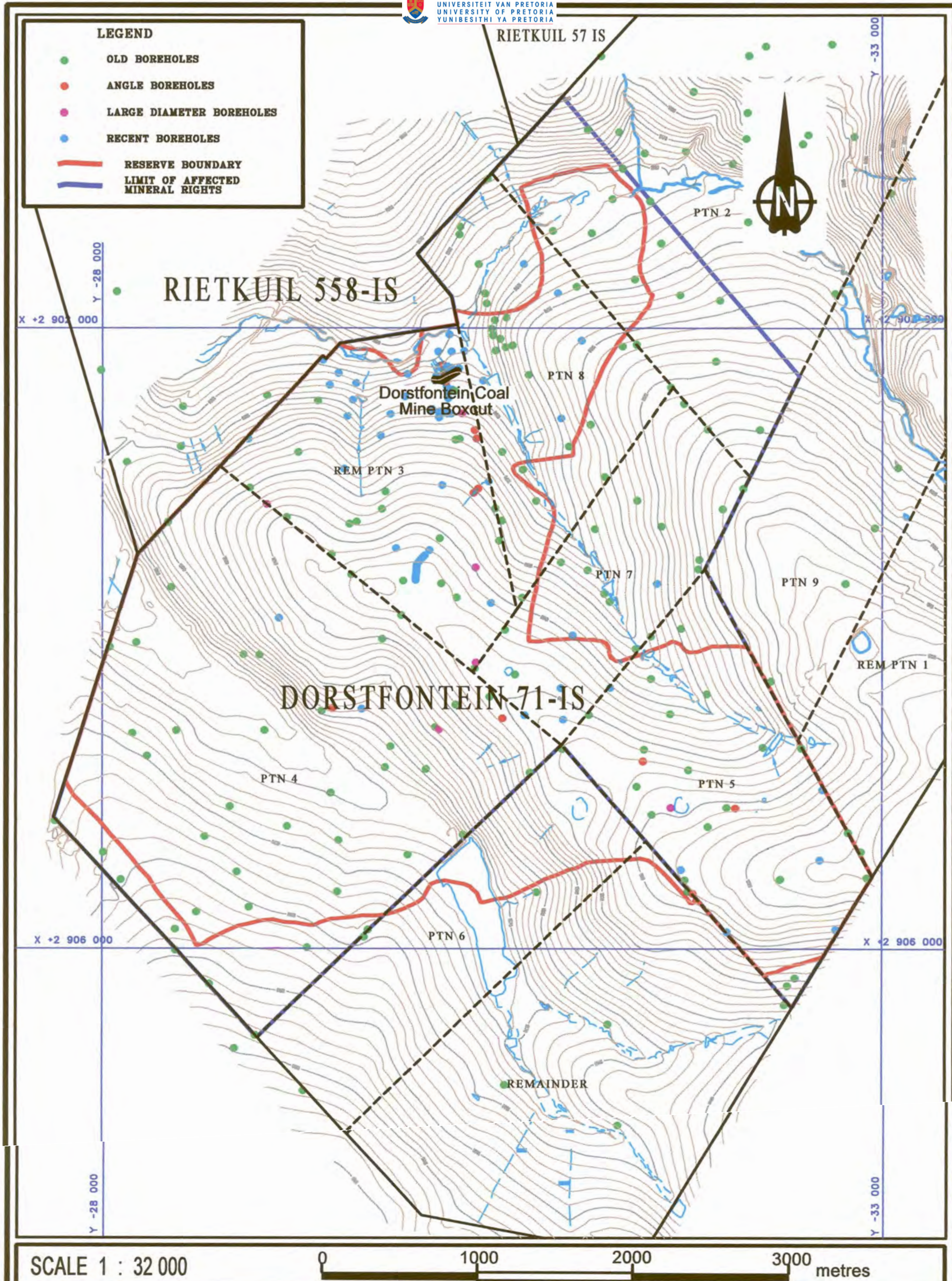


Fig. 3.4. Surface Topography and Borehole Plan.
(after Stewardson and Saunderson, 1999)

3.2 Exploration

Since the early 1960's up to 1999 a total of 174 holes were drilled in the then Dorstfontein resource area of which 19 holes were angled holes to confirm dolerite dyke positions (Fig. 3.4) (Stewardson and Saunderson, 1999). Subsequently another 64 holes were drilled in the reserve area since mining started in 1999.

Anglo American Corporation carried out the earliest exploration in the mid-1960s. Between 1974 and 1975, South Cape Exploration (Pty) Ltd drilled 47 holes on Dorstfontein. A further 43 holes were drilled by Sun Mining and Prospecting during the period 1975 to 1978 (Stewardson and Saunderson, 1999). These holes had limited washability data for the No. 2 Seam as only the No. 4 Seam was prospected for (see Stratigraphical Log, Fig. 3.5). In some cases only proximate analysis were performed on raw coal from the No. 2 Seam. All of the prospecting companies cancelled their optioning agreements and prospecting rights as the No. 4 Seam is of inferior quality and regarded as uneconomical. Options were taken out by AngloVaal Minerals when they considered the No. 2 Seam as mineable. This company drilled another 60 boreholes between 1980 and 1982 with a further 105 holes between 1996 and 1998. All AngloVaal Minerals' boreholes and subsequent T.C.S.A. holes were analyzed at 10 density fractions to get a better understanding of the washability of the coal.

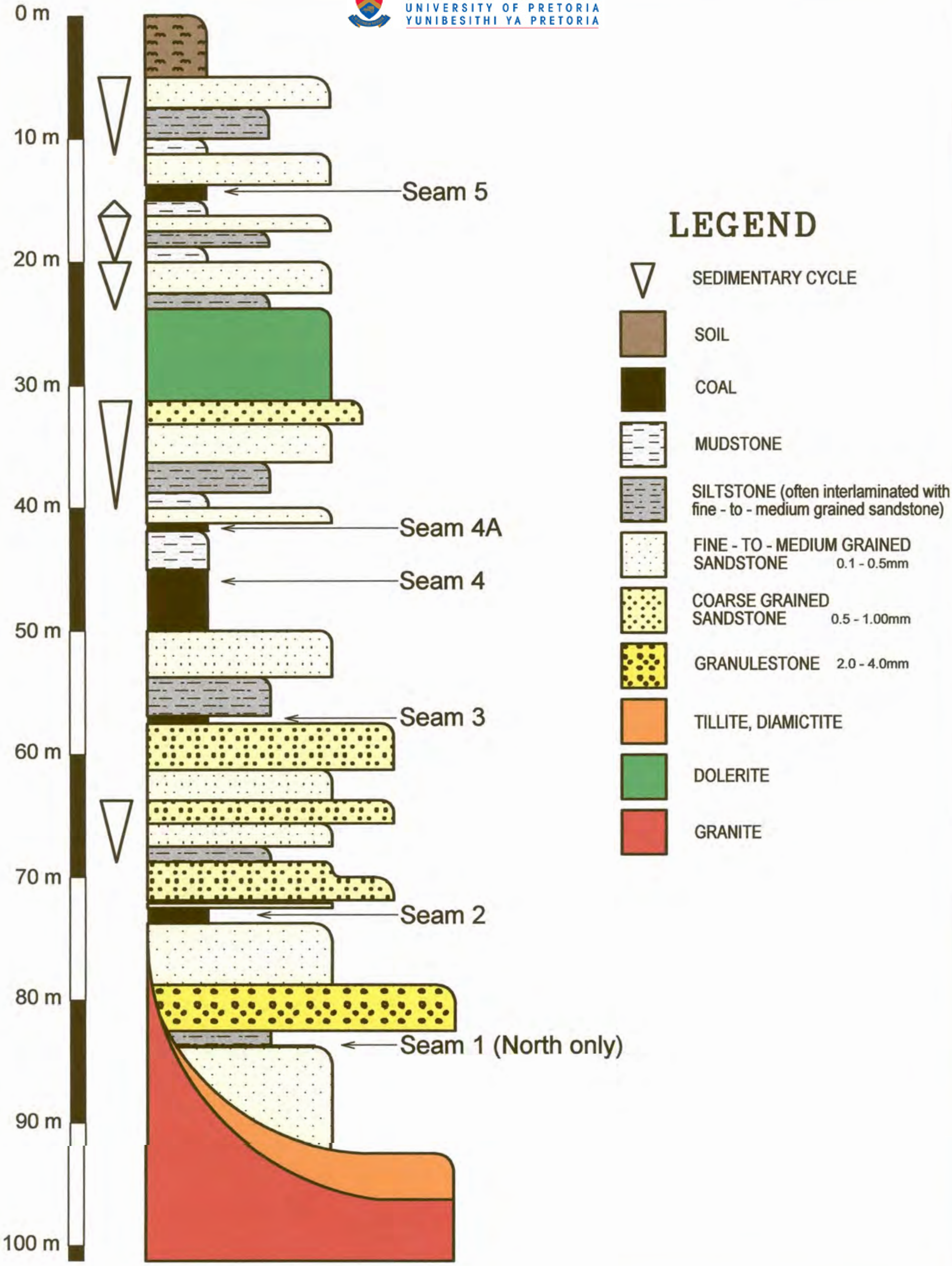
In 1995 a helicopter-borne high resolution aeromagnetic survey was conducted to define magnetic dykes (Stewardson and Saunderson, 1999). Some anomalies were confirmed by drilling angled holes and by ground magnetometry. In 1997 a helicopter-borne EM survey was carried out to define some non-magnetic dykes. Anomalies were identified and angled boreholes drilled which confirmed some of these anomalies to be dolerite dykes (Stewardson and Saunderson, 1999). Most of the major dolerite dykes in the mining area were correctly predicted and very few surprises were encountered during mining. Only a few thin dolerite dykes/stringers were intersected during mining and a few situations the positions of the major dykes were out by not more than 25 meters.

3.3 Stratigraphy

The Pre-Karoo basement rocks consist of granite of the Nebo Granite Suite of the Bushveld Complex and in a few places Transvaal shales and sandstones (SACS, 1980). The granite outcrops close to the box-cut position and defines the northern mining reserve boundary. The basement is overlain unconformably by diamictites and associated glacial sediments of Dwyka age (Winter et al., 1987). These in turn are conformably overlain by sediments of the Vryheid Formation that comprise of a series of stacked upwards-coarsening sequences of siltstone and sandstone. Each sequence is capped by a coal seam (Fig. 3.5).

Five major seams are present and numbered from the base upwards as Seams No. 1 to 5 (Snyman, 1998 and De Jager, 1976). Thickness and distribution of the seams were controlled by paleotopography as well as pre- and syndepositional events (Winter et al., 1987). The best developed and most extensive seam is the No. 4 Seam which reaches maximum thicknesses of up to seven meters. Unfortunately this coal has a very low yield for export products and the calorific value and volatile matter of the seam renders it only suitable for use as steam coal. Currently an oversupply of this type of coal exists but there is always the possibility that some market might become available in the future. The No. 5 Seam is developed only in the topographically elevated areas and the negative experience of other No. 5 Seam producers discourages any mining of this seam. The No. 1 Seam is only locally developed in a small palaeo-valley in the northeast of the mining reserve. It is of inferior quality and uneconomical. The No. 3 Seam is very localized and thin and occurs only in a few places in the deposit. Currently the No. 2 Seam is the only economic viable seam in the deposit and a detailed description is to follow (Fig. 3.6).

Late Jurassic time dolerite intrusions, which coincided with the Gondwana breakup, have resulted in some areas of burnt and or devolatilised coal (Jordaan, 1986). The migration of dolerite sills to different stratigraphical levels had resulted in seam displacement but had only a limited effect on the No. 2 Seam reserve area. The eastern mining reserve boundary has been defined using such a migrating sill as reserve limit (Stewardson and Saunderson, 1999).



SCALE 1 : 500

Fig. 3.5. General Stratigraphic Log. (after Stewardson and Saunderson, 1999)

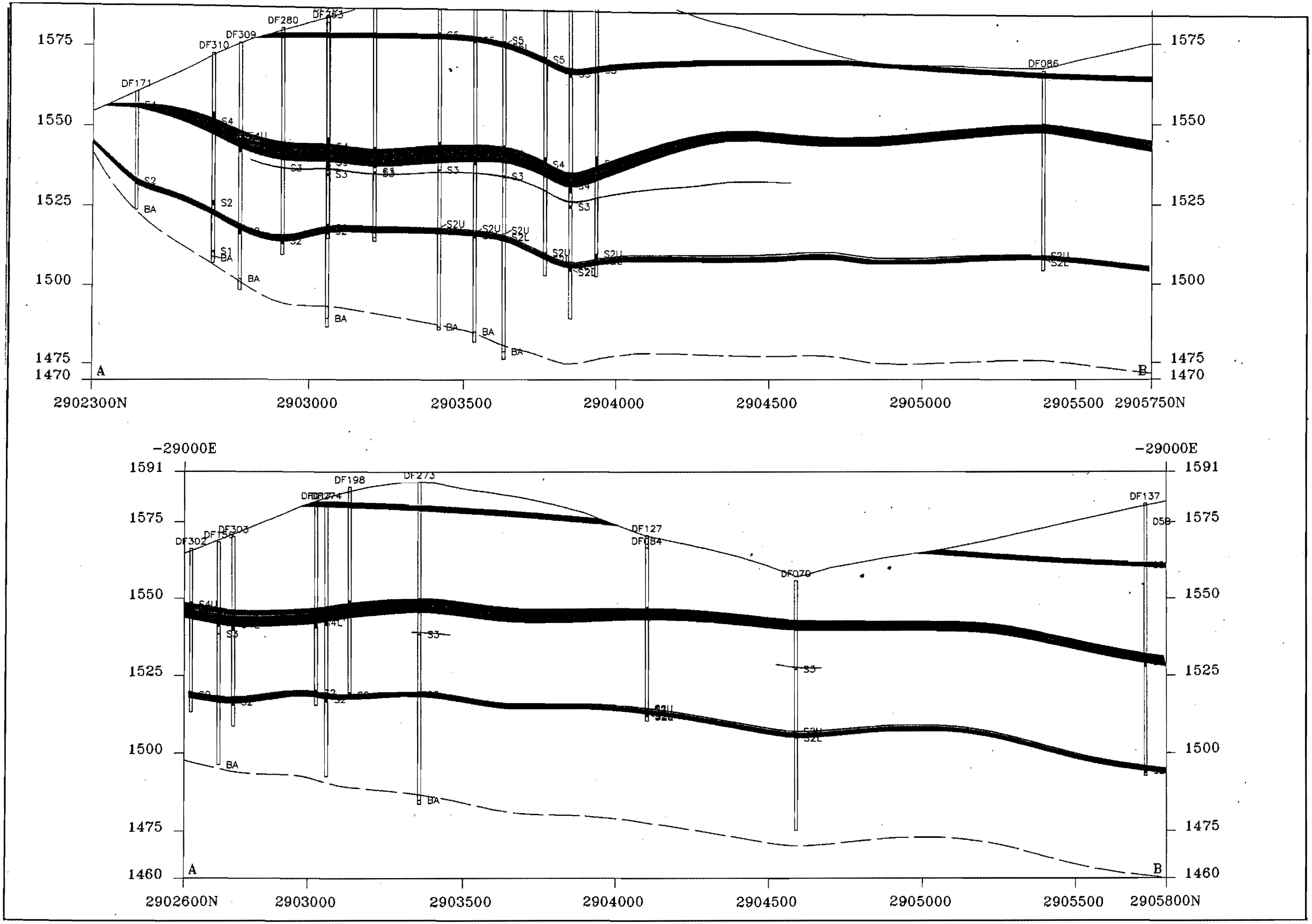


Fig. 3.6. North-South cross-section of the Dorstfontein Deposit (after Bartkowiak, 2001).

3.4 No. 2 Seam and No. 2 Lower Seam

The palaeo-basement geometry determined the geometry and thickness of the No. 2 Seam (Fig. 3.6 and 3.7). The rate at which the surface subsided during peat accumulation controlled the thickness and character of the coal. Height variations can be attributed to pre- and syndepositional geological events (Stewardson and Saunderson, 1999).

3.4.1 Seam splitting

The single coal seam in the north is split into an upper and lower seam in the south by a persistent sandstone parting (Fig. 3.7 & 3.8). The parting is positioned towards the top of the seam and ranges from 0.0 to 0.75m in thickness. The No. 2 Upper Seam is thin (0.01 to 0.35m thick) and only the No. 2 Lower Seam forms an economic unit. In the No. 2 Thin Seam area the parting is thick and as only 0.3m is enough to form a safe beam, this parting will form a proper roof for the lower, mineable part of the No. 2 Seam (Spengler, pers. comm., 2002).

3.4.2 Seam Elevation

The elevation of the base of the No. 2 Thin Seam ranges from the 1511 to 1518m AMSL (Fig. 3.9). The seam topography reflects the Pre-Karoo relief with the seam dipping gently from east to west towards a north-south trending paleovalley. The overall regional dip of the seam is from north to south, that is from the granite outcrop towards the depositional basin. In the study area the coal seam is flat with a barely noticeable dip towards the south.

3.4.3 Seam Thickness

The total thickness of the No. 2 Seam, including the parting, is illustrated in Fig. 3.10. The central area of maximum thickness reflects the zone of maximum parting thickness. In the study area the seam thickness below the parting ranges from 1.2 to 1.4m (Fig. 3.11).

In the area where the seam splitting occurs, the No. 2 Upper Seam is developed and ranges in thickness from 0.01 to 0.35m. There is no correlation between the No. 2 Upper Seam thickness and the underlying parting thickness. The clean, well-sorted sandstone that overlies the No. 2 Upper Seam has generally a thin, silty zone at its base. This suggests disturbance of the peat surface during transgression. The absence of rip-up clasts indicates little or no erosion of the seam (Stewardson and Saunderson, 1999).

3.4.4 Main Parting

The parting thickness ranges from 0.0 to 0.75m with its maximum thickness in an east-west linear zone (Fig. 3.8). The parting consists of an upwards-coarsening sequence grading from lenticular-laminated siltstone through interlaminated sandstone-siltstone to cross-laminated sandstone at the top. The lithology and geometry suggested a crevasse splay deposit, which emanated from a channel system in the east of the reserve area (Stewardson and Saunderson, 1999).

Mechanical strength tests were done on core from the 2002-drilling programme (Spengler, 2002). The results indicated that the parting is competent and will not collapse during mining and that it will form a safe beam if bolted with full column resin roofbolts. The only provision is that the mining method below the parting should not be the conventional drill and blast methods but preferably mechanical continuous mining methods. Mechanical mining methods cause the least disturbance and possible separation of the laminated strata, which could result in the beam thinning to dangerous proportions. In Chapter 7 there is a detailed discussion on the testing of the parting and instructed support pattern.

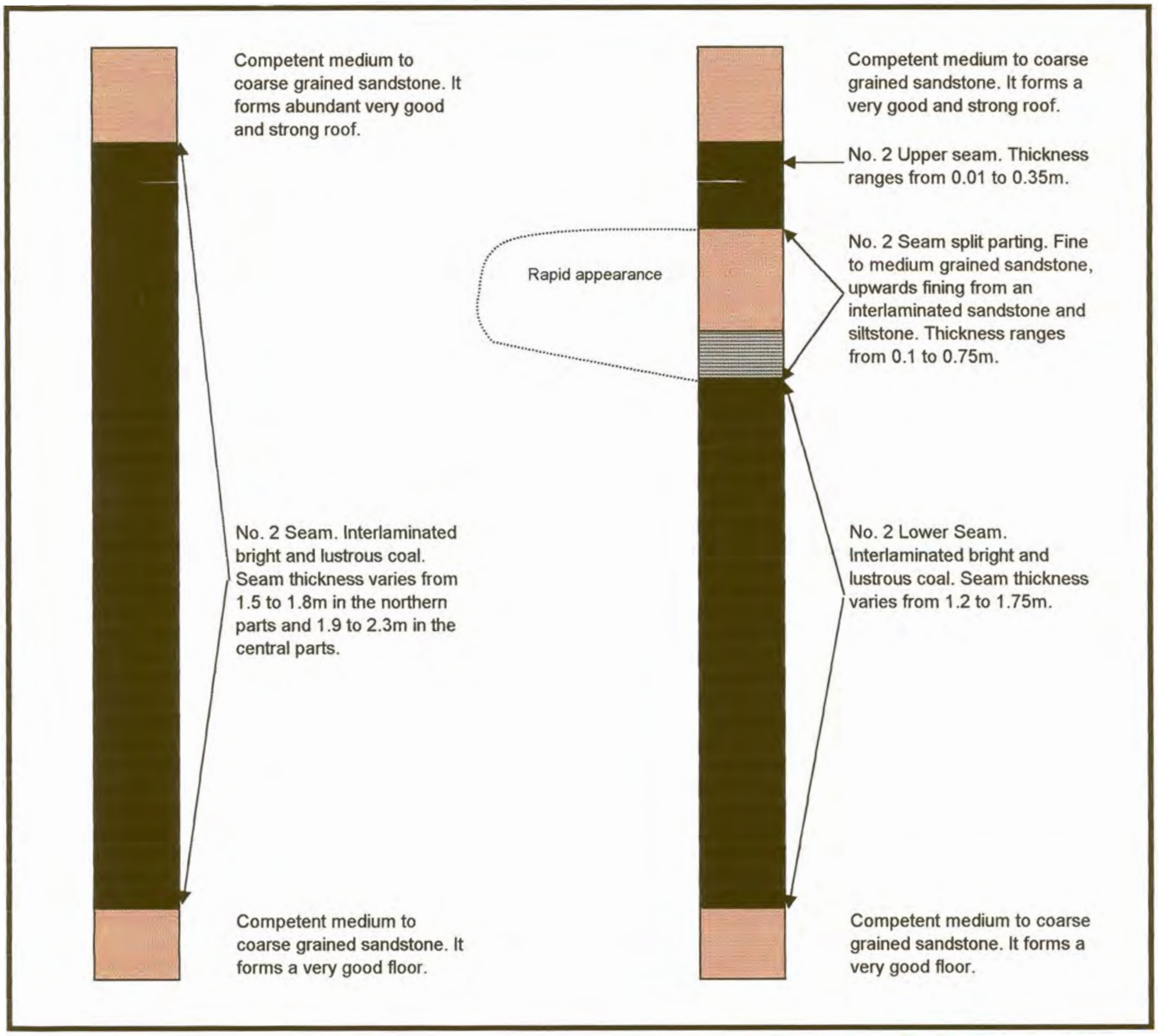


Fig. 3.7. No. 2 Seam and No. 2 Seam Seam-Split Parting.

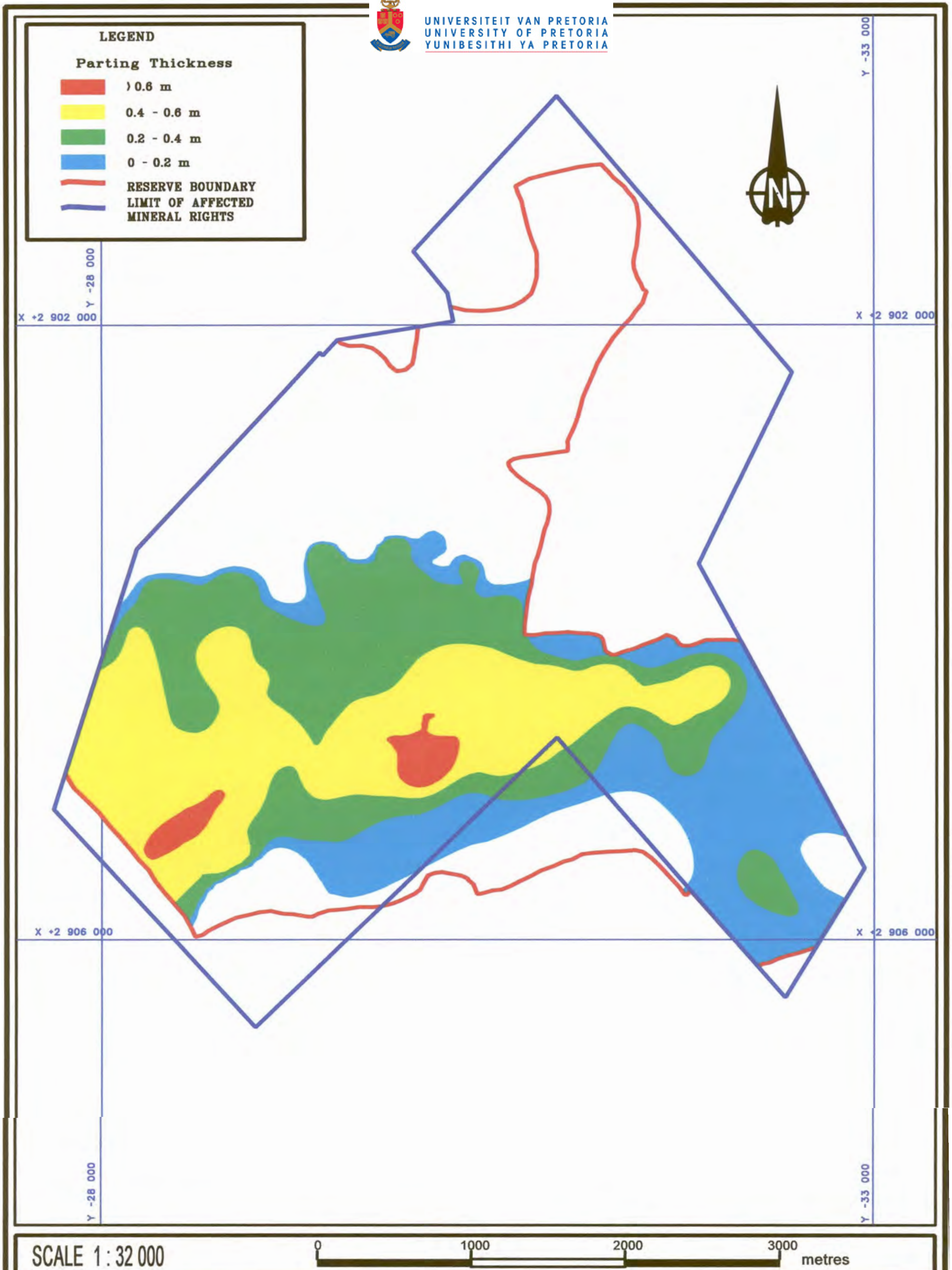
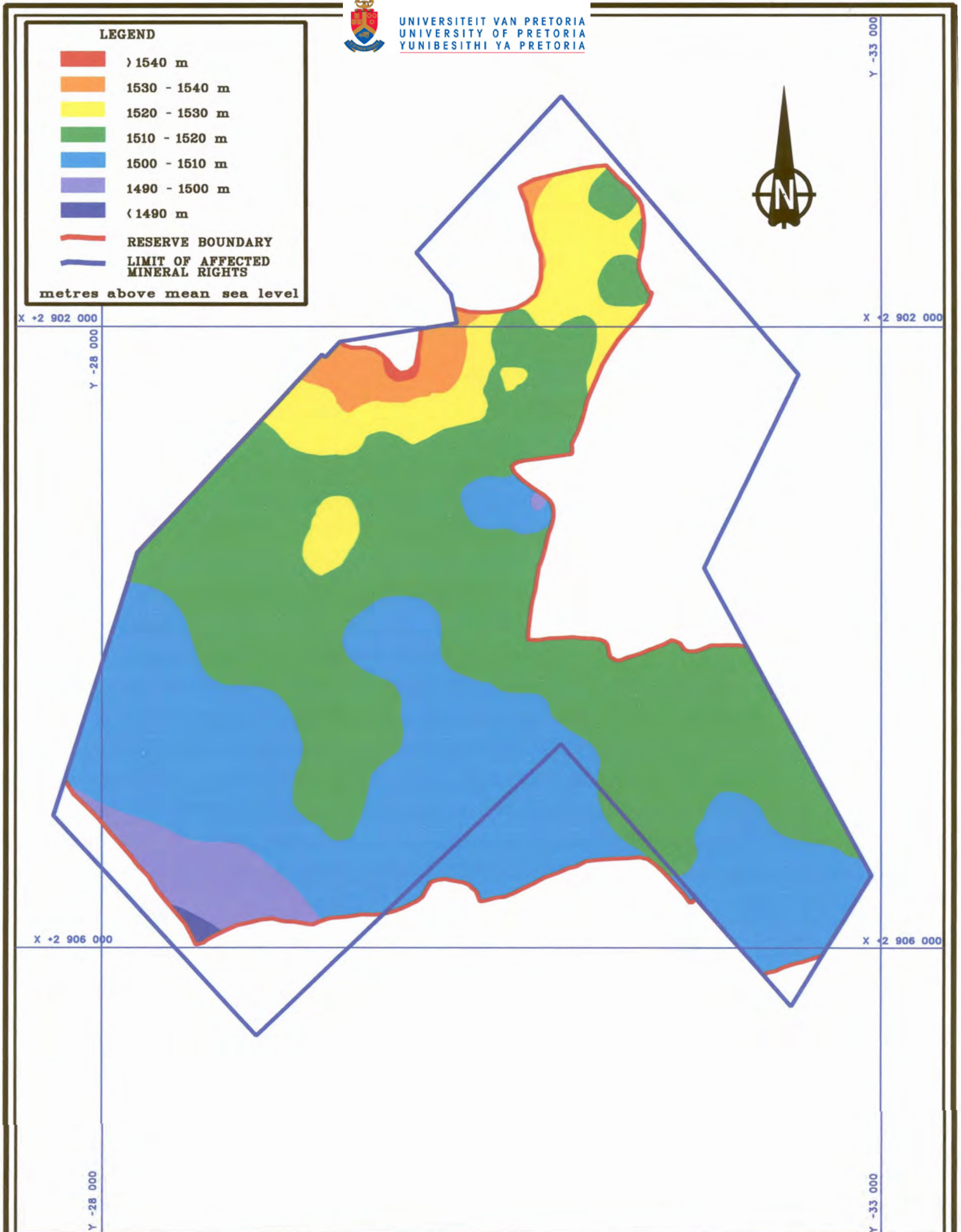


Fig. 3.8. Seam-Split Parting Thickness and Position.
(after Stewardson and Saunderson, 1999)



SCALE 1 : 32 000 — 0, 1000, 2000, 3000 metres

Fig. 3.9. Base elevation of the No. 2 Seam. (after Stewardson and Saunderson, 1999)

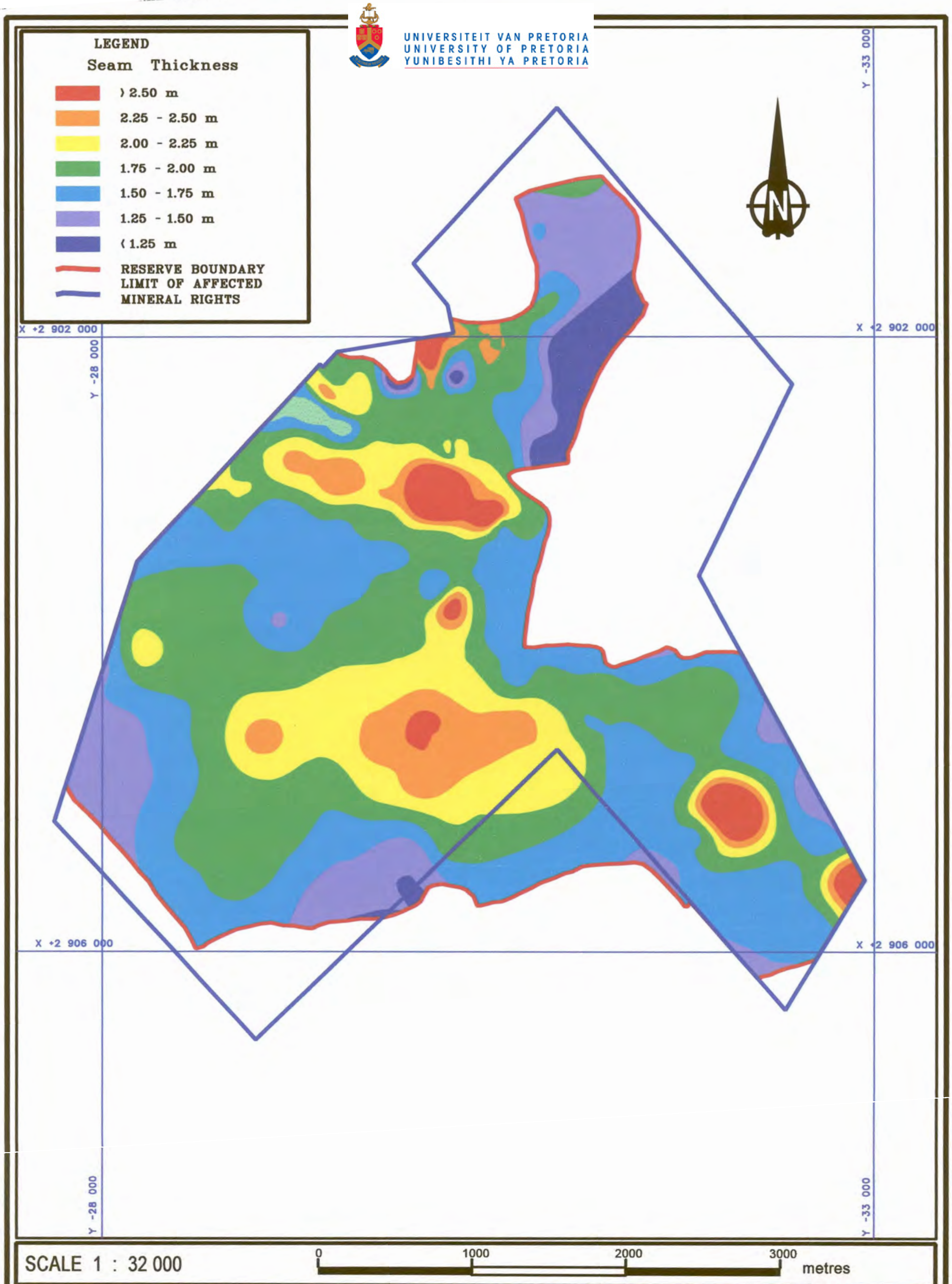


Fig. 3.10. Total Thickness of the No. 2 Seam, including Seam-Split Parting.
(after Stewardson and Saunderson, 1999)

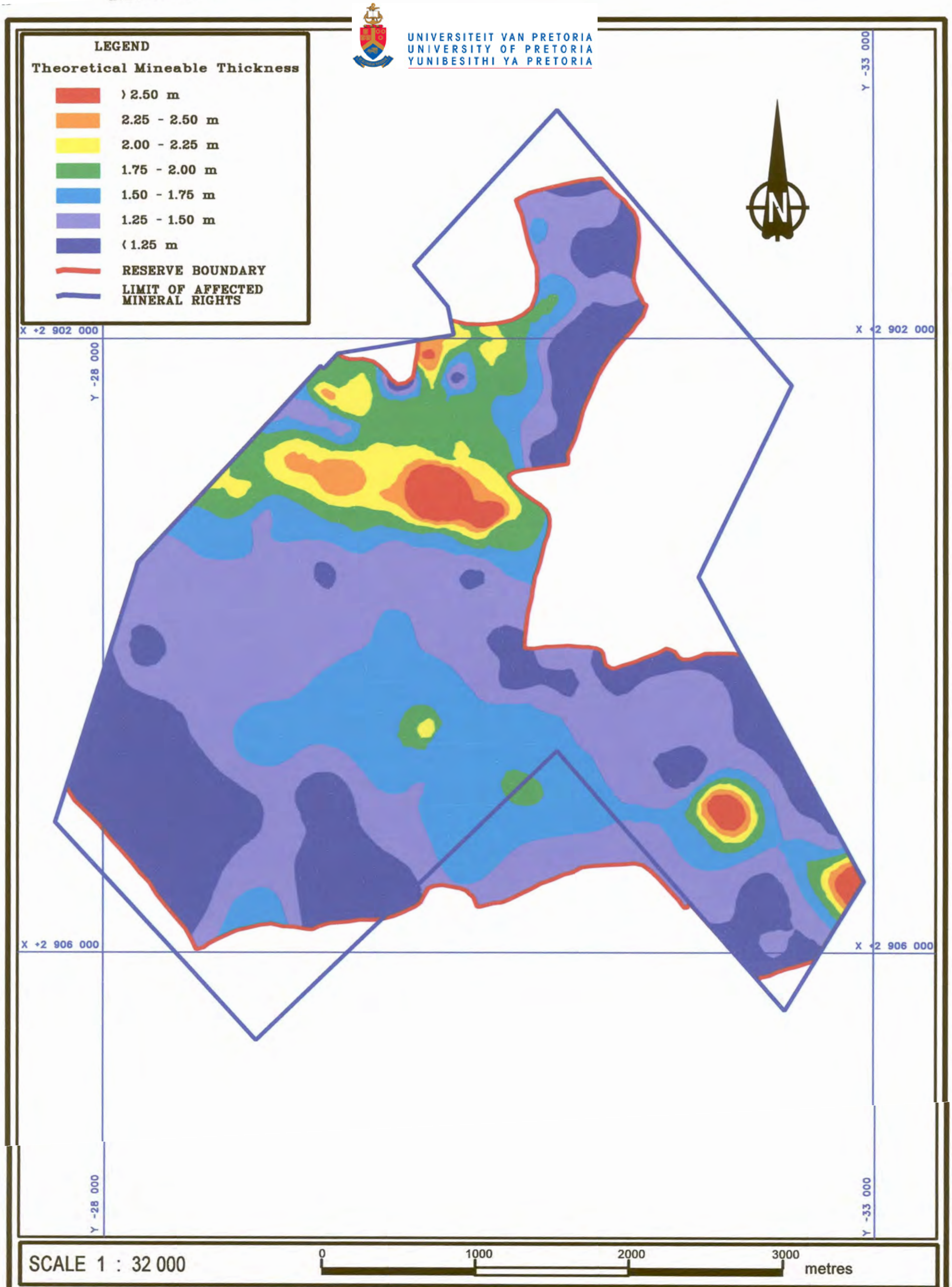


Fig. 3.11. Mineable Seam Thickness of the No. 2 Thin Seam.
(after Stewardson and Saunderson, 1999)

3.4.5 Seam Roof

The purpose of this study is to determine the result and affect if the seam-split parting forms the roof in the study area. However, it would be necessary to do roof stripping (parting) in the belt road and main travel roads to increase heights for the people and vehicles to move. The stripping, normally done to a height of 1.8m, will expose the overlying fine grained, homogeneous, clean and well-sorted sandstone unit, which currently forms the roof. This unit is mostly unbedded and lack silty laminae. Occasional occurrences of bioturbation and cross trough bedding are developed. These occurrences do not have any negative effects on overall rock strength (Stewardson and Saunderson, 1999). All roof rock (parting) will be mined as a second cut and be stowed underground to prevent contamination of the mined coal.

3.4.6 Seam Floor

Competent, medium grained sandstone underlies the seam. The sandstone floor forms the final depositional stage of a prograding delta platform upon which the coal seam developed (Stewardson and Saunderson, 1999). In currently mined areas and old workings, the floor is still competent and did not scale or break-up during vehicle movements. It is expected to behave the same in the thin seam areas.

3.5 Dolerite Intrusions

Magnetic and non-magnetic dykes as well as magnetic dolerite sills occur (Fig. 3.12). These were detected using both geophysical surveys and borehole intersections (Stewardson and Saunderson, 1999). In the study area and the current reserve, dolerite sills do not underlie the mineable seam. In the east of the current reserve a dolerite sill cuts vertically across the strata to outcrop on surface. It underlies the No. 2 Seam in the east, displaces the seam upwards the same distance as the dolerite thickness and thus renders the coal inaccessible and unmineable due to this discontinuity. This position of the sill transgression was used to define the eastern boundary of the current mineable reserve.

In the south of the study area a major magnetic dyke was identified using an aeromagnetic survey. It trends more or less east - west and is near vertical. Mining through this dyke has proved its thickness to be 2.8m at the locality it was intersected.

In the western part of the study area, 3 dykes occur. Mining confirmed their positions during a southern development towards higher seam areas, the so called South Main area. All of these dykes were relatively thin (< 2 m thick) and had no serious effect on the coal seam. It is concluded that these dykes should not pose any serious problem for mining the thin seam area.

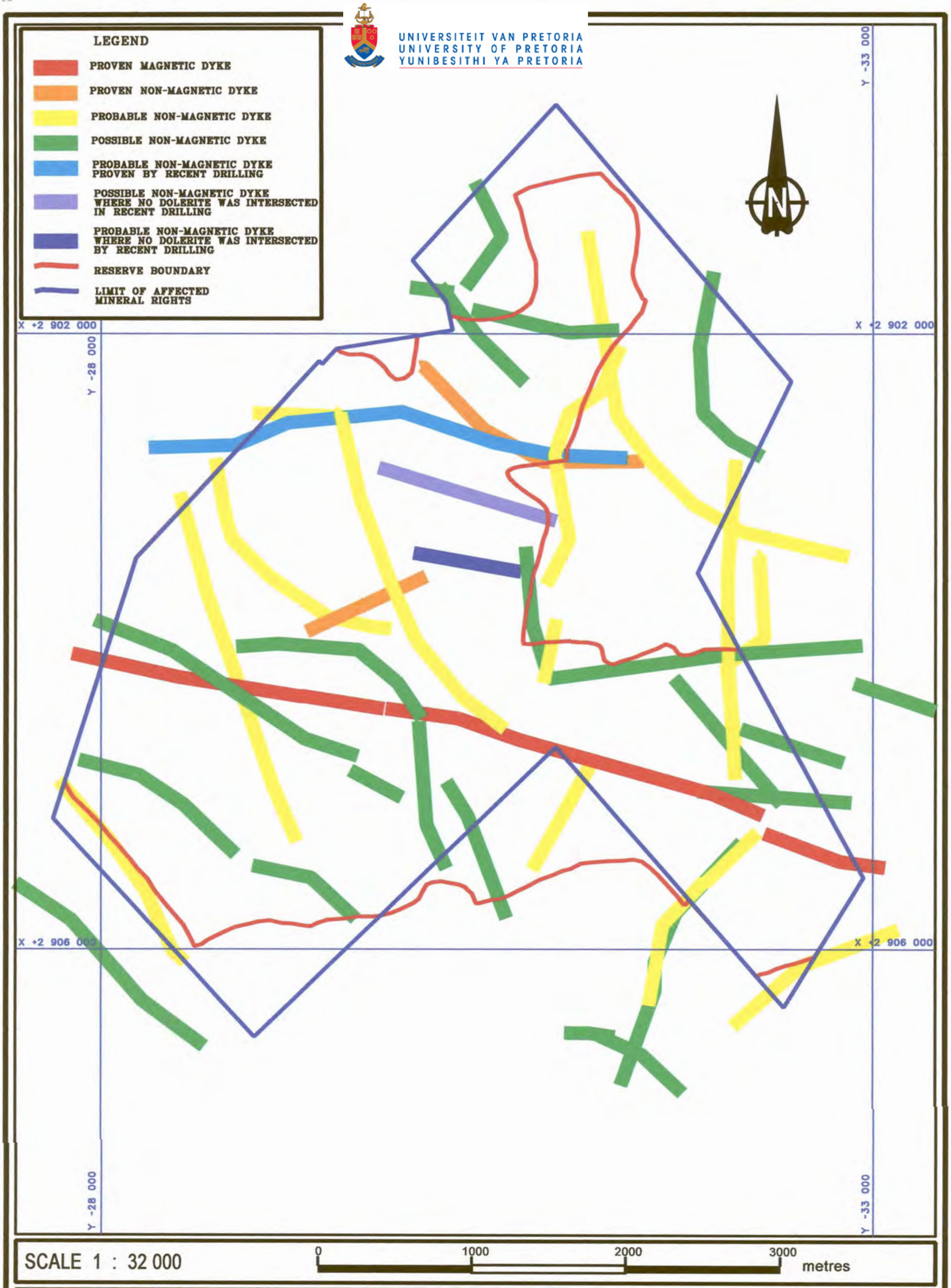


Fig. 3.12. Dolerite Positions. (after Stewardson and Saunderson, 1999)

3.6 Resource estimate and Grade

3.6.1 Summary of Resources and Grade

Tabulated below is the spread of the No. 2 Seam resource at various height intervals.

Table 2. Resources for the No. 2 Thin Seam area (Bartkowiak, 2002).

Thickness Interval (m)	Thickness Cut-Off (m)	Mass per Interval (kt)	Cumulated Mass (kt)	Percentage of Total (%)	Cumulative Percentage
< 1.0	1.0	335.21	335.21	1.17	1.17
1.0 to 1.1	1.1	233.16	568.37	0.82	1.99
1.1 to 1.2	1.2	1513.29	2081.66	5.30	7.29
1.2 to 1.3	1.3	2960.82	5042.48	10.37	17.66
1.3 to 1.4	1.4	4100.36	9142.84	14.36	32.02
1.4 to 1.5	1.5	3707.62	12850.46	12.99	45.01
1.5 to 1.6	1.6	4325.97	17176.43	15.15	60.16
1.6 to 1.7	1.7	5176.15	22352.58	18.13	78.29
1.7 to 1.8	1.8	2011.15	24363.73	7.04	85.33
1.8 to 1.9	1.9	774.86	25138.59	2.71	88.05
1.9 to 2.0	2.0	1007.57	26146.16	3.53	91.58
above 2.0	above 2.0	2405.01	28551.17	8.42	100.00

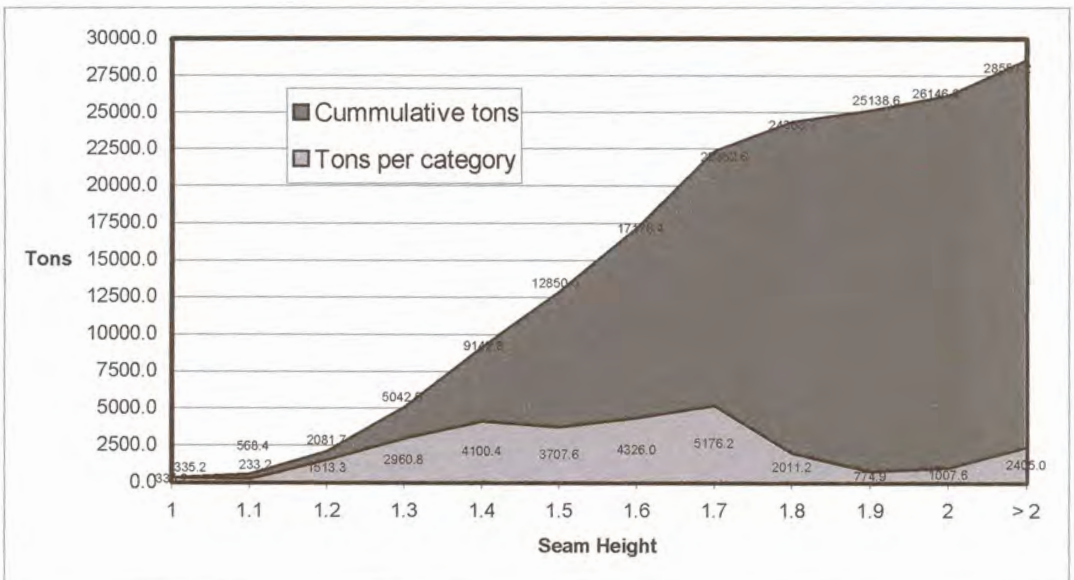


Fig. 3.13. Tonnage distribution against seam heights.

As can be seen, the 1.2 to 1.4m resource constitute 24.73% of the total No. 2 Seam resources. The 1.2 to 1.4m intervals will form 26.68% of the mineable resource if the tonnages below 1.2m are omitted. The estimated mineable resource of the No. 2 Thin Seam coal is 7.06 mt in situ.

Tabulated below is the theoretical washtable for the thin seam resources in the study area. Detailed descriptions of the coal qualities and product parameters will follow in the next paragraphs. Currently Dorstfontein targets the metallurgical market that specifies a minimum calorific value of 26.0 MJ/kg, volatile matter above 26.5%, sulphur content below 1.0%, low ash content and very low phosphorous content (< 0.010%).

Table 3. Average washtable for the No. 2 Thin Seam area (Air dried) (Bartkowiak, 2002)

Float	Yield	CV	Ash	H_2O	Vol	FC	S	Phos
1.35	22.49	30.07	7.21	3.12	32.18	57.49	0.49	0.007
1.37	33.27	32.17	7.57	3.17	32.32	57.95	0.46	0.007
1.40	46.00	29.65	8.14	3.15	30.16	58.56	0.43	0.007
1.45	65.95	29.15	9.23	3.18	28.33	59.27	0.40	0.006
1.50	80.32	28.72	10.25	3.17	27.18	59.41	0.40	0.007
1.60	89.22	28.31	11.25	3.15	26.52	59.08	0.42	0.007
1.70	91.65	28.18	11.59	3.14	26.42	58.86	0.44	0.007
1.80	92.97	27.93	12.20	3.12	26.27	58.42	0.49	0.007
2.20	100.00	26.65	15.54	3.00	25.72	55.75	1.26	0.007

Product	Yield	CV	Ash	H_2O	Vol	FC	S
13.5	95.7	26.1	13.5	3.1	27.4	57.38	0.79
1.6	89.2	28.3	11.3	3.2	26.5	59.08	0.42

From the washtable and product extraction it is clear that the thin seam resource meets the specifications of the established markets. Further more it can be seen that the product yield is high in the regions of the product specifications.

To conclude: the thin seam resource consists of 7.06 mil. tons in-situ coal of the same quality as the current mining reserve. By factoring in an extraction rate of 70% and a geological and mining loss of 10% each, the recoverable (run of mine) tons comes to 3.56 mil. tons. By applying the product yield at a 13.5% ash content (yield = 95.7%) the product tons are 3.41 mil. tons and by applying the yield at RD=1.6 (yield = 89.2%), the product tons are 3.18 mil. tons.

3.6.2 Thin seam resource limits

The main study area is defined by the 1.2 to 1.4 m seam height contour line (Fig. 3.14). A mined-out area forms the northern boundary, while a sill transgression line defines the eastern boundary. There will no other restrictions placed on defining the resource area.

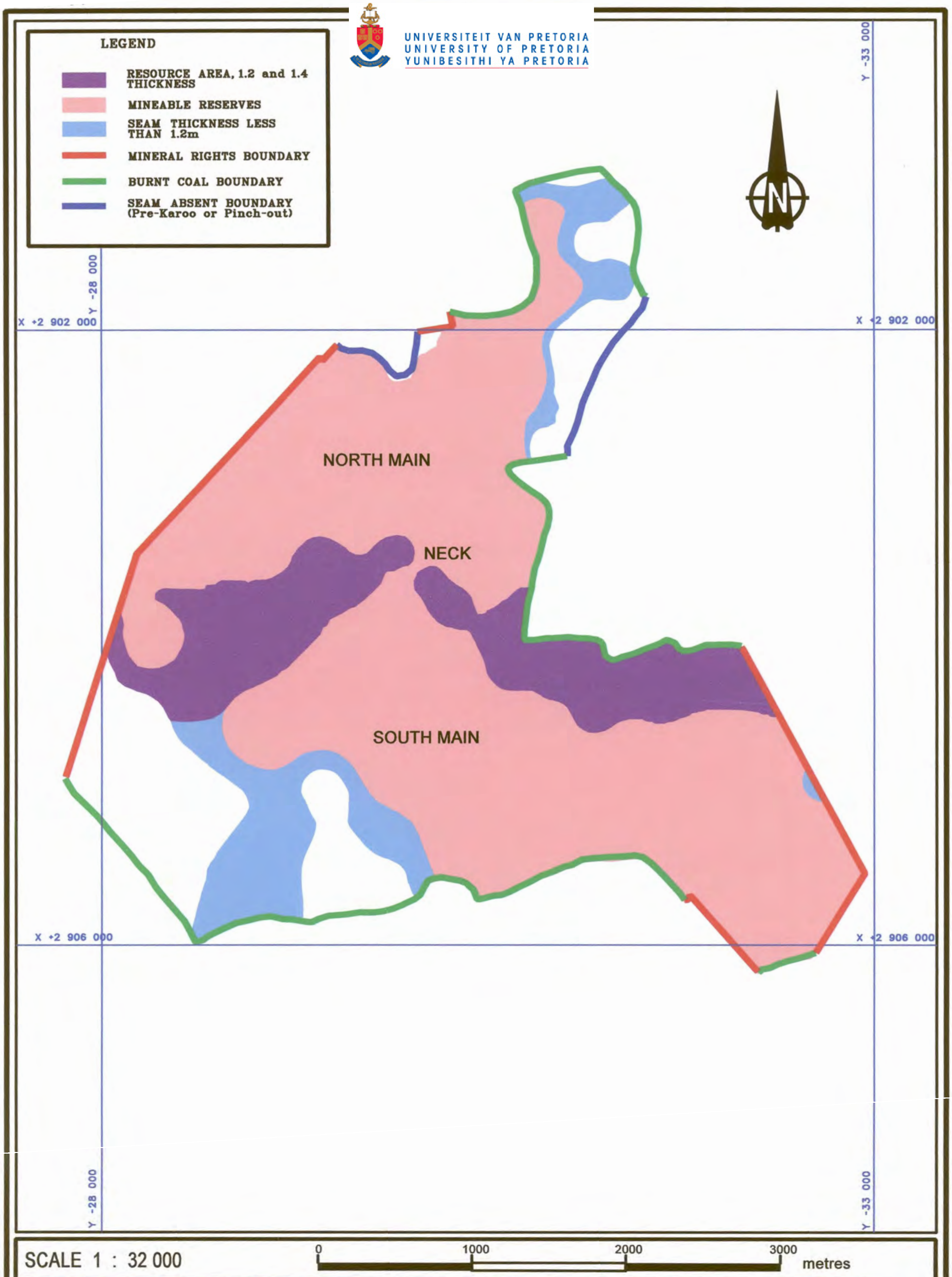


Fig. 3.14. Resource Limits (after Stewardson and Saunderson, 1999)

3.7 Seam Quality

3.7.1 General

For the geological study of 1999, coal quality values were quoted as at R.D.=1.6 on an air-dried basis (Stewardson and Saunderson, 1999). At this wash fraction the mine was viable and the coal could be economically exploited. The quality parameters normally quoted are: Yield, Calorific Value (CV), Ash %, Moisture Content %, Volatile Matter % (Vols), Fixed Carbon % (FC), Sulphur % (S) and Phosphorous % (P). In practice it has been found that it is more practical to wash the coal to achieve 13.5% ash content (Air dry). The market also readily accepted this quality as little change was brought upon the volatile matter and calorific value. Therefore all current qualities are quoted as for an ash content of 13.5%. In the study area the ash content is 11.6% at a R.D. = 1.6. The direct effect of an increase in ash content is an increase in yield. Therefore, in the study area the average yield of 89.2% at R.D. = 1.6 has gone up by 6.5 percentage points to 95.7% at an ash content of 13.5%. This relates to an increase of approximately 2100 tons per month more of saleable coal from the thin seam area alone.

3.7.2 Qualities

Coal and Mineral Technologies, a subsidiary of the SABS, did all resent analysis according to the ISO 1928 standards (The South African Coal Processing Society, 2002). Various other laboratories were used in the past but most of them have closed. Analysis from some of the older borehole data could be used but many of the older holes did not intersect the No. 2 Seam. Since AngloVaal Minerals drilled a 500m grid and T.C.S.A. closed the grid to 250m, enough borehole information exists to confidently predict the coal qualities and tonnage for the thin seam area.

Some of the more important qualities for the thin seam coal are briefly discussed. For Fixed Carbon and Moisture Content, see the details tabulated in Table 4.

3.7.2.1 Yield

The theoretical yield for the thin seam area is 95.7% at an ash content of 13.5% (R.D. = 1.99) and 89.2% at a R.D. = 1.6 (air dry).

3.7.2.2 Calorific Value

The CV in the study area is 28.31 MJ/kg at a R.D. = 1.6 and 27.21 MJ/kg at an ash content of 13.5% (air dry).

3.7.2.3 Volatile Matter

The volatile matter at R.D. = 1.6 is 26.52% and 26.15% at an ash value of 13.5% (air dry), showing very little difference between the two products.

3.7.2.4 Sulphur

It was initially perceived that Dorstfontein had a sulphur problem but the markets steadily accepted slightly higher sulphur values so that the mine is currently meeting all the product specifications. Most of the resource area has an average sulphur content of 0.42% at the R.D. = 1.6 float fraction. At an ash of 13.5% the average sulphur content is 0.79% and in some mining blocks it can go as high as 1.25% because of the free pyrite occupying the cleats. Because of this, the current beneficiation practice to wash to an ash content of 13.5% will not be suitable to produce low sulphur coal. The wash density will have to be reduced to a suitable fraction of between 1.6 and 1.8 to make a low ash and low sulphur product.

3.7.2.5 Phosphorus

The phosphorus content of the entire deposit is below 0.010%. This low value makes the Dorstfontein coal well sought after as a product used in the metallurgical industry.

3.7.3 Additional Analysis

No additional analyses were done on core from boreholes in the study area. It is recommended that the following additional analysis be done for future market requirements (The South African Coal Processing Society, 2002):

- Ultimate Analysis: Carbon, Hydrogen, Nitrogen, and Oxygen.
- Full Ash Analysis: SiO_2, Al_2O_3, Fe_2O_3, TiO_2, CaO, K_2O, SO_3, P_2O_5, MgO, Na_2O.
- Ash Fusion Temperatures.
- Hardgrove Grindability and Abrasiveness
- Forms of Silica.
- Forms of Sulphur.
- Swell and Coking Properties.

It should therefore be concluded that based on the continuity of the No. 2 Seam and the consistency of the seam quality, that a product meeting the market specifications could be produced from the No. 2 Seam thin area.

CHAPTER 4: PREVIOUS AND CURRENT MINING METHODS.

4.1 Introduction

a.) Numerous coal-winning methods have been used on the mine during its four years of existence. The current methods must be judged on the economic factors and their advantages and disadvantages.

b.) During the history of the mine, rapid variations in seam heights were encountered. These were attributed to the irregular nature of the roof and floor. It has been proved that conditions improve as mining proceeds southwards. The roof conditions generally vary according to the mineable portion of the seam selected. Currently the whole seam is mined and the roof conditions have proved to be very good. Isolated instances of roof slumping have occurred, which in turn led to difficult mining conditions in those specific areas.

c.) Some areas have a mudstone roof but even this kind of roof has proved to be competent and the coal mineable.

d.) The floor is generally very competent sandstone.

4.2 Mining method and equipment

4.2.1 General

The bord and pillar mining layout will be maintained because of its reliability, flexibility, low capital cost, low working costs and large skills source availability (Woodruff, 1966). Increasing mechanization has resulted in an increasing production in the amount of the fine coal fractions, which attract significantly lower prices. The introduction of the continuous miner in some areas has decreased the amount of the higher valued coarser fractions. From the start a combination of two conventional drill and blast sections and one continuous miner with a continuous haulage were used. The haulage system was abandoned 1 year ago due to numerous breakages and expensive repairs and its inflexibility in problem areas. It was replaced with 3 Stamler thin seam battery haulers. A revolving stone crew undertakes the development of dykes and does the roof brushing to 1.8m in thinner seam areas. When the need arose a contractor was employed to catch up with the roof brushing and in some cases install additional roofbolts.

4.2.2 Continuous Miner Section

From early days on the trend in bord and pillar mining was towards continuous miners (Woodruff, 1966). More recently there has been an increasing trend in the industry to replace the traditional shuttle cars, battery cars and scoops by continuous haulage systems. The opposite took place at Dorstfontein Coal Mine where shuttle cars are preferred for their flexibility and low running costs.

In the CM-section the continuous miner cuts between 7 and 11 roadways, depending on the preferred layout at the time (Fig. 4.1, 4.2). Pillar and bord widths are 6.8m, giving a coal extraction in the region of 70 to 75%. In Figure 4.1it is illustrated that the CM cuts a split of 6.8m wide to the right of the travel road (marked 1) and while resin bolts are installed in this split the CM cuts a straight (marked 2) and another split (marked 3) of 6.8m wide. During the support of these last two cuts, the CM moves back into the right side of the panel and cuts numbers 4 and 5. Since it is illegal to work under unsupported roof, the CM has to wait while cut 4 is supported before moving to cut number 6 and 7. The whole cycle is repeated and the installation time of the support determines the cutting time of the CM.

Figure 4.2 illustrates the ventilation layout of the CM section. Ventilation is very important for a healthy working environment and even more important in thin seam mining were only small volumes of air can pass through the restricted and narrow workings. The intake air moves in on the right side of the section and ventilates the coal face, removes all the methane and dust and returns on the left side of the panel. Some leakage does occur since the temporary scoop brattices or curtains, installed to direct the air, are not airtight and sealed properly. Some of these temporary curtains are removed to allow the haulers to move from the face to the tip. These curtains are later replaced by brick walls as the section moves forward.

The section is equipped with the following:

- 1 x Joy 12HM15 Continuous miner with a 1,12 meter drum.
- 3 x Thin seam Stamler BH10 Battery Haulers (1m high).
- 1 x Self-propelled thin seam roofbolter.
- 1 x Battery scoop.
- 1 x Feeder-Breaker.
- 1 x Mobile 750 KVA transformer.
- 1 x Mobile switch trailer with flameproof gate end boxes.
- 1 x Portable jet fan.

The manpower is:

- 1x Miner.
- 1x CM operator and assistant.
- 1 x Cable handlers.
- 3 x Hauler drivers.
- 1 x Roofbolter operator and assistant.
- 2 x Feeder-Breaker operators.
- 7 x General labourers.

The total number of persons per shift is 16.

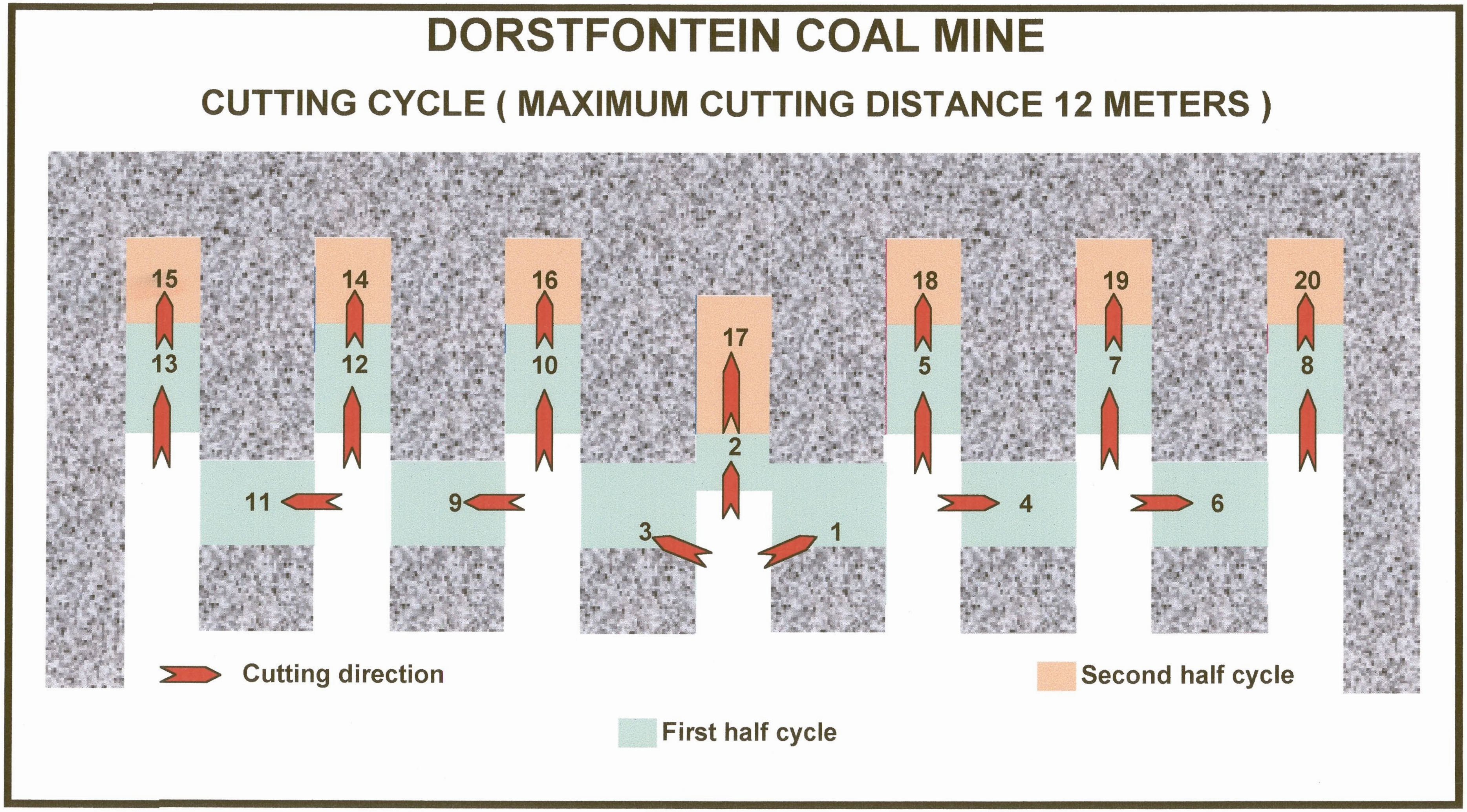


Fig. 4.1. Cutting sequence for CM section (Van Zyl, 2001)

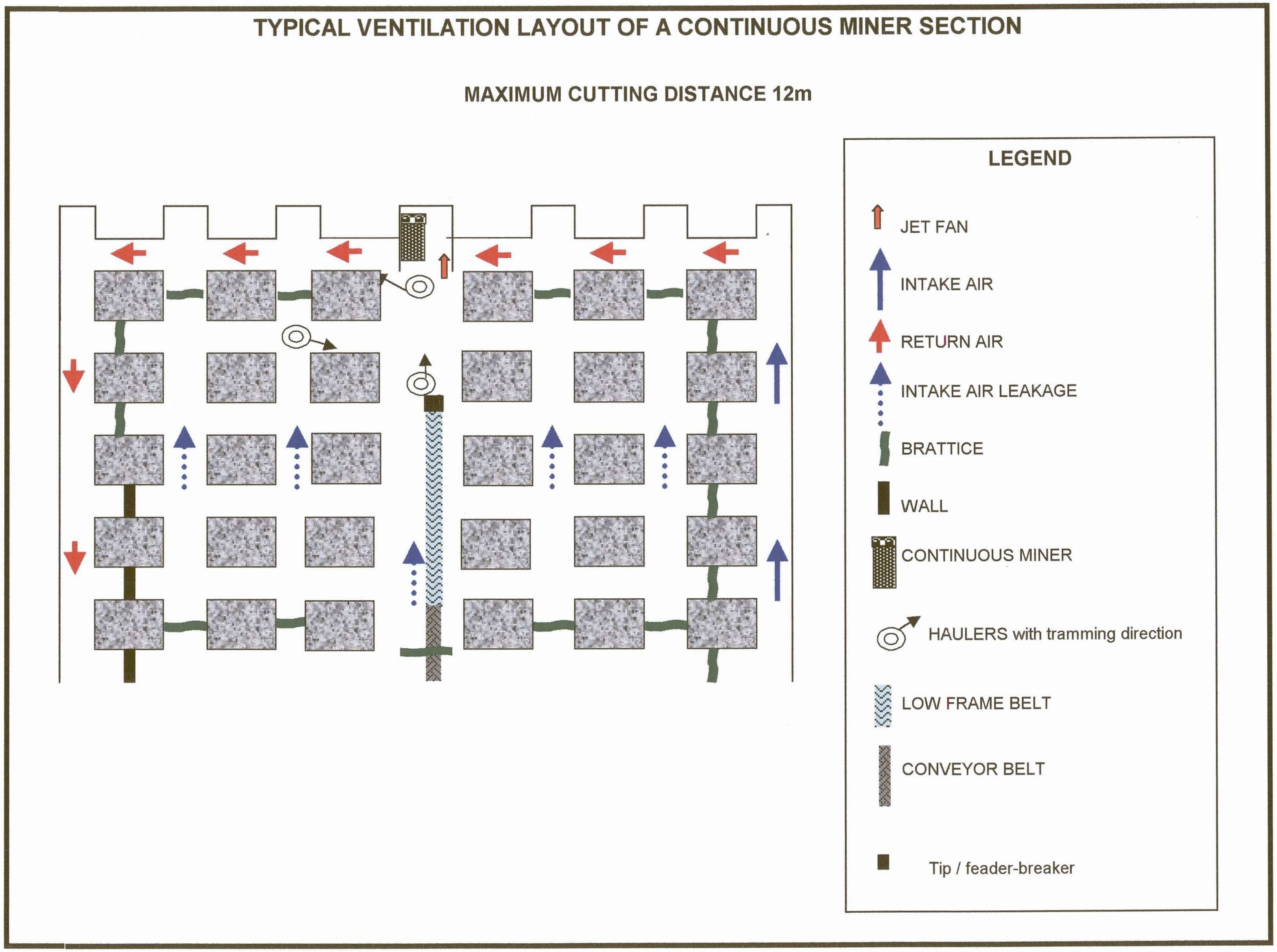


Fig. 4.2. Ventilation layout (Van Zyl, 2001).

4.2.3 Conventional Drill and Blast Section

The flexibility of the conventional drill and blast sections in negotiating geological obstacles together with the improved creation of the financially attractive coarser fraction product, is still important factors in the use of this method of mining. Unfortunately this method only works effectively for seam heights above 1,6m as production rates decrease exponentially with the reduction of heights. In the current thin seam area the parting is included in the mining to provide the necessary height for this section. The yield decrease is significant by including this parting.

In this section an amount of 11 roadways are been mined with pillars width 6.8m and bords 6.8m, giving an extraction in the region of 70 to 75%.This section is equipped with the following:

- 1 x Coal loader.
- 1 x Roofbolter.
- 1 x Feeder- Breaker.
- 1 x Coal Cutter.
- 2 x Joy Shuttle Cars.
- 2 x Electric Coal Drills.
- 1 x Mobile 750 KVA transformer.

The manpower is:

- 1 x Miner
- 2 x Electric Coal Drill operators and 2 x assistants.
- 2 x Drill assistants (jackhammer).
- 1 x Coal Cutter operator and assistant.
- 1 x Coal Loader operator and assistant.
- 2 x Shuttle Cars drivers.
- 1 x Feeder-Breaker operator.
- 1 x Roofbolter operator and assistant.

- 5 x General labourers.

The total number of persons per shift is 21.

4.2.4 Stone Work Team

The mine has a dedicated stonework team, whose duties include:

a.) Mining through dykes exposed by coal winning.

b.) Brushing and supporting of the roof to 1.8m heights in roadways and belt roads.

c.) Brushing and supporting of the roof designated for ventilation and mine infrastructure e.g. air crossings.

d.) Installation of superior and additional support in areas where poor roof conditions prevail.

The stonework team is operating on a single shift but can be changed to a double shift when conditions dictate. Additional contractors were introduced to help with specialized support and to assist where additional support was required.

The stonework team is equipped with the following:

- 1x Self propeller roofbolter
- 1x Mobile 500 cpm compressor
- 3x Pneumatic drills (jackhammer) and air legs
- 1x Mobile switch trailer with flame proof gate end boxes
- 1x Mobile 500 kVA transformer
- 1x Portable explosives magazine

The manpower is:

- 1x Miner
- 2x Drill operators (jackhammer)
- 2x Drill assistants (jackhammer)
- 2x General labourers

The total number of persons per shift is 7.

All external waste mined, such as roof rock, dyke material and burnt coal is stowed underground in such a manner so as to minimize the risk of spontaneous combustion.

4.3 Risks.

4.3.1 Geological.

a.) In-seam partings. These partings result in a drop of yield and cause materials handling problems, which in turn adds to the cost of maintenance on equipment and conveyor belts.

b.) Roof slumping and compaction structures. Sudden changes in roof heights lead to difficult mining conditions. This so-called "pinching" of seam heights creates difficult working conditions for hauler- and shuttle car operators.

c.) Unexpected laminations in the roof. Thin laminations of silty material in the roof lead to dangerous conditions as delamination of the roof can result in rock falls, which can cause injury and fatalities.

d.) Changes in coal quality. The drop in product yield directly results in an increase in production costs. Unexpected quality changes might result in dissatisfied customers, which can result in the cancellation of contracts. The highest risk in this category is the possibility of high sulphur values.

e.) Floor rolls. These occurrences are as unpredictable as their extent is limited. Floor rolls caused dangerous conditions during machine movements. The continuous miner had difficulty moving over these rolls as the length of this machine caused the rear end to "hang up" on the roof as the front-end traverse down the slope of a roll.

f.) Dykes. Dolerite intrusions normally cause a section to come to a halt as the roads need to be developed through the dyke by the stone crew. Dykes result in burned or devolatilised coal, which can not be sold. Dykes form gas traps for methane and often have bad roof conditions associated with them.

4.3.2 Production.

Many of the production problems encountered at Dorstfontein mine were associated with geological features. Unexpected thin seam conditions (1.5m and thinner) resulted in a sudden halt of production in many sections. For a continuous miner section a serious geological thread is the appearance of an in-seam parting. Production losses may be as much as 50% when these features occur in the CM-section. For the conventional sections the most deleterious conditions are sudden drops in seam height due to roof slumping. The fixed set of mining equipment in a conventional section makes it almost impossible to negotiate this kind of problem. Production losses may be as much as 70% of normal production as roof stripping needs to be done for the haulers to move around. A loss of production means less product coal to sell which results in a loss of income. Production losses also mean an increased unit cost, as the fixed cost component remains constant.

4.3.3 Safety.

Many of the geological risks may result in a serious injury or fatality. Currently Dorstfontein mine has a very good safety record with almost 2000 fatality free shifts (will be achieved June 2003) and a lost time injury frequency rate (LTIFR) below 2. This has only been achieved by the continuous awareness of the workers of the difficult mining conditions encountered so far. Another factor contributing to the good safety record is the fact that during most of the mine's life it has been producing in the higher seam areas (1.5 - 2.5m). The occasional, unpredicted and localized geological problems were negotiated in a safe and efficient manner. The largest part of the remaining reserve will be in similar or even better conditions. The risks, which may possibly result in injury or fatality, have been identified and are well managed by a dedicated management team and workforce.

4.3.4 Costs.

High costs are a fact of mining but sudden increases in working cost is a huge risk for a small operation. Unexpected changes in geological conditions, the drop of production rates, fluctuations of the exchange rate and increased cost of maintenance due to damage caused by mining conditions, may unexpectedly increase working costs to a point where it may spiral out of control. The management of costs and the above mentioned risks is of the utmost importance to keep the mine running for its planned life. All above mentioned risks contributed to the cost of mining at Dorstfontein and the management of these risks has so far lead to a productive and economically successful mine. In Fig. 4.3 it can be seen how the run of mine cost escalates with a decrease in production.

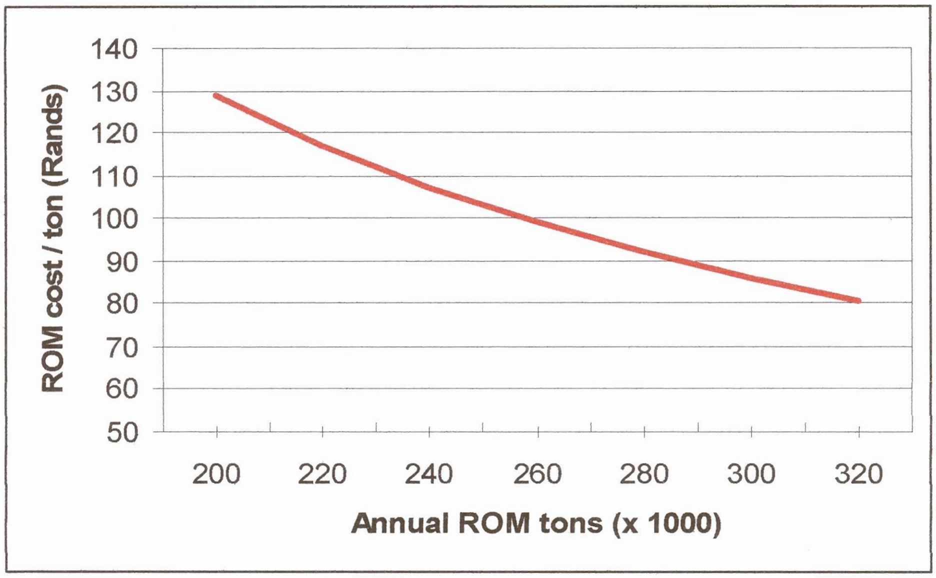


Fig. 4.3. Cost : Volume relationship.

The greatest risk may be regarded as managerial risks where people become lax and relaxed due to a good safety record. This may very easily develop in to an unconcerned attitude towards risks. It is difficult and challenging to keep people motivated and vigilant towards the risks and dangers involved in coal mining.

CHAPTER 5: THIN SEAM RESOURCES.

5.1. International.

They are only 3 main areas in the world where significant quantities of thin seam coal are mined namely the U.S.A., Europe and the former U.S.S.R. (Clarke et al., 1982). Of these the former U.S.S.R. produced more than 75 percent of all thin seam coal worldwide and that mainly from the Ukraine. In Europe the mining techniques have been developed for deep mining conditions while in the U.S.A. shallower and flatter seams have allowed for room and pillar methods.

The largest producers of thin seam coal are the former U.S.S.R. and the U.S.A. Other countries produce smaller tonnages but still have significant output. Countries like Spain, the U.K., Czechoslovakia, Poland and Colombia produced significant quantities of coal from thin seams. In the late 1980s and during the 1990s most of the U.K. mines were closed principally because of economic reasons following decreases in state subsidy. Some of the old mines like Trimdon Colliery (1840 – 1925) worked seams with heights of 3 feet 8 inches (1.11m) at depths of 195m using drill and blast methods. Very small tonnages are still produced in the U.K. and this country has become a net importer of coal.

Table 4. Thin seam definition in various countries (Clarke et al., 1982).

COUNTRY	m	in
Belgium, U.S.A.	0.60	24
Germany	0.70	28
U.K.	0.91	36
France, Poland, Ukraine, Czechoslovakia	1.00	39
Former U.S.S.R.	1.20	48
Bulgaria	1.30	51

Table 5. Thin seam output as percentage of total coal output. (Clarke et al., 1982)

COUNTRY	% OF RESERVES
Spain	70.0
Colombia	50.0
Former U. S. S. R.	47.6
Belgium	38.4
Czechoslovakia	30.0
U.S.A.	10.8
France	7.8
U.K.	7.0
Poland	2.0
Germany	1.1

Since thin seam mining has become unfavourable due to its low production rate and output, these figures could have changed subsequently, as some countries have closed their thin seam mines. Countries like France, Belgium and Germany produced significant tonnages from thin seams in the 1960s but have ceased production from these mines. In Annexure 1 the various thin seam reserves are described.

No information about thin seam mining in China could be obtained. It is not even known if they do mine thin seams, as information coming from that country is either non-existent or not translated. It is well known that China has almost tripled its coal production and has become one of the major coal producing countries.

In Australia collieries are focused on high output from 30m seams and consist mainly of opencast mines. Some information about Australian thin seam mining was obtained from the Internet (http://fueltaxinquiry.treasury.gov.au/content/Submissions/Industry/NewHope_294.asp). This article was about a proposal to introduce tax incentives and a fuel rebate at the Jeebropilly, New Oakleigh and New Acland Mines in the Walloon Coal Measures of the Surat-Moreton Basin. It appears that multiple thin seams are being mined in open pits and that the greatest cost is diesel for the equipment. Since no information on the Australian definition of a thin coal seam could be found and multiple publications exist about their high coal production and exports, the assumption must be made that in Australia the true thin seam coal mine does not exist.

It can safely be assumed that both China and Australia do not have major output from thin seam mines.

In the U.S.A. the following eight states contain together 86% of the total U.S.A. thin seam resources.

Table 6. Distribution of thin coal seams in the U.S.A. (Clarke et al., 1982)

STATE	% OF STATE RESOURCES
Alabama	41.0
Indiana	6.0
Kentucky	41.0
Missouri	67.0
Ohio	35.0
Pennsylvania	21.0
Virginia	52.0
West Virginia	16.0

Other states with potential thin seam mines do exist and their potential was investigated in more recent times. In Annexure 1 it can be seen that West Virginia has introduced a tax reduction and new tax formula for thin seam mines. Other states have made similar proposals to their legislators in order to keep thin seam mining and their communities alive and to promote the opening of new thin seam mines.

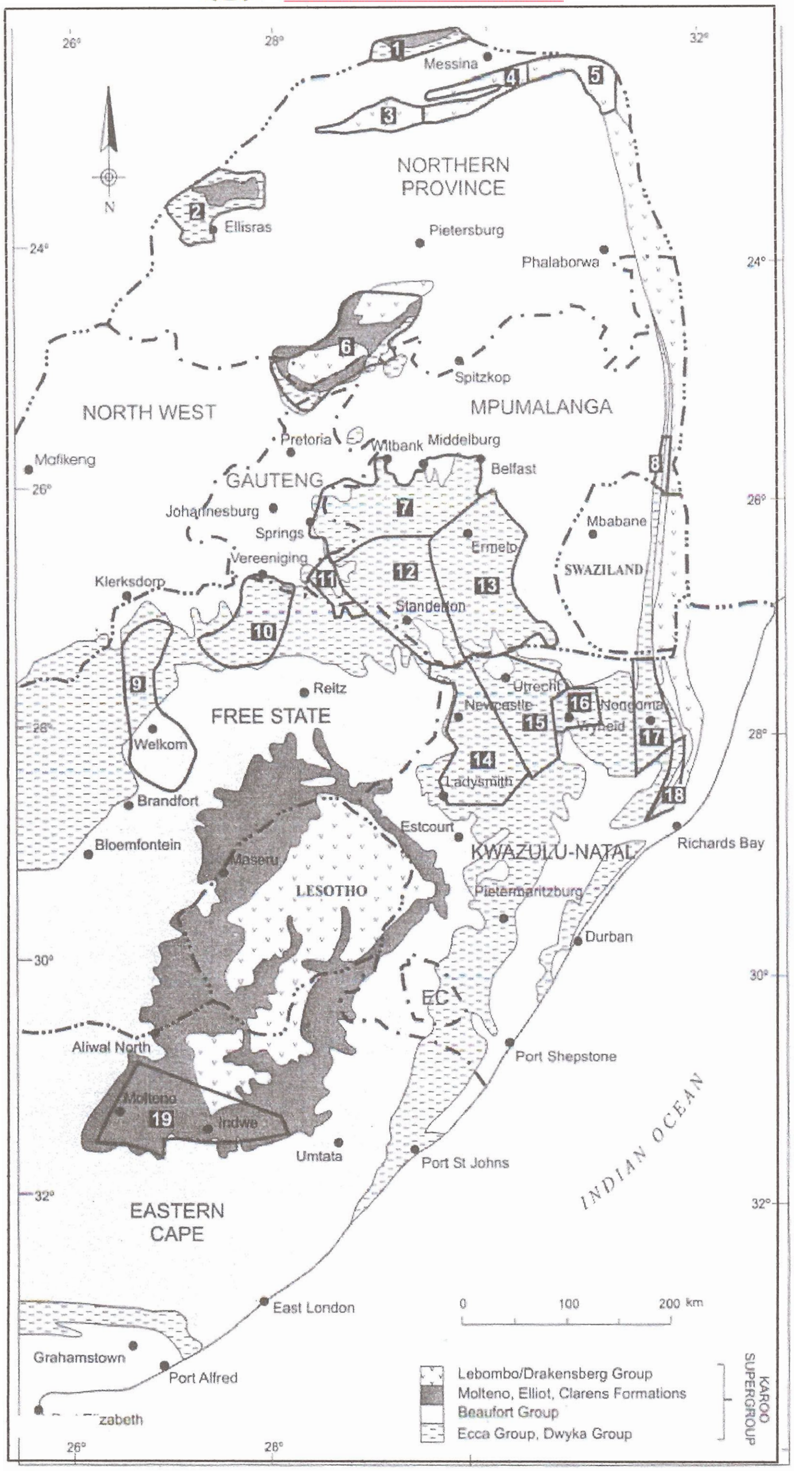
5.2. Republic of South Africa. (See fig. 5.1)

In the South African scenario most of the thin seam coal mining took place in the KwaZulu-Natal Coalfields. Some thin seam mining of the No. 5 Seam took place in the Highveld and Witbank Coalfields for example the old Blesbok, Landau, Springbok and Greenside collieries. The No. 5 seam does not fit our definition of the thin seam as the average thickness of this seam in the Highveld and Witbank regions is 1.8m (Jordaan, 1986). Even today some successful mining of the No. 5 Seam (1.5 – 1.8m thick) is taking place at Bank Colliery and with variable success at Matla Coal Mine (1.8m thick).

The two largest collieries in the Eastern Transvaal Coalfield (Greenfields, 1986), namely Usutu and Ermelo Mines, were closed due to adverse geological conditions. These two mines occasionally mined thin seams although their focus was not exclusively thin seam mining (Jacobs, 1989). At Ermelo Mines some roof brushing had to be done when 1.2 m seam thicknesses were intersected. As this mine was not equipped and focused on thin seam mining, this development was mainly done to work through thin seam areas to access thicker seams beyond. Similar conditions prevail at the currently operating Spitzkop and Strathrae collieries (Fig. 5.2) (Greenshields, 1986). Carolina Coal Company produces (drill and blast method) 11,000 ton per month from the 1.0m thick C-Seam and 16,000 tons per month from the 1.45m thick B-Seam. Eastside Colliery has similar seam heights but only produce from an open pit (Mr. J. Ackerman - Owner/operator, 2003, Pers. comm.)

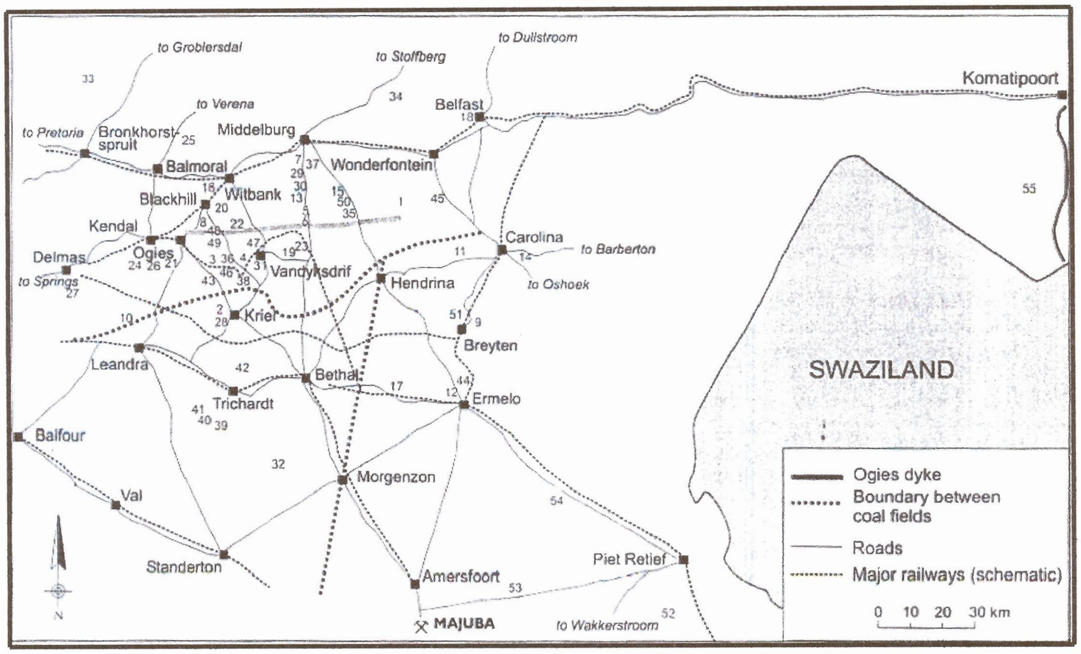
It is therefore safe to say that a very small amount of coal is produced from thin seam mining in the Highveld, Witbank and Eastern Transvaal Coalfields.

Exclusive thin seam mines were those operating in the KwaZulu-Natal Coalfields. Many of these mines are now defunct with only a handful still producing low tonnages for strategic purposes. In the past most of these mines supplied anthracite and coking coal to the then fully operational Newcastle steelworks of ISCOR and the export anthracite market. As ISCOR has closed down and scaled down many of their operations it directly affected the production of the thin seam collieries in the KwaZulu- Natal Coalfields. A downturn in the international anthracite market as well as the introduction of new metallurgical processes, such as direct reduction and briquetting in the steel industry, has obviated the need for coking coal. Small output from these collieries would not make them economically viable for the inland market only. Most of them were kept open for strategic reasons and heavily subsidized by a captured market (the then government-owned ISCOR). ISCOR found alternative sources for coking coal, Groottegeluk at Ellisras and Tsikondeni near Mussina, and could therefore close down not their Natal mines.



The distribution of coal fields in the five relevant provinces. 1. Tuli, 2. Ellisras, 3. Mopane, 4. Tshipise, 5. Pafuri, 6. Springbok Flats, 7. Witbank, 8. Kangwane, 9. Free State, 10. Vereeniging–Sasolburg, 11. South Rand, 12. Highveld, 13. Ermelo (formerly Eastern Transvaal), 14. Klip River, 15. Utrecht, 16. Vryheid, 17. Nongoma, 18. Somkele, 19. Molteno–Indwe.

Fig. 5.1. Coalfields of South Africa (Snyman, 1998)



Collieries in Mpumalanga. 1. Anglo Power (Arnot), 2. Anglo Power (Kriel), 3. Arthur Taylor, 4. ATCOM, 5. Bank 2, 6. Bank 5, 7. Blackwattle, 8. Boschmans, 9. Bothasrust, 10. Delmas, 11. Dover, 12. Driehoek (Wesselton), 13. Duvha, 14. Eastside, 15. Eikeboom, 16. Elandsfontein, 17. Ermelo, 18. Glisa, 19. Goedehoop, 20. Greenside, 21. Khutala, 22. Kleinkopje, 23. Koornfontein, 24. Lakeside, 25. Landau (Kromdraai), 26. Leeuwfontein, 27. Leeuwpan, 28. Matla, 29. Mavela, 30. Middelburg, 31. New Clydesdale, 32. New Denmark, 33. Northfield, 34. Olifantslaagte. 35. Optimum, 36. Phoenix, 37. Polmaise, 38. Rietspruit, 39. Secunda: Bosjesspruit, 40. Secunda: Brandspruit, 41. Secunda: Middelbult, 42. Secunda: Syferfontein, 43. South Witbank, 44. Spitzkop, 45. Strathrae, 46. Tavistock, 47. Van Dyks Drift, 48. Waterpan, 49. Witbank Consolidated, 50. Woestalleen (Noodhulp Section), 51. Consbrey Dump, 52. Protea, 53. Mpisi, 54. TBS, 55. Nkomati Anthracite (after Smith and Whittaker 1986; Jordaan 1986; Schoeman and Boshoff 1996)

Fig. 5.2 Collieries in Mpumalanga. (Snyman, 1998)

Currently there are only a handful of operating collieries of which Zululand Anthracite Colliery (ZAC) is the largest (Fig. 5.3) producing 60 000 tons per month from various drill and blast as well as continuous miner sections. In recent years many of these mines have changed hands from larger to smaller companies, for example the selling of Utrecht Colliery by Ingwe to Kangra. Some of these smaller operators try to keep these mines open by opencasting the sub-outcrop coal. Most of the old mines were only successful by mining the thicker seams above the thin Gus and Dundas Seams. These thin seams range in thickness from 0.7 to 3.3m, with an average of 1.0m for the Gus and 0,7m for the Dundas Seam (De Jager, 1976). The main mining method was bord and pillar with a very few mines using longwall methods. Seam thickness, dolerite intrusions and seam dips made mining very difficult and expensive.

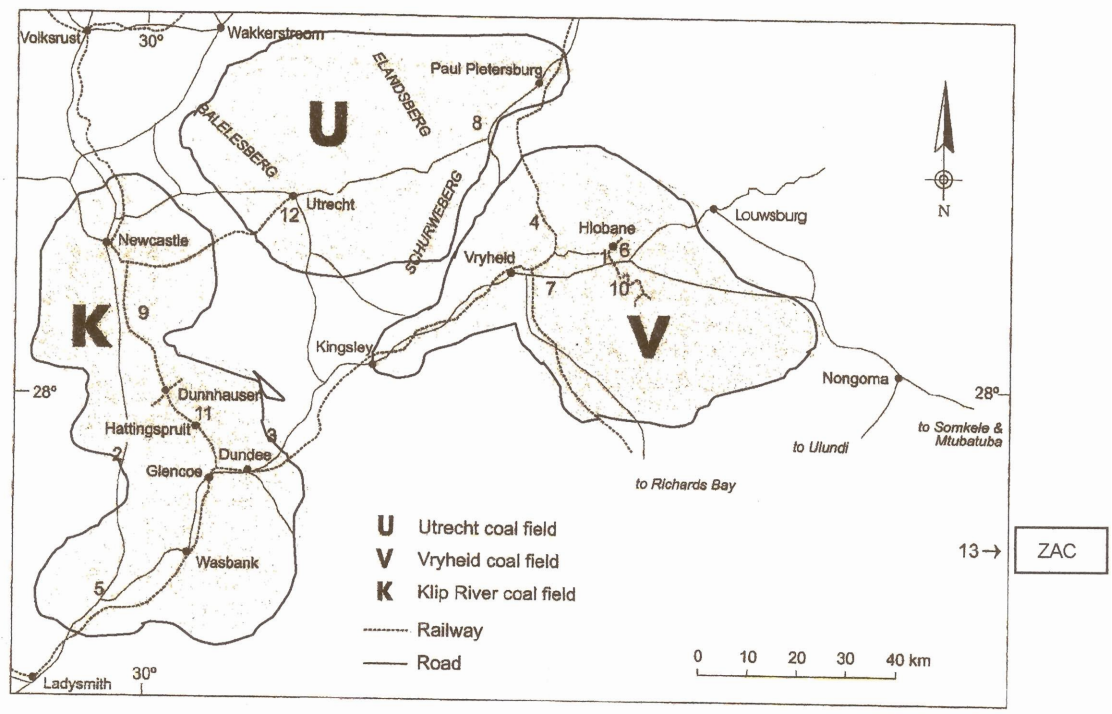


Fig. – Major coal fields (U = Utrecht, V = Vryheid, K = Klip River coal field) and more important producing collieries in KwaZulu-Natal. 1. Alpha Anthracite; 2. Durban Navigation; 3. Gladstone; 4. Heritage; 5. High Carbon (Newcastle–Platberg Dump); 6. Hlobane; 7. Jacksons Anthracite (Mpofini); 8. Long Ridge; 9. Macalman; 10. Rusplaas; 11. Springlake; 12. Umgala; 13. Zululand Anthracite (after Spurr et al. *1986; Bell and Spurr 1986a, b; Schoeman and Boshoff 1996).*

Fig. 5.3. Coalfields of KwaZulu-Natal (Snyman, 1998).

Table 7. Some defunct collieries in KwaZulu-Natal (Spurr et al., 1986)

Colliery	Coalfield	Date closed
Balgray	Utrecht	?
Boemendal Consolidated	Utrecht	1966
Constantia Coal Mine	Vryheid	?
Dumbe	Utrecht	1938
Dumbe	Utrecht	1975
Elandsberg Anthracite	Utrecht	1966
Enyati	Vryheid	1971
Hlobane	Vryheid	Late 1980s
Kilbarchan	Kliprivier	1990s
Kempslust	Utrecht	?
Longridge	Utrecht	Mid 1990s
Makateeskop	Utrecht	?
Mooihoek	Utrecht	1966
Mooiklip	Vryheid	?
Pivaan	Utrecht	1979
Vryheid Coronation	Vryheid	Late 1980s
Vryheid Coke	Vryheid	?
Vryheid Export	Vryheid	?
Weltevreden Anthracite	Vryheid	?

The above listed mines are not necessarily exclusive thin seam mines but most were a combination of thin and thick seams extracted simultaneously. Spurr et al., (1986) and Bell and Spurr, (1986) listed many other mines of which many were not exclusively thin seam mines. At Newcastle, in the Kliprivier Coalfield, the main seam mined was the Upper Seam. The Middle Seam is the thin seam, 0.94m thick, but was not always mined. The main coal produced from this area was anthracite and currently some small operators still reclaim dumps and mine small pits e.g. AfriOre at Springlake.

Table 8. Active collieries in KwaZulu-Natal (Pinheiro, 1999)

Colliery	Coalfield
CBR Mining	Kliprivier
Duiker Heritage	Vryheid
Duiker Nyembe	Vryheid
Durban Navigation	Kliprivier
Springlake	Kliprivier
Umgala	Utrecht
Welgedacht	Utrecht
Zululand Anthracite	Somkele

There are currently only 2 operating mines in the Vryheid area but in its prime this area had a huge output of coking coal and anthracite for the export market. In the whole KwaZulu–Natal Coalfield there are still some substantial thin seam resources left but no market exists for these costly to mine thin seams. The total indicated resource for all coal types in all the coal seams in the KwaZulu-Natal coalfields is 3,035 Mt in situ of which unknown proportions are thin seams (Barker, 1999).

CHAPTER 6: RISKS ASSOCIATED WITH THIN SEAM MINING.

At Dorstfontein Mine all of the mining has taken place in seam heights exceeding 1.5m. The risks and associated mining problems identified during the life of the mine were discussed in Chapter 4 and differ from that identified by Clarke et al. (1982) for very thin seam mining. This chapter discusses the risks as well as the health and safety issues associated with thin seam mining (at Dorstfontein below 1.4m heights). Although some of these risks may be more applicable to hand-got coaling, they may not be omitted as although continuous miners replaced the pick and shovel, people still work and move around in these thin seam CM-sections.

6.1. Geological.

a.) Seam heights. One of the greatest risks in thin seam coal mining is unexpected decreases in the already thin seam height. These changes are unpredictable and may be attributed to various factors for example floor rolls and slumping structures in the roof. These kind of geological features could bring a section to a standstill.

b.) Quality changes. In Chapter 3 it is apparent that the coal quality and product yield of the thin seam areas could be extremely good. Unexpected changes in product yield may increase costs, and might terminate this difficult way of mining. The sulphur content is one of the most important quality parameters that must be monitored carefully. Coal analysis has showed that in some areas the sulphur tends to be high due to free pyrite in the coal seam. An increase in the sulphur content, outside the product specifications, would create a problem on the marketing side.

c.) In-seam partings. Throughout all the exploration programmes there were few in-seam partings intersected. This does not exclude the possibility that extra thin shale bands and flood sheets may occur. This will reduce the yields and create problems for continuous miner production.

d.) Change of parting lithology. The seam-split parting will form the roof of the thin seam section and exploration has shown that this parting has an upwards-coarsening sequence with a lower section of interlaminated sandstone and siltstone. This parting can be supported, as tests have shown, as long as it stays upwards coarsening. Changes in the laminations of this parting may render it a dangerous roof and create production- and yield problems.

e.) Water. Excessive discharge of water from either the coal seam, overlying roof strata or dyke developments would create problems for people working in such conditions. The thin seam does not allow ease of movement and in the event of excess water people would get wet which will lead to health problems. Excess water would also enter machinery and motors and result in breakdowns. Slippery working conditions would lead to injuries.

f.) Unpredicted dykes. Most of the dykes in the thin seam area have been predicted and some of them were intersected during the South Main development. In the unlikely event that some unpredicted dykes do occur it will create a serious problem for production and could result in adverse roof conditions. Some dykes discharge a great amount of water, which could lead to mining problems and health and safety issues.

6.2. Mining Accidents.

An accident has been defined as "any unplanned exchange of energy which degrades the system in which it occurs". The effect of an accident on mine personnel is the most noticeable and the recording of such injuries provides the bulk of the statistical information on accidents. In most countries this wider concept of an accident is reflected in mining legislation that demands more records and reporting of certain dangerous occurrences that may or may not cause personal injury. The major factor in determining whether an accident is recorded and reported is the nature of the injury sustained. That is the effect in terms of disability and the time the injury prevented the person from working (Clarke et al., 1982).

In the United States a relatively low number of incidents were reported in thin seam coal mining. There was no significant variation of the frequency of fatalities between thick and thin seam mining. The average rate for accidents was higher for thin seams than for medium to thick seams. The frequency rate of disabling injuries was approximately 100 times higher than the fatality rate. It was found that the accident rate was significantly higher in the thin seams than in the thicker or medium seam mines. The increase in the level of hazards may be explained by the decrease in lighting and comfort in thin seam working conditions. In the case of injuries from falls of roof, it was suggested that it was more difficult to avoid an imminent fall in the more cramped conditions of the thin seam. Another possible explanation was the lack of protective cabs and canopies on thin seam face equipment (Clarke et al., 1982).

In contrast to the disabling accidents, the reverse trend was apparent for non-disabling accidents. The frequency rate of non-disabling accidents was lower for thin seam than for thicker seam mines. This can be explained by the fact that thin seam coal accidents are likely to be more serious when they occur since it is harder to get away from or to correct a potential accident situation owing to the confined space. It was found from analysis of sub categories of fall of roof that higher proportions of accidents in thin seams occur during installation of timber or other support, than in thicker seams. The difficulty of installing roofbolts was identified and the protrusion of such support resulted in obstructed travel ways, which could lead to head and back injuries during machine movement (Clarke et al., 1982).

It was found that at mines with low accident rates the morale of the people was good, the geological conditions in terms of strong roofs and floors were good and that increased mechanization has led to fewer injuries. The most common single injury on the thin seam mines was that of a sprained back (Clarke et al., 1982).

In the British collieries there was a steady decrease of the accident level as miners became more safety conscious. The fatality rates have decreased from 4 per 1000 men to 0.25 per 1000 men. The most common injuries were from falls of roof and machinery and haulage movement. The fall of roof rates for the thin seam in the U.K. mines are much higher than for all other mines. This may be attributed to the lack of mobility in the thin seam sections and the support tended to be of a lighter construction to maximize available traveling and working space. A relatively small proportion of accidents from machinery and haulage movement occurs at the face. Most accidents in this category appear in the load-out and out-bye areas. The rate in all haulage and transport accidents is higher for thin seam mines than for thicker seams. In the U.K. mines accidents of this nature contributes to over one third of all serious accidents (Clarke et al., 1982).

In the U.K. mines serious accidents from the use of hand tools in thin seam areas are rare. Stumbling and falling accidents account for the highest number of total accidents in a single category. This high rate is reflected in the serious accident category and shows a higher rate for thin seam than for thicker seam. The rate for serious accidents resulting from slip or falls is much higher for thin seams than for all other mines (Clarke et al., 1982).

In the former U.S.S.R. few statistics exist about their thin seam mining operations. It is noted however that augering operations in the thin seam mines have had no accidents. The conclusion can be drawn that remote operation was much safer than any other mining method. No certain conclusions can be made about any of the former U.S.S.R. mining operations (Clarke et al., 1982).

In the Republic of South African most of the thin seam coal mining was done in Kwa-Zulu Natal. The accident rate in the thicker seam levels is lower than in the thin seam levels, except where the No. 5 (not a thin seam) seam has been worked in the old Transvaal province (now Mpumalanga). Accidents from roof falls were more common in these operations due to the weaker mudstone roofs. Haulage and transport accident frequencies were also high due to the use of track equipment and tubs in thin seam mines (Clarke et al., 1982).

In Colombia most of the coal production is from thin seam mines. The collection of accident statistics is not reliable as there is no legal obligation to report and record accidents. The reportedly high accident rate in this country can be attributed to the lack of controls and standards and not so much to thin seam conditions (Clarke et al., 1982).

To conclude: the U.S.A. experience indicates that the accident frequency rate per million man-hours of exposure in thin seams is higher than in medium or thick seam mines. If the accident frequency rate is calculated on the basis of accidents per million tons mined, the thin seam rates are substantially higher than that for medium or thick seams due to the lower productivity in thin seams. In the U.S.A. the occurrence of hazards, involving mobile machinery in thin seams, are partly due to the difficulty of working by means of bord and pillar methods which involves frequent moving of large items of machinery in confined spaces. The difficulty in supporting the roof is another contributory factor. The U.K. and the former U.S.S.R. trials with remote mining systems have indicated that men may be removed from the face with the expected improvement in safety.

6.3. Health and Safety.

Hazards that result in physical injuries are easier to identify than those that affect the health of workers. The reason for this is that the injury normally occurs as a result of some violent event and the object that cause the accident is directly identified. The detrimental effect on health takes place over a period of time and until some loss or impairment of body function has occurred, the employee may not be aware that the process is taking place. The more obvious hazard to health is that affecting the respiratory system, named pneumoconiosis. In thin seams another health problem is beat diseases, which are caused by working and traveling in unnatural positions. Beat diseases are more common in ultra thin seams where miners work on their knees and elbows. These diseases are described as sores, abscesses and swellings due to constant beating of limbs against the roof and floor. Correctly fitting and comfortable knee and elbow pads are important (Clarke et al., 1982). This condition is less likely to develop where remote control equipment is used and the operator sits while working, but may be common amongst the roof support crew and cable handlers.

Other environmentally related health problems are those associated with working in close contact with water and oil, the danger to eyes from particles picked up by high air velocities, noise and poor illumination (Clarke et al., 1982).

Hazards to respiratory health in coal mining come mainly from inhalation of respirable dust particles. In general the relationship between health and dust apply to all seam conditions. The problem may be more acute in thin seams owing to higher velocities of air needed to supply the right velocities to the coalface. In the U.S.A. some thin seam mines required dilution of methane and the only way to get enough volume for the dilution was to increase the velocity. High velocities may produce a counter effect by causing dust pickup. Velocities above 2 m/s cause appreciable pickup of dry dust but, when the dust is wet, velocities of above 4 m/s can be tolerated. Particle size also affects the pickup of dust. Items of equipment in roadways can cause restrictions in cross sectional areas and result in funneling of air with a resultant increase velocity at the restricted point. In the vicinity of any cutting machine at the coalface, the area is reduced causing funneling of the air with an increase in velocity at that point. It is particularly important in thin seam coal mining that adequate dust suppression equipment be used (Clarke et al., 1982).

In thick and medium seam collieries, water on the floor is merely a problem that should be dealt with. In thin seams however the problem is more severe when miners become sodden from crawling and sitting on wet floors. The use of hydraulic fluids in equipment and machinery causes skin diseases such as dermatitis. Spillage must be kept to a minimum and protective gloves must be worn at all times. Complaints such as colds, influenza and rheumatism may develop where the ventilating air is cold and the wet miners move in and out of this cold air (Clarke et al., 1982).

The amount of noise in thin seam working conditions is much more pronounced than in larger working spaces. It is therefore imperative that all workers wear hearing protection at all times. The advantages of remote control operations are obvious as in the case of noise as the operator is physically removed from the source of this noise (Clarke et al., 1982).

6.4. Production rate and costs.

In thin seam mining a greater area of ground has to be mined in order to extract an equivalent tonnage to that from thicker seams. Many of the tasks that have to be performed in underground mines are related to linear advance and so for a given output they must be carried out more frequently in thin seam mining. Extensions of rail track, conveyor belts, water- and power lines can reduce the productivity in thin seam sections. Other tasks such as sweeping and stone dusting needs to be done and are directly related to area extracted and not tonnage mined. These factors reduce productivity in thin seam mining. In the late 1960s many mines still operated at 10 tons per manshift. This production output has increased with the introduction of longwall mining methods and bigger and more powerful continuous miners. The greatest risk to the production rate is the lack of availability of mining equipment, adverse geological conditions, high equipment maintenance and downtime on the transport systems (Clarke et al., 1982).

The direct result of a low productivity is the escalation of cost. Although the fixed costs cannot be changed, its component in the Rand / ton cost of the R.O.M. tons, will increase. With the high output this component becomes less pronounced in the Rand / ton costs of the R.O.M. tons e.g. if the fixed component equal R 200 000.00 per month and the section produces 20,000 tons per month, the R.O.M. fixed cost is R 10.00 / ton. If the section only produces 10,000 tons for that month, the R.O.M. fixed costs will be R 20.00 / ton. Likewise the variable cost will be influenced by additional maintenance and repair costs during adverse mining conditions. It is common for collieries to have a high fixed cost and relatively small proportion of variable cost. This feature of a mine makes it imperative that output targets are achieved. Nearly all the profits come from marginal tonnage i.e. tonnage mined over and above the base tonnage.

Another risk factor that seriously affects the cost of thin seam mining is the yield. By either cutting the floor or the roof the yield from the thin seam sections would be reduced which in turn would increase the costs. Therefore it is imperative that mining horizons being maintained to produce is much coal as possible and exclude contaminants.

CHAPTER 7: CURRENT THIN SEAM MINING TRIAL.

7.1. Continuous Miner and Battery Haulers.

In 2002 the German company Maschinen- und Bohrgeräte Fabrik GmbH designed a thin seam continuous miner that is capable of cutting as low as 1.0m. It is called the Wirth Paurat H4.30. (For specifications see Annexure 3). The main purpose of this design was to directly compete with the American company, Joy Mining Machinery (a subsidiary of Joy Global Inc. Company), which has a huge market share in the U.S.A. coal mining industry and in the R.S.A. and who also specializes in thin seam mining equipment (pers. comm.). T.C.S.A. management heard about the new development and enquired about the possibility to test this machine at Dorstfontein Mine and compare it to the current Joy 12HM15 on the mine. It was agreed to, with the arrangement that Dorstfontein uses and tests the machine for 1 year at a fixed rent after which T.C.S.A. has the option to buy the machine at a reduced price. The Wirth arrived at the mine in middle December 2002 and moved into a section where the seam height is 1.6m. For the coal haulage there are 2 Stamler BH10 thin seam battery haulers (For specifications see Annexure 4).

The Wirth is equipped with a Debbex/Kennametal double rotating drum, which has been designed to be able to cut thin stone bands. The configuration of the cutterhead is such that a fair amount of the large coal fraction is produced and the fine fractions kept to a minimum.

Initially there were problems with the power supply and software of the Wirth as this machine was built and assembled in Germany and needed to be adapted for South African conditions. A few minor design errors also needed to be corrected on mine to suit our specific conditions. Once the Wirth was in operations it was clear that this machine is well constructed and built and should easily cut in-seam partings and even be able to pull down the seam-split parting in areas where roof brushing is necessary. Presently the parting is being blasted down by drilling holes into the upper coal seam as there exist the potential to damage the machine. Further problems needed to be sorted out during the following few months in order to achieve full production. During March 2003 the standing time became less and availability started to increase. The increased availability has led to another problem regarding the availability of the Stamler BH10 thin seam battery haulers. The Wirth machine cuts too fast for the 2 battery haulers and has to wait before it can discharge more coal from its bin. It became apparent that there is a need for another thin seam battery hauler.

The installation of roofbolts to support the parting is quick and no delay times have been experienced during their installation.

The Wirth has a cutting range between 1,0 and 2.8 m but will spent most of the trial time cutting between 1.5 and 1.6m. The maximum allowed cutting depth is 12m, for safety reasons, after which the parting needs to be supported before the machine can cut that heading again. Roof brushing is currently been done only in the combined travel and belt road, while full support of the parting is done in all the other roads. The planned production rate is 1250 tons per day for the first year after which production will be increased to 1500 tons per day for six years and then again reduced to 1250 tons per day for the last three years. This gives an average production rate of 1400 tons per day for ten years. The lower production rate in the first year is to allow time for all the problems with the new machine to be solved while the lower production in the last three years is to allow lower productivity in the very low seam areas.

The current labour complement is as follows:

- 1 x Miner
- 1 x Continuous miner operator
- 1 x Continuous miner assistant
- 2 x Hauler drivers
- 1 x Feeder-breaker overseer

- 1 x Roofbolter operator
- 1 x Roofbolter assistant
- 4 x General labourers

A total of 12 persons per shift.

7.2. Ventilation

The primary consideration when determining the ventilation requirements for thin seam mining is the provision of healthy, safe and comfortable working environment. Sufficient fresh air must be supplied to the workings to keep the concentration of methane in the general body within the legal limits which prescribes an concentration in the air below 1,4% per volume, reduce dust concentration to at least 1,0 mg/m^3 and maintain air velocities of not less than 1,0m/s along the last through road in the section. As shown in Chapter 6 equipment in roadways can cause dust pick-up and choking of the airflow to the face (Clarke et al., 1982).

Methane emission tests are done on a regular basis by taking core samples from a production face at the mine. Some of the results are tabled below.

Gas Content (m^3/ton)	Emission rate (liters/tons/min)
0.95	34.3

Normally a thin seam does not emit large quantities of methane (small volume of coal) but caution should be taken near dykes and where dolerite sills overlie coal seams to form a cap that prevent degassing of the strata during secondary coalification. This is not the case at Dorstfontein Mine and methane gas should not be a risk in the thin seam areas. The maximum allowable concentration of methane in the general body of the air in any place where people are required to work or travel is 1,4% by volume. If a limit of 0,1% is used to determine the dilution volume of air, then a safe volume of air of at least 15m^3/s will be required to ensure that the methane content of the return air volume does not exceed this 0,1%.

Calculation (Van Zyl, 2001, pers. comm.):

- m^3/ton/min = 34.3 liters / ton / min ÷ 1000 => 0.0343 m^3 / ton / min
- The CM cuts 22 tons / min => 22 x 0.0343 = 0.7546 m^3 / min of gas released during cutting.
- To get to the ventilation needed:

  0.7546 m^3 / min ÷ 60 = 0.01257667 m^3 / sec gas released.
- The dilution needed is 0.1%:

  0.01257667 m^3/sec ÷ 0.1% = 12.577 m^3/sec

  To be safe, use 15 m^3/sec

The air volume necessary to ensure healthy and safe working conditions will be more than that required to dilute the methane. The ventilating air will be distributed to at least the last two through roads from the faces at a minimum velocity of 1,0 m/s. This will require a quantity of air calculated as follows:

Average seam height: 1,3m

Bord width: 6,8m

Section air quantity = last through road area x velocity

= (6,8 x 1,3) m^2 x 1,0m/s

= 8.8 m^3/s

By allowing 40% for leakage (Van Zyl, 2001, pers. comm.) and adding 15 m^3/sec for dilution, the volume must be increased to at least 27 m^3/s. A conservative figure of 30m^3/s for the Wirth-section will be sufficient which is not much less than the 35 m^3/s currently supplied to the sections on the mine.

The current practice of erecting brick stoppings between pillars to separate the intake and return air roadways will be maintained. A jet fan capable of handling an air volume of 4m^3/s will be used to positively ventilate the advancing face in the Wirth-section. Directional water sprays in association with a dust scrubber are currently been used on the Wirth. So far it has effectively controlled the dust liberated during cutting operations. The dust scrubber installed on the Wirth currently handles an air volume of $7m^3/s$.

In order to achieve a last through road velocity of 1.0 m/s the total amount of air to the section should not be less than 30 m^3/s. The current ventilation fan on the mine is capable of supplying this additional air to an extra underground section. To channel the air to the new working area, some additional aircrossings will have to be constructed at a current cost of R 15,000 each, which have been catered for in the financial evaluation.

7.3. Rock mechanics.

7.3.1. Split-seam parting tests and results.

Detailed evaluations of the seam-split parting were done by Mike Spengler, the practicing rock engineer on the mine. These tests involved impact splitter as well as compressive strength tests. A detailed report is attached as Annexure 6. From these tests it was clear that the parting is strong and competent enough to form a safe beam to undermine. Due to safety reasons and to uphold the safety record of the mine, it was decided to construct a double safe beam by suspending the parting and upper coal from the proper roof using 1.5m full column resin bolts as well as clamping the layers together to for a strong beam (Spengler, 2002).

7.3.2. Support pattern and cutting sequence.

For the support pattern and cutting sequence that will be introduced in the thin seam areas, see Fig. 7.1 and 7.2. The generally accepted safety factor for coal mines is 1.6 where the probability of pillar failure is only 0.998468 (Van der Merwe and Madden, 2002). For shallow to medium depth mines with a very competent roof is general practice to design the bord widths to seven meters while six meters is used in mines with poor roof conditions. With this knowledge and working to a safety factor of 1.6, the pillar widths can be calculated using Salamon's Formula (Van der Merwe and Madden, 2002, p. 51). At Dorstfontein the centers (from the middle of the pillar to the middle of the bord) is 13.5m at a safety factor of 1.6.

7.4. Advantages of thin seam coal mining.

It is human nature to follow the easiest way to reach a goal. So why would companies pursue thin seam coal mining and why would Dorstfontein specifically pursue the thin seam resource? There are many reasons and some of it has been dealt with in other chapters of this treatise. The current mining trial at Dorstfontein Mine has confirmed what has been suspected for a very long time. The following reasons make it worth pursuing the thin seam coal beneath the seam split parting:

a.) During the mining trial with the Wirth machine, the yields increased significantly by 8 percentage points from about 72% to about 80% within a matter of a few days of mining below the parting. In this, one of the most important objectives of this exercise were met namely to improve the yield by undermining the seam-split parting.

b.) There is less standing time due to discharge shoot- and crusher blockages caused by the seam-split parting breaking up in huge lumps and fouling up the coal chain to the plant.

c.) One big advantage is the saving in belt replacements and maintenance. When the seam-split parting gets dumped on to the main belt going out of the mine, holes are punctured into the belt due to the weight and shape of the stone. This has been reduced, as there is less stone coming from this section.

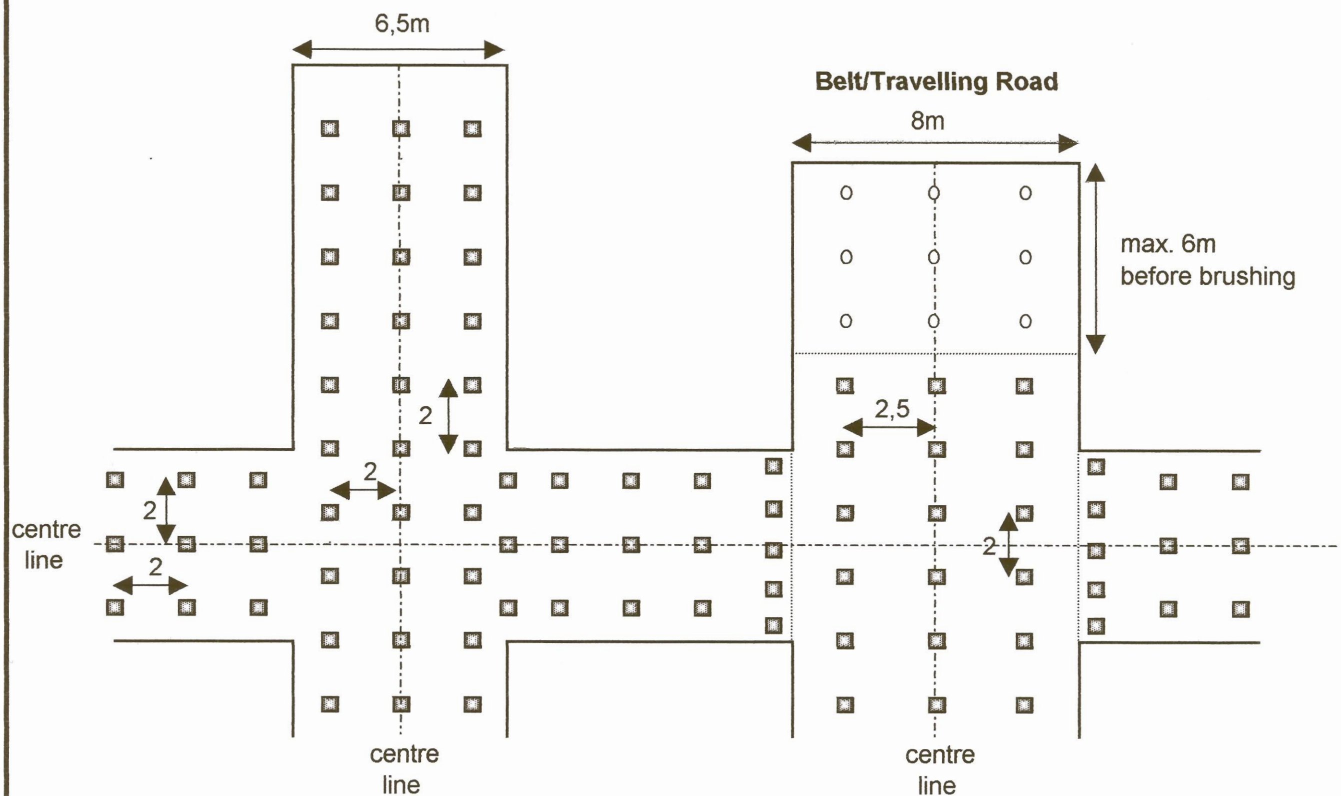
d.) In order to increase yields and prevent damage to the belts the section crew picked some of the stone by hand to be stowed underground. Fortunately no injuries occurred during the handling of the stone, but a chance existed that an accident could have occurred. This kind of injury is now less likely as the current handling of stone underground, has been reduced.

e.) The biggest and most important advantage is the extension in the life of the mine and the longer utilization of existing facilities. Further more there is the extraction of the whole No.2 Seam reserve and the additional revenue coming from this thin seam resource.

DORSTFONTEIN COAL MINE

ROOF SUPPORT: PARTING AND COAL ROOF

REFERENCE	**RED001D**
REVISION No.	
REVISION DATE	
RISK ASS. REF.	**N.A.**



Brow
Rockbolt
O Rockbolt after brushing

NOTE
All measurements are metres

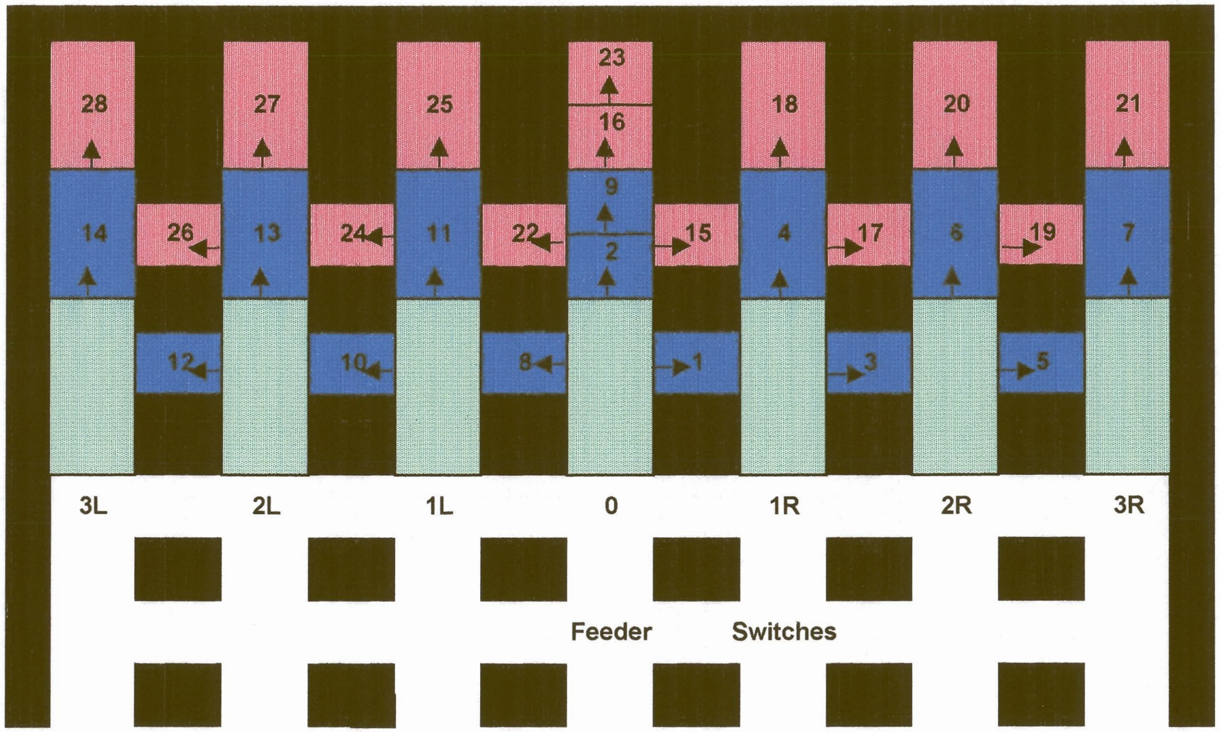
RULES
1 Support shall be installed soon as practicable after the installation position has been exposed.
2 Support installation may only advance from secure or previously supported ground.
3 Temporary support shall not be removed until the installation of the roofbolt has been completed.
4 The support will be full column resin anchored 1,2m or 1,5m x 20mm (or equivalent strength) rockbolts.
5 The length of bolt must be at least 400mm longer than the thickness of the parting and the upper coal seam.
6 Rockbolts will be torqued to 150Nm at initial installation.
7 Rockbolts will be installed normal to the roof unless otherwise instructed.
8 No person will be allowed to enter the unsupported 6m undercut. Should it be necessary to enter this area then at least 2 approved mechanical props will be installed every 2m where people are are required to work or travel. The installation of these props will be done under the direct supervision of the responsible miner.
9This standard describes the minimum support requirements and additional support must be installed where necessary.

Fig. 7.1. Roof support: parting and coal roof (Spengler, 2002)

DORSTFONTEIN COAL MINE

CUTTING SEQUENCE: 8m WIDE BRUSHED BELT ROAD

REFERENCE	MS 18b DCM
REVISION No	
REVISION DATE	
RISK ASS. REF.	N.A.



1) Cut split (1) and advance 6m in straight (2).
2) Support split (1) and brush and support straight (2).
3) Cut split (3) and advance 12m in straight (4).
4) Support split (3) and straight (4).
5) Cut split (5) and advance 12m in straight (6).
6) Support split (5) and straight (6).
7) Advance 12m in straight (7).
8) Support straight (7).
9) Continue to follow the mining sequence shown in the sketch and the support sequence described above.

Fig. 7.2. Cutting sequence: 8m wide brushed belt road. (Spengler, 2002)

CHAPTER 8: ECONOMICS OF THIN SEAM COAL MINING.

8.1. Introduction.

It is a known fact that thin seam mining can be very expensive, both in monetary value and in human life. The main decision to pursue thin coal seams is made on both strategic and financial factors. In modern society and with legislated protection the human cost will outweigh economic factors. With modern technology and the speed of modern day transport, strategic reasons do not play such a big role as in the early to middle decades of the previous century. Countries have become less dependent on coal and with open market economies and the “global village” concept any grade of coal can be sourced and delivered in very short periods of time. It seems that financial evaluations dictate decision making but risks (human lives) can terminate these same decisions. The use of computer constructed financial models makes it easy to calculate a Net Present Value (N.P.V.) and Internal Rate of Return (I.R.R.) for a specific project. It also has the added benefit that sensitivity parameters can be built in which can be changed to see the effect on the N.P.V.

8.2. Notes on the Financial Model.

It has been assumed that the thin seam area will be mined concurrently will the other sections and will be fully extracted by the time the mine closes. This exercise must not be regarded as a stand-alone evaluation for a new mine. The thin seam area will supply additional coal to the current operation and markets and may in future replace some of the current sections as these tail down towards the end of the mine’s life.

In the construction of the thin seam financial model a few assumptions were made. The following were regarded as sunk cost:

a.) Cost of lease or rights to the mine.
b.) Cost of exploration and evaluation.
c.) Cost of establishing the surface infrastructure.

d.) Cost of establishing and developing the underground facilities. The thin seam resource forms the northern boundaries of the current southern mineable reserve, called South Main (see fig. 3.14). All current and future mining in this area will be done up to where the seam thins down to below 1.5m. Extending these mining panels into the thin seam resource should not cost additional money and may not need any development through barren grounds. Some development had taken place to reach South Main (called the "Neck Development", after the thin and narrow area that needed to be developed, see Fig. 3.14) and an established main conveyor belt, ventilation road and travel road were established to connect South Main with the northern part of the reserve.

e.) Cost of the washing plant.

f.) Some costs already incurred and accounted for during previous mining to develop the South Main Area, for instance the 2 thin seam battery haulers, roof bolters and belting infrastructure.

g.) All yields quoted are theoretical yields but in the financial model a plant factor has been used in order to get to a practical yield. All financial calculations are based on the practical yield.

The financial model was constructed for a ten year period of thin seam mining. This period coincides with the closing of the mine in the year 2013 and the introduction of the thin seam must be done sooner than later as only one operating section on the mine will not be feasible. An extraction rate of 70%, geological loss of 10% and a mining loss of 10% were factored in to reduce the 7,06mil. in situ tons to 3,56 mil. R.O.M. tons. The production rate was calculated by using current knowledge gained from current mining with the Joy 12HM15 and the Wirth trial. Zwaigin's method of production rate calculations (Class notes, 2002) is inappropriate for this exercise as his formula provides for a multiple section mining operation. Using his formula (tons/annum = 390 x (in situ tons)$^{0.5}$) will yield a tonnage of 4100 tons per day, which is about three times more than practical for a single thin seam section.

The first year's production rate will be at 1,250 tons per day R.O.M. which is 50% of the current Joy production in a 1.8 to 2.0m thick seam. This tonnage should add up to 313,750 tons per year. The production rate will increase to 1,500 R.O.M. tons per day, or 376,500 tons per year, for six years. This increase is due to the learning experience during the first year of production. The production will fall back to 1,250 R.O.M. tons per day for the last three years due to the very low area to be mined at the end of the mine's life. These production rates will result in the extraction of 99.55% of the potential R.O.M. thin seam resource.

The main capital equipment is the full cost of the Wirth machine at R 15,000,000 while provision was made for an additional Stamler hauler in year 2004 at a current cost of R 4,000,000. Financing of this equipment has come from the operating profits of the mine and has been budgeted for in the preceding year. All other capital and operational expenditure from the year 2003 onwards were allocated pro-rata to the thin seam section at a rate of 30%. This was done on the assumption that eventually 30% of ROM production will come from this section.

The overhauling of the Wirth will be done every four years while money is allowed for continuous repairs and overhauling of the other machines (haulers and roofbolters) throughout the life of the project. This should see the equipment through till the end of the life of mine as this kind of mining machinery practically have an unlimited life and only become redundant when new technology replaces them.

Provision was made for additional support of the parting. Extraordinary support was provided for under Capex while normal support falls under the contractor's rates. The current contractor, LTA Grinaker, will charge 10%

above his normal rate for working in thin seam areas. This additional cost was factored in under Operational Costs and includes the additional labour cost for thin seam mining.

No additional costs were allocated for roof brushing. The combined belt and travel road will be one intersection, 8 m wide, and is the only road that will be brushed to a height of 2.0m. The only additional roof brushing will be done in the very low areas of 1.2 m, where the total seam height is in the order of 1.7 m. The brushing will mainly entail the pulling down of the parting and upper seam and in a few instances the blasting into the proper sandstone roof above the No. 2 upper Seam.

The discount rate of 15% was chosen based on T.C.S.A. policy. This rate is based on country risk and market related risks used by the foreign mother company (TOTAL) to evaluate projects in South Africa. Escalations are factored into the financial model (Annexure 10) only as from year 2004 as all costs and increases for 2003 are fixed for the rest of the year. The inflation rate is based on government guidelines to keep inflation between 3 and 6% per annum. The maximum figure is used. The P.P.I. (production price index) used is 5%, based on current figures of 5.4% (Finansies en Tegniek, 9 April 2003, p. 62). The capital escalates at 10% based on the annual devaluation of the Rand against the U.S. Dollar and worldwide inflation of about 2% per year. Inland selling price increases at inflation rate (6%) while the average dollar-selling price for the past 5 years is used for the export sales (US$ 27.77). All other operational costs escalate at 6% per annum.

The yield used in the model is that of the current practice at Dorstfontein to beneficiate to an ash content of 13.5%. At this yield value (95.7%) the N.P.V. is R 27,206 mil. at a discount rate of 15% while the I.R.R. is 305.2% compared to the company's hurdle rate of 15%. The equivalent values at a R.D. =1.6 cutting point is R 15,180 mil. and 120.7% (See

Table 10 Fig. 8.1). The I.R.R. seems to be extremely elevated but it must be kept in mind that this area forms part of the current mine and the capital expenditure is minimal. From Fig. 8.1 it can be seen how the N.P.V. and I.R.R. are influenced by yield increases.

Table 9. N.P.V. and I.R.R. at different product bases.

Product Base	Yield	N.P.V (R mil.) at 15% discount rate	I.R.R (%)
R.D. = 1.6	89.2	15,180	120.7
R.D. = 1.8	93.0	22,252	204.7
Ash = 13.5%	95.7	27,206	305.2

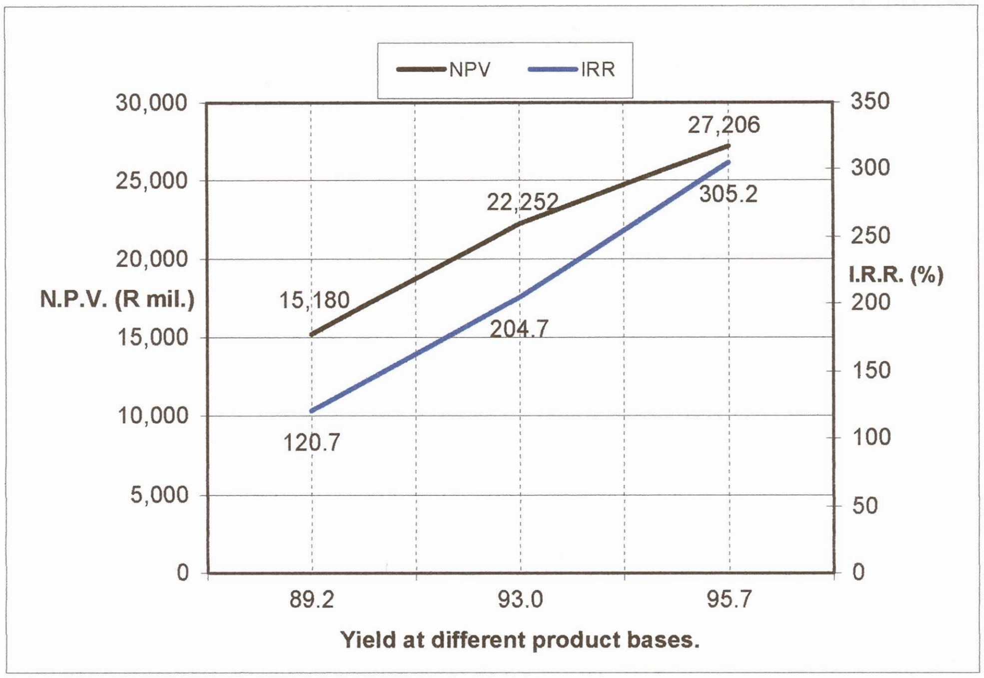


Fig. 8.1. N.P.V. and I.R.R. at different product bases.

The plant factor used in the model might appear to be elevated but it must also be assumed that this factor will increase as less parting is being mined and thus the dilution decrease. It is therefore very important that the mining horizon be maintained and that minimal floor and roof be cut.

Table 10. The N.P.V. for the product bases at different discount rates.

Discount Rate	0.0%	2.5%	5.0%	7.5%	10.0%	12.5%	15.0%	17.5%	20.0%	22.5%	25.0%
13.5% Ash	46,302	41,878	38,078	34,790	31,926	29,417	27,206	25,246	23,501	21,939	20,536
RD = 1.6	26,170	23,686	21,527	19,637	17,972	16,496	15,180	14,000	12,939	11,979	11,107

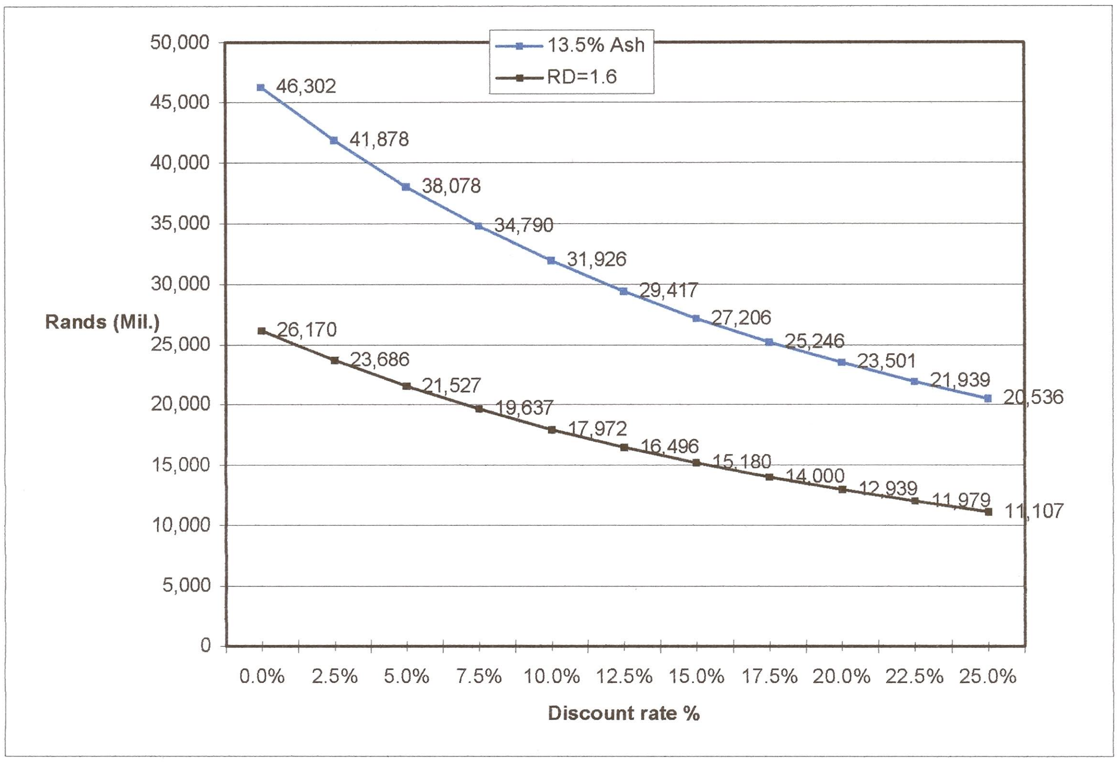
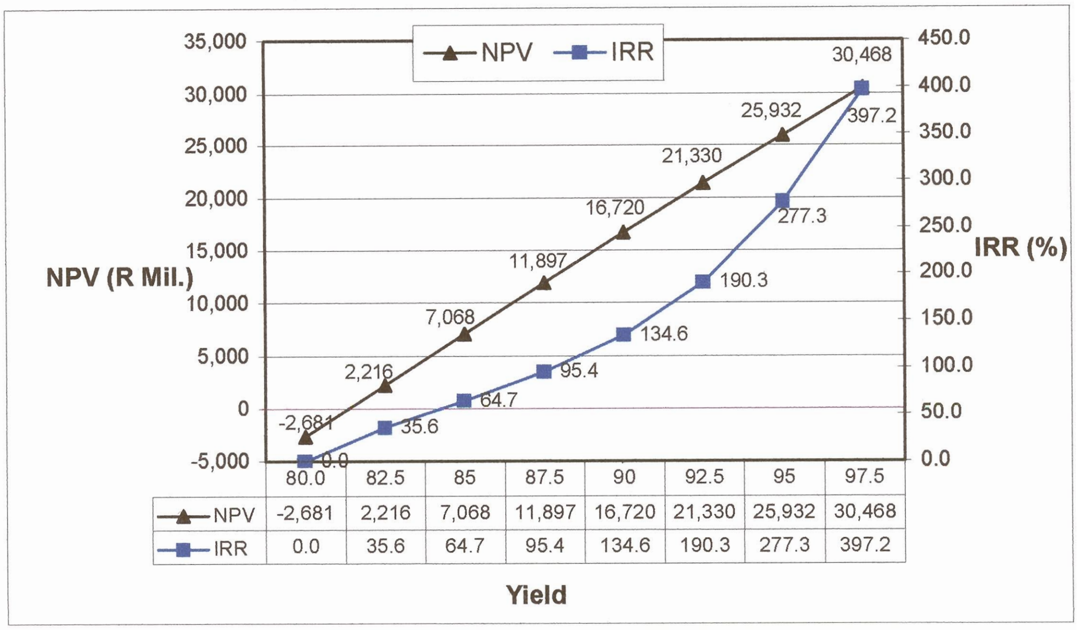


Fig. 8.2. The N.P.V. for the product bases at different discount rates.

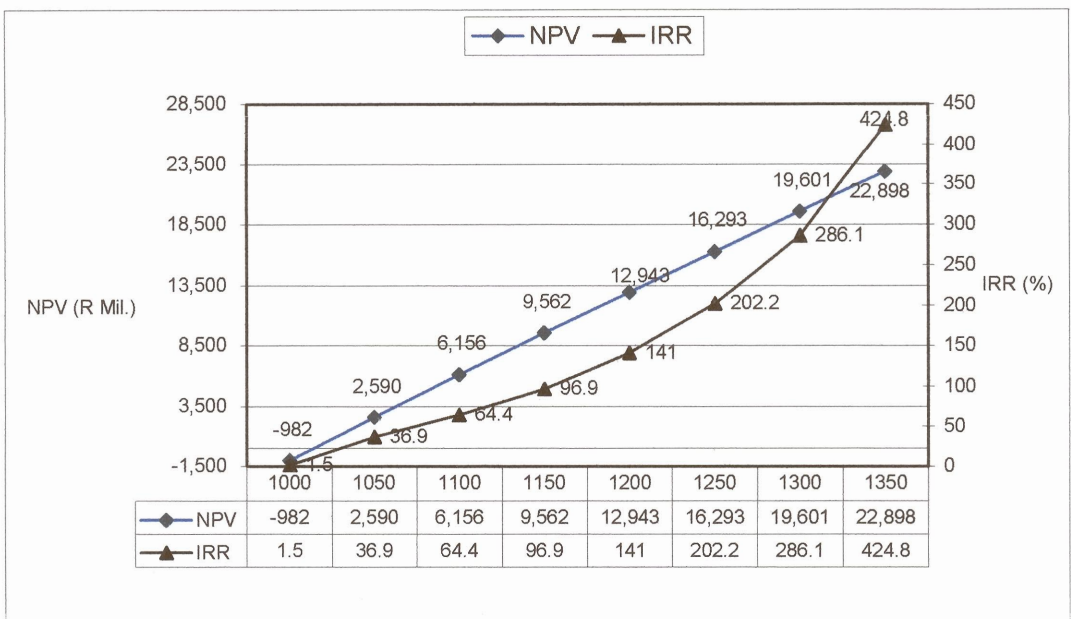
The breakeven yield for the thin seam area is 81.4% (Fig. 8.3) at a discount rate of 15%. This breakeven point is very important as a small amount of contamination can easily make the thin seam uneconomical. This stresses the fact that the seam-split parting should not be mined and that strict horizon control must be maintained during mining. As can be seen the N.P.V. follows a linear line of increase while the I.R.R. exponentially increase or decrease with yield changes. The exponential curve of the I.R.R. line can be explained by the increase in capital expenses during years of machine overalls while all other factors remain constant for the N.P.V.



	80.0	82.5	85	87.5	90	92.5	95	97.5
NPV	-2,681	2,216	7,068	11,897	16,720	21,330	25,932	30,468
IRR	0.0	35.6	64.7	95.4	134.6	190.3	277.3	397.2

Fig. 8.3. N.P.V. and I.R.R. against Yield at a 15% discount rate.

The average breakeven production is 1014 tons per day (Fig. 8.4) at a discount rate of 15%. The payback period is 1.84 years.



	1000	1050	1100	1150	1200	1250	1300	1350
NPV	-982	2,590	6,156	9,562	12,943	16,293	19,601	22,898
IRR	1.5	36.9	64.4	96.9	141	202.2	286.1	424.8

Fig. 8.4. N.P.V. and I.R.R. against daily production rate at a 15% discount rate.

From Fig. 8.4 it can be seen that the N.P.V. an I.R.R. follow similar trends to that of the yield. It is imperative that the production be kept above 1100 tons per day to make this thin seam economical. Many factors influence production in thin seam mines and these adverse conditions will need constant monitoring and management.

8.3. Sensitivity analysis.

Sensitivity analyses were done for the N.P.V. and the I.R.R. at the 13.5% ash content yield and at a discount rate of 15% for the N.P.V.

The following parameters were used to construct a spider diagram (Fig. 8.5) for the sensitivities:

a.) Operating Costs
b.) Selling Price (Export)
c.) Selling Price (Domestic)
d.) Yield
e.) Production
f.) Capital Expenditure

Table 11. Sensitivity of the N.P.V. to certain parameters.

		N.P.V. at 15% discount rate	
Parameter	Variation	Low (R mil.)	High (R mil.)
Operating Costs	30% to -40%	-7,008	70,024
Selling Price (Export)	-40% to 30%	-38,799	68,729
Selling Price (Domestic)	-40% to 30%	-4,515	49,647
Yield	-40% to 0%	14,562	27,206
Production	-40% to 30%	20,072	32,155
Capital Expenditure	30% to -40%	15,425	42,274

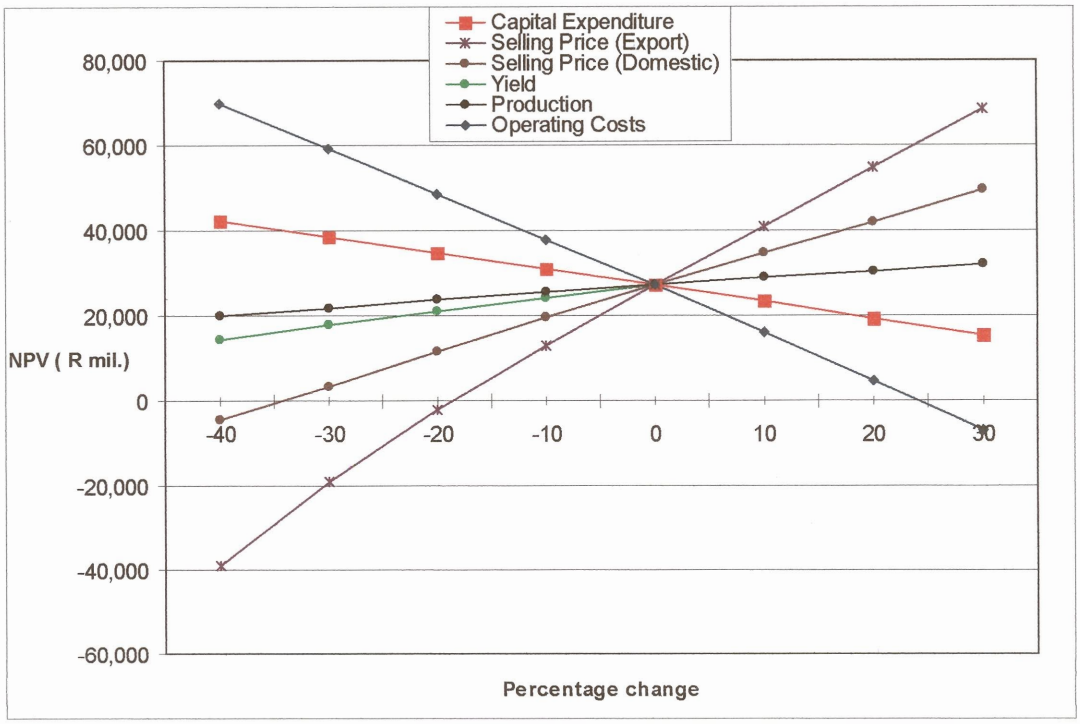


Fig. 8.5. Spider diagram of N.P.V. changes influenced by changes in the sensitivity parameters. (Yield at 13.5% Ash and 15% discount rate)

From Fig. 8.5 it is clear that the project is very sensitive towards export and domestic selling price as well as operating costs. The project is only sensitive to extreme changes in the other parameters (-40% and +30%) which are only likely to happen during or after catastrophic events. See Annexure 7 for detail on the cash flow, Annexure 8 for capital expenditure, Annexure 9 for working cost and Annexure 10 for the escalations used in the financial model.

Table 12. Sensitivity of the I.R.R. to certain parameters.

		I.R.R.	
Parameter	Variation	Low (R mil.)	High (R mil.)
Operating Costs	20% to -10%	54.5	893.4
Selling Price (Export)	-15% to 5%	55.0	561.5
Selling Price (Domestic)	-20% to 5%	101.0	403.6
Yield	-15% to 0%	14.2	305.3
Production	-25% to 10%	37.0	751.5
Capital Expenditure	20% to -10%	126.9	558.9

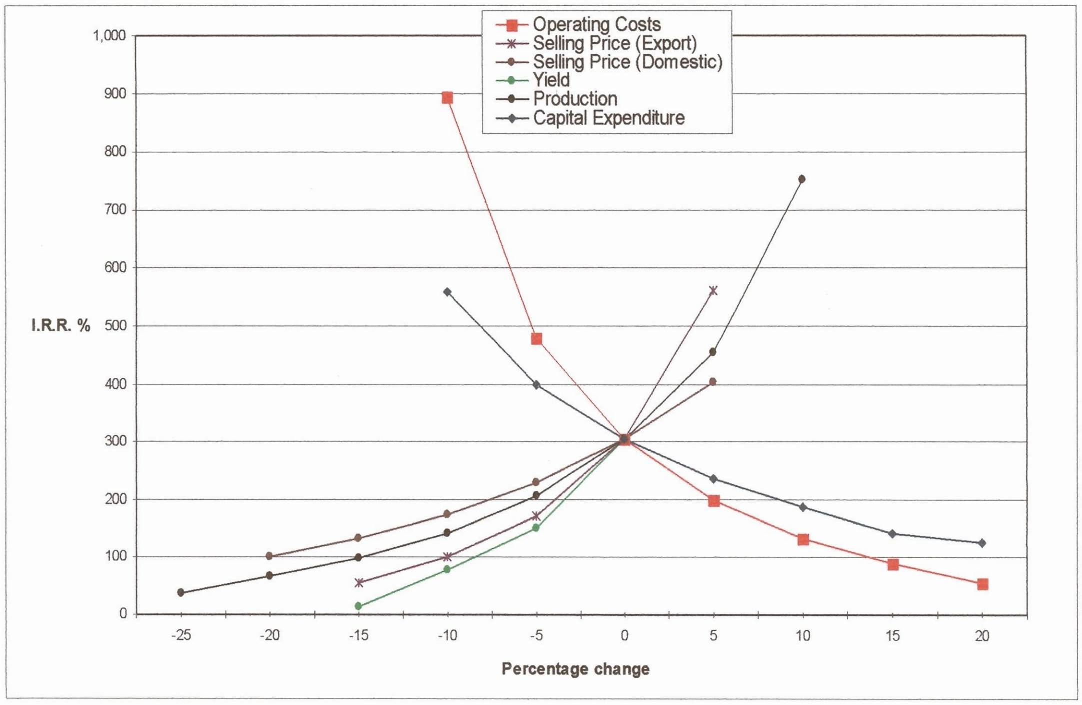


Fig. 8.6. Spider diagram of I.R.R. changes influenced by changes in the sensitivity parameters. (Yield at 13.5% Ash)

The I.R.R. is very sensitive to any changes in the parameters. Small amounts of capital input, the low production rate from the thin seam and sensitivity towards the yield and export prices, result in a very sensitive I.R.R. There is enough confidence in the yield to make the project economic but the greatest uncertainty is the US$ export price which yields a negative I.R.R. at a price drop of about 17% and influences the N.P.V. The good quality of the coal should bring confidence that the export price should not drop excessively low as this rank of coal is well sought after and buyers are willing to pay a premium for this product.

8.4. Shortcomings

There are some shortcomings in the construction of any financial model. The most common problem is the length of time the model should cover. For a shorter life of the mine the model is more accurate than for longer periods and more accurate N.P.V.s and I.R.R.s can be calculated.

Another shortcoming is the escalations that should be built in over a period of time. Few would have predicted that inflation would be 13% in 2002 when it was around 6% in 2001. Who knows what inflation will be in 2008?

Unknown global events will have or can have a huge effect on profitability of an operation. Incidents such as terrorist attacks (U.S.A. bombings), changing governments and legislation can impact heavily on the profitability of an operation.

Another shortcoming is the unpredictability of market requirements. Overseas clients can change specifications on coal, which might negatively affect the coal price. Changes in environmental legislation in the European Community can negatively affect the perception of coal.

Another unpredictable factor is the production rate in the future. There may be a sudden increase in demand from customers, which could positively affect the profitability but negatively affected the life of mine. The opposite is true as a decreasing demand for coal can negatively affect the profitability, as operating costs per R.O.M. ton (R/t) will increase as production decreases. These outside influences are unknown and unpredictable and cannot be accounted for in the discount rate and financial model.

CHAPTER 9: SUMMARY, CONCLUSIONS AND RECOMMENDATIONS.

Dorstfontein Coal Mine has some 7.06 mil. tons of bituminous coal in a thin seam resource with heights varying from 1.2 to 1.4 m. The geological setting of this colliery makes it is a more difficult mine to operate and result in many mining problems. Some mining related problems in the past four years were due to geological features encountered during mining. Experience gained from the past mining can be employed in the thin seam resource area. The big advantage at Dorstfontein is the high quality of the coal. High yields, low sulphur, low phosphorous and low ash values make it a well sought after product. The proposed product from the thin seam resource is a coal with an ash value of 13.5% at a theoretical yield of 95.7%. An alternative product is achieved at a relative density cut of 1.6, which gives a yield of 89.2% and an ash content of 11.25%. Initially this was the product specification until the market started to accept slightly higher sulphur and ash values.

In the south of the Dorstfontein Mine reserve, a persistent seam splitting parting exists varying in thickness from 0.1 to 0.75m. The upper coal seam is very thin (0.1 to 0.35m) and is uneconomical while the lower seam varies from 1.0 to 1.75m in thickness. In the study area the lower seam ranges from 1.2 to 1.4m in thickness and forms the lower economic unit. During extraction of the lower seam the seam-split parting will form the roof. The purpose of this study was to:

a.) determine the possibility of mining the thin seam resource,

b.) study the possible risks,

c.) review and comment on other thin seam coal mining,

d.) to determine the economic value of the thin seam deposit and

e.) quantify the resource in the category 1.2 to 1.4m seam thickness.

The extraction of thin seam coal occurs throughout the world. The definition of thin seams varies from country to country and some countries regarded the cut-off seam height at 24 inches (0.60m) while some European countries regard the cut-off as 1.0m. The largest thin seam coal producers are the U.S.A. and the

Ukraine. Many countries in Europe produced coal from thin seams. Thin seam coal mining became unfavourable due to its low production rate and very high cost of mining. Only a few strategic mines were kept open and heavily subsidized by governments in order to keep them in production for the higher quality coal they produced and to prevent small villages becoming ghost towns due to unemployment. In the U.S.A. many small thin seam collieries exist around West Virginia and the other southern states. Production from these mines is very low and tax incentives were introduced to keep them operating and to act as an incentive for new mines.

In South Africa most thin seam mining took place in KwaZulu-Natal. The most common product was anthracite produced for the steel industry and the export market. Most of the Natal mines are now defunct and only a few small-scale operators are mining under very difficult conditions. The largest operating colliery in KwaZulu-Natal is the Zululand Anthracite Colliery, owned by the Ingwe Plc group. Current production at the mine is 60 000 tons per month.

Most of the literature used in this study comes from old publications since many of the worldwide thin seam collieries were mined in the 1900s and started to tail off in the 1970s to 1980s. A publication by Clarke et al. (1982), Thin Seam Coal Mining Technology, was very helpful in order to determine the risks of thin seam coal mining, health and safety problems and to get an overview of worldwide thin seam mines in the 1970s. In the R.S.A. a few publications on the defunct KwaZulu-Natal collieries was helpful in order to determine the coal fields mined, defunct collieries and current operations.

The advent of the mechanized continuous miner technology has changed the economic parameters pertaining the mining of thinner seams and especially within the range of 1.2 to 1.4m as considered here. This combined with the dwindling high quality thick seam resources may be seen as the principal reason for the current investigation.

One continuous miner section and two drill and blast sections undertake the current mining at Dorstfontein. Most of the mining took place at seam heights in excess of 1.5m. Various mining methods were looked at to extract the thin seam but the most cost effective and productive current technology is the continuous miner method. Currently a thin seam mining trial with an imported German-designed continuous miner (the Wirth Paurat) is taking place. Numerous problems were encountered during the first few months of the trial but availability and production has started to increase. Much of this section's equipment is already on the mine and the only capital expenditure required is that for the continuous miner and possibly a third Stamler thin seam hauler. Additional support needs to be installed to keep the parting up. In the mining trial the parting behaves well and forms a strong beam under which safe working conditions exist.

In the financial model an extraction rate equivalent to 10 years of mining was used. An average daily production of 1400 R.O.M. tons per day for 10 years means that a total extraction of 99.55% of the possible R.O.M. tons can be achieved. The Net Present Value (N.P.V) is R 27,206mil. at a discount rate of 15% and at the yield equivalent to 13.5% ash content. The corresponding I.R.R. is 305.2%. The distorted and high I.R.R. is the result of very low capital expenditure due to the fact that much of the capital equipment has been regarded as sunk costs. Furthermore it must be kept in mind that this is not a stand-alone mine but forms an additional reserve block to the current mining reserve. The real cash flow, at a discount rate of 15% and yield equivalent to 13.5% ash content, is listed in Table 13 and illustrated in Fig. 9.1. The effect of the annual 10% escalation in the capital expenditure is pronounced in year 2010.

Table 13. Cash flow.

Year	2003	2004	2005	2006	2007	2008	2009	2010	2011	2012
Real Cash Flow (Mil. Rands)	-2,884	8,861	9,847	3,936	8,020	8,358	6,244	-3,678	3,140	4,459
Accumulated cash flow (Mil. Rands)	-2,884	5,976	15,823	19,759	27,779	36,137	42,381	38,703	41,843	46,302

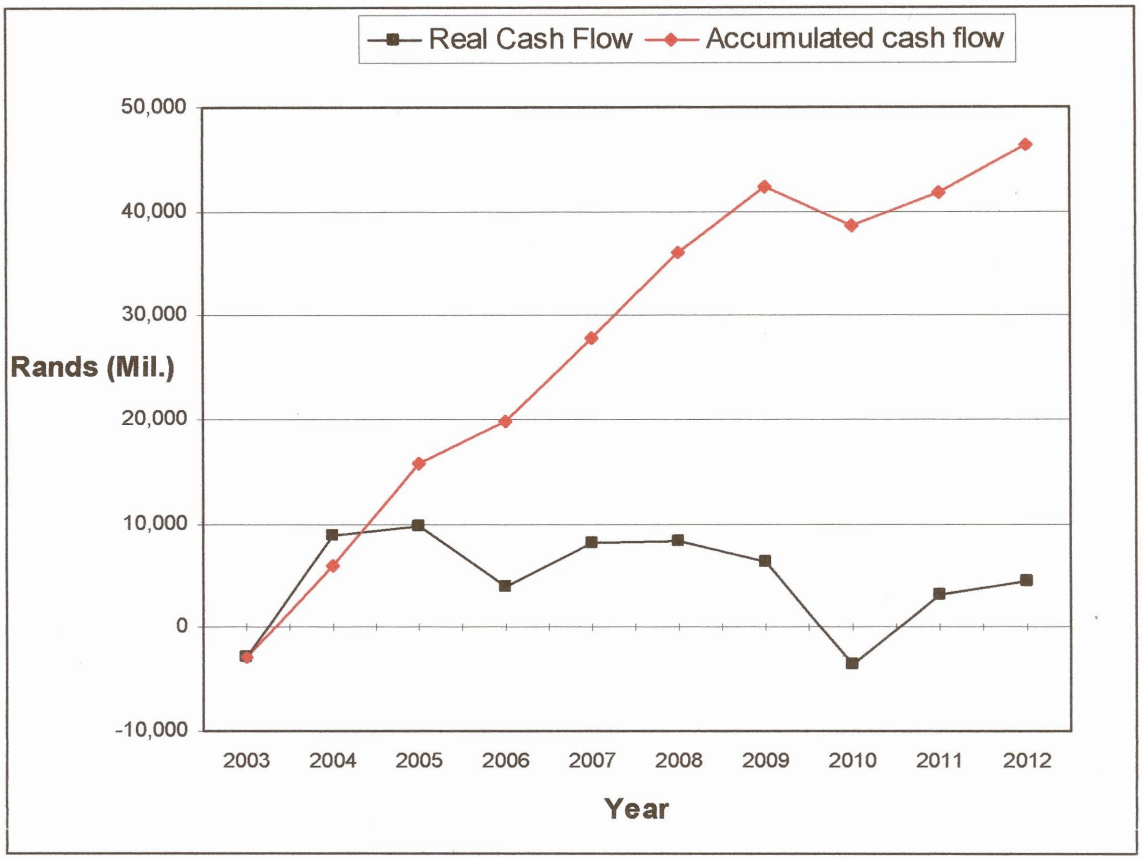


Fig. 9.1. Cash flow.

The project is the most sensitive to the export selling price, operational cost and yield. The sensitivity to the other parameters only becomes a factor at large changes (-40% and 30%). Due to the influence of the sensitivities, especially the yield, on both the N.P.V. and I.R.R. it is recommended that the thin seam coal be washed to a 13.5% ash as this project will not make the hurdle rate of 15% at yields lower than 81.4%. At a yield of 89.2% at RD = 1.6, little margin for error exists.

From this study it is clear to see that the thin seam coal resource at Dorstfontein Mine is of economic importance and of great value. The thin seam coal mining will not be easy and numerous problems may exist. It is worth pursuing the thin seam areas for their high quality coal, high yields and the additional life of mine it presents. It is important to note that mining the thin seam will involve more management input in addressing the risks, auditing the health and safety issues and the training of people to familiarize them in the working of those more difficult mining conditions. Better training and difficult mining conditions will add to more expensive labour. In the financial model the additional labour costs has been added to the contractor's rate.

It is recommended that a similar study is undertaken on the thin seam resources below 1.2 m. Heights ranging from 1.0 m to 1.2 m are regarded as intermediate seam heights in many countries. It is further recommended that short-wall mining methods are investigated for the whole of the thin seam resource within the height range of 1.0 to 1.4 m. Many overseas countries, especially the U.S.A., make successfully use of short-wall mining methods and achieve good production rates under these conditions.

To conclude, one must stress the fact that thin seam mining is only one of the avenues, and probably the most difficult, to increase the potential resource of high grade export and metallurgical coal. The successful and economic exploitation of the thin seam coal reserve at Dorstfontein Mine will impact on the potential for further investigation of similar deposits which may lead to the successful establishment of small scale mining operators from previously disadvantaged communities.

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ANNEXURES

Annexure 1. Thin seam coal deposits of major producing countries.

Annexure 2. U.S.A. Tax.

Annexure 3. Wirth brochure.

Annexure 4. Stamler BH10 brochure.

Annexure 5. Development of a floor and roof classification applicable to collieries.

Annexure 6. Geotechnical analysis.

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Annexure 8. Capital expenditure.

Annexure 9. Operational costs.

Annexure 10. Escalations.

ANNEXURE 2

West Virginia State Tax Department

Publication TSD-398
(Rev. August, 1999)

REDUCED SEVERANCE TAX RATE FOR THIN SEAM COAL PRODUCED FROM NEW MINES

Information contained herein is of a general nature and should be used only as a reference and not a substitute for tax laws or tax regulations.

Coal severance activities are subject to both a State tax, equal to the greater of 4.65 percent of gross receipts (less credits) or 75 cents per ton minimum tax on coal, and a local tax equal to 0.35 percent of gross receipts.

For tax years beginning after April 11, 1997, coal severance activities associated with new underground mines or underground mines not in production between October 14, 1996 and April 11, 1997 are subject to a reduced severance tax rate if the seam thickness of such mines is forty-five inches or less. The determination of actual seam thickness would be based upon a report by a professional engineer who uses an isopach mapping technique.

For qualified mines with a seam thickness of less than thirty-seven inches, the State tax equals the greater of 0.65 percent of gross receipts (less credits) or 75 cents per ton. The local tax remains at 0.35 percent of gross receipts.

For qualified mines with a seam thickness between thirty-seven inches and forty-five inches, the State tax equals the greater of 1.65 percent of gross receipts (less credits) or 75 cents per ton. The local tax remains at 0.35 percent of gross receipts.

If a coal processor purchases coal from a qualified thin seam mine then additional processing activities associated with such coal would be subject to the same reduced tax rate as applicable to the initial severance activity. However, processors must maintain a log with records of qualified tons and receipts subject to alternative tax rates.

Thin seam coal produced from qualified mines remains subject to the 75 cents minimum tax. The minimum tax provides some degree of tax equity among all West Virginia coal producers. Absent such an equalizer, qualified mines subject to preferential tax rates would enjoy a significant competitive advantage over other West Virginia mines. The minimum tax provisions should mitigate potential losses of employment, production and tax receipts at those mines not subject to preferential tax rate treatment.

Taxpayers must separately account coal receipts subject to the three alternative State tax rates of 4.65 percent, 0.65 percent and 1.65 percent. The following may provide some guidance:

Example 1:

KL Mining Company begins operations at a new low seam mine. First year coal sales total 200,000 tons at $30.00 per ton. The seam thickness as determined by isopach mapping techniques is less than 37 inches. The following tax calculations apply:

§ 11-12B Tax		
State Minimum Tax: 200,000 tons x $0.75/ton	**= $**	**150,000**
§ 11-13A Tax		
Gross Receipts: 200,000 tons x $30.00/ton	**= $**	**6,000,000**
Tax Rate on Receipts: 0.65% + 0.35%	**= x**	**1.0%**
Gross Tax: State Local	**$**	**60,000**
Annual Exemption Credit:	**-**	**500**
Net Tax:	**= $**	**59,500**
State Share: (0.65/1.00) x $59,500 = $38,675		
Net Minimum Tax: ($150,000 - $38,675)	**= +**	**111,325**
Total Tax (including local share):	**= $**	**170,825**

Example 2:

MSM Mining Company begins operations at two new low seam mines. First year coal sales total 300,000 tons at $30.00 per ton. The seam thickness as determined by isopach mapping techniques is less than 37 inches at Mine A (production = 100,000 tons) and between 37 and 45 inches at Mine B (production = 200,000 tons). The following tax calculations apply:

§ 11-12B Tax		
State Minimum Tax: 300,000 tons x $0.75/ton	=	$ 225,000
§ 11-13A Tax		
Gross Receipts: 300,000 tons x $30.00/ton	=	$ 9,000,000
Tax Rate of Receipts:		
(100,000 tons x $30.00/ton)/$9,000,000 x 0.65%		
(200,000 tons x $30.00/ton)/$9,000,000 x 1.65%		
1.3167% + 0.35%	=	x 1.6667%
Gross Tax: State Local		$ 150,000
Annual Exemption Credit:		- 500
Net Tax:	=	$ 149,500
State Share: (1.3167/1.6667) x $149,500 = $118,105		
Net Minimum Tax: ($225,000 - $118,105)	=	+ 106,895
Total Tax:	=	$ 256,395

Example 3:

JM Mining Company Produces 1,000,000 tons from various mines that have been in operation for several years. Coal from these mines is sold under contract for $30.00 per ton. JM reopens Low Profit Mine, a low seam (i.e., less than 37 inches) not in operation since 1989. JM sells 150,000 tons of coal from Low Profit Mine at an average price of $25.00 per ton. JM also opens New Mine and sells 100,000 tons of coal from this mine at an average of $24.00 per ton. The seam thickness as determined by isopach mapping techniques is less than 37 inches at Low Profit (production = 150,000) and between 37 inches and 45 inches at New Mine (production - 100,000 tons). JM also has a Coal Loading Facility Credit equal to $30,000. The following tax calculations apply:

§ 11-12B Tax		
State Minimum Tax: 1,250,000 tons x $0.75/ton	=	$ 937,500
(1,000,000 + 150,000 + 100,000)		
§ 11-13A Tax		
Gross Receipts: 1,000,000 tons x $30.00/ton		
+ 150,000 tons x $25.00/ton		
+ 100,000 tons x $24.00/ton	=	$36,150,000
Tax Rate of Receipts:		
(1,000,000 tons x $30.00/ton)/$36,150,000 x 4.65%		
(150,000 tons x $25.00/ton)/$36,150,000 x 0.65%		
(100,000 tons x $24.00/ton)/$36,150,000 x 1.65%		
4.036% + 0.35%	=	x 4.386%
Gross Tax: State Local		$ 1,585,500
Credits (Coal Loading & Exemption):		- 30,500
Net Tax:	=	$ 1,555,000
State Share: (4.036/4.386) x $1,555,000 = $1,430,909		
Net Minimum Tax: ($937,500 - $1,430,909)	=	+ 0
Total Tax:	=	$ 1,555,000

If you have further questions regarding reduced severance tax for thin seam coal, please contact the Sales Tax Unit, Internal Auditing Division. A question in writing should be submitted to:

West Virginia State Tax Department
Internal Auditing Division - Sales Tax Unit
Post Office Box 425
Charleston, West Virginia 25322-0425

You may also telephone (304) 558-3333 or toll-free at: 1-800-982-8297

ANNEXURE 1

Geological Parameters	Former USSR	USA	Spain	United Kingdom	Czechoslovakia	Poland	Colombia
Definition of thin seams	1.2 m (48 inches)	-	-	0.91 m (36 inches)	1.0 m (39 inches)	1.0 m (39 inches)	-
Seam dip	Gentle to very steep	Mainly flat	0-90	0-45 Mostly 0-6	0-16 51% 16-36 34% +36 15%	0-10 39% 1 0-45 54% +45 7%	Flat to steep
Seam depth	300- 1,1 00 m (984-3,609 ft)	Reserves calculated to a depth of 1,000 ft	500 m (1,640 ft)	1,100 m (3,609 ft)	400-600 m (1,312-1,968 ft) Some at 1,000 m (3,281 ft)	0-800 m (0-2,625 ft)	-
Coal strength	Variable	Variable	-	Hard	Hard	-	-
Roof	6.4% sandstone 8.0% limestone Rest shale	Generally good and strong. Frequent draw slate	Strong, variable	Shale	-	Sandstone, silts and conglomerates	Variable
Floor	Mostly clay shales	Medium	Strong, variable	Mostly clays	-	Sandstone, silts	-
Extent of Seams	Donetz and Lvov-Volynsky	Wide areas but thickness in seams varies over area	Fragmented	Northumberland, Durham, York-shire and Derby-shire	Ostrava Karvina and Eastern Bohemia	-	
Water	Mostly dry, but some very wet	Fairly dry when worked above drainage table	Variable	Mostly dry	-	-	-
Faults	Normally undisturbed	Normally undisturbed	Highly disturbed	Mainly undisturbed	Highly disturbed except Wales and Scotland	-	Disturbed
Cleat	Mostly well defined	Not generally well defined	-	Mostly well defined	-	Mostly well defined	Variable
Spontaneous combustion	Variable risk	Variable risk	-	Low risk in thin seams	-	-	Low risk
Methane	Variable emission	Generally low emission	Low emission	Mainly gassy	All gassy	Mainly gassy	Mainly non-gassy
Quality	Coking coal	Often coking and low sulphur	-	Often coking	High quality coking coal		-

Annexure 1. Thin seam coal deposits of major producing countries (Clark et al., 1982)

Geological Parameters	France	Belgium	Germany	China	Bulgaria	Romania
Definition of thin seams	1.0 m (39 inches)	0.6 m (24 inches)	0.7 m (28 inches)	-	1.3 m (51 inches)	-
Seam dip	0 -20 47% 20-45 46% +45 7%	0-45 mostly 0-30	0-10 63% 10-20 9.5% +20 27.5%	Flat or slight 69 7% 10-25 22.4% +25 7,9%	10-90 mostly -45	5-70
Seam depth	-	275-1,160m (902-3,806 ft)	Maximum 1,200 m (3,937 ft)	Mostly <200 m (656 ft)	150-300 m (492-984 ft)	-
Coal strength	-	Variable	Soft but hard in the Saar		-	-
Roof	-	Competent shale and sandstone	Shale, sandy shale in thin seams	-	Hard sandy shales	-
Floor	-	Good shale and sandstone	Shales, sandy shales	-	-	-
Extent of Seams	Nord, Pas de Calais, Cevennes	Charieroi-Namur, Liege	Aachen and lower Saxony	Widely distributed	Svoge Basin and Balkan field	Valea-Jiului and Anina
Water	-	-	Mostly dry	-	-	Dry
Faults	-	Undisturbed	Mainly undisturbed	Undisturbed	Highly disturbed	Highly disturbed
Cleat	Mostly well defined	Variable	Mostly well defined	-	-	Not generally well defined
Spontaneous combustion	-	-	Variable risk	-	Low risk	-
Methane	Mainly gassy	-	Low emission	Some gassy	Some gassy	Mainly gassy
Quality	-	Anthracite	Coking coal	-	Anthracite	Coking coal

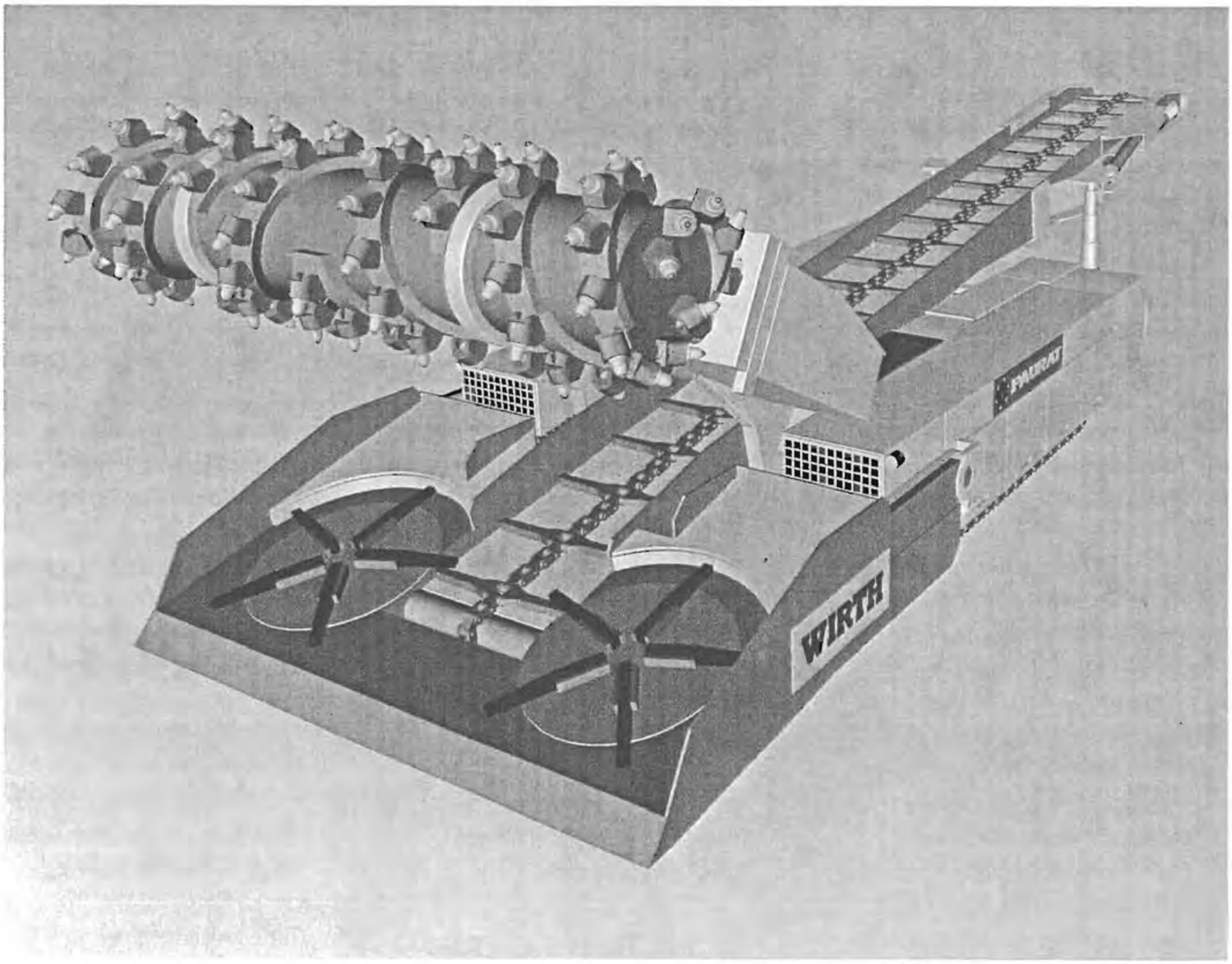
Annexure 1 cont. Thin seam coal deposits of major producing countries (Clark et al., 1982)

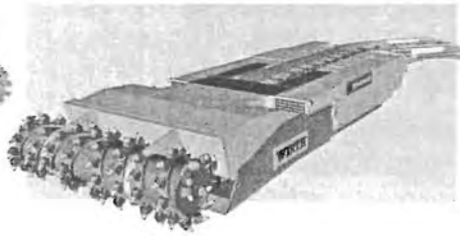
ANNEXURE 3

WIRTH

Low Seam Coal Header

H4.30

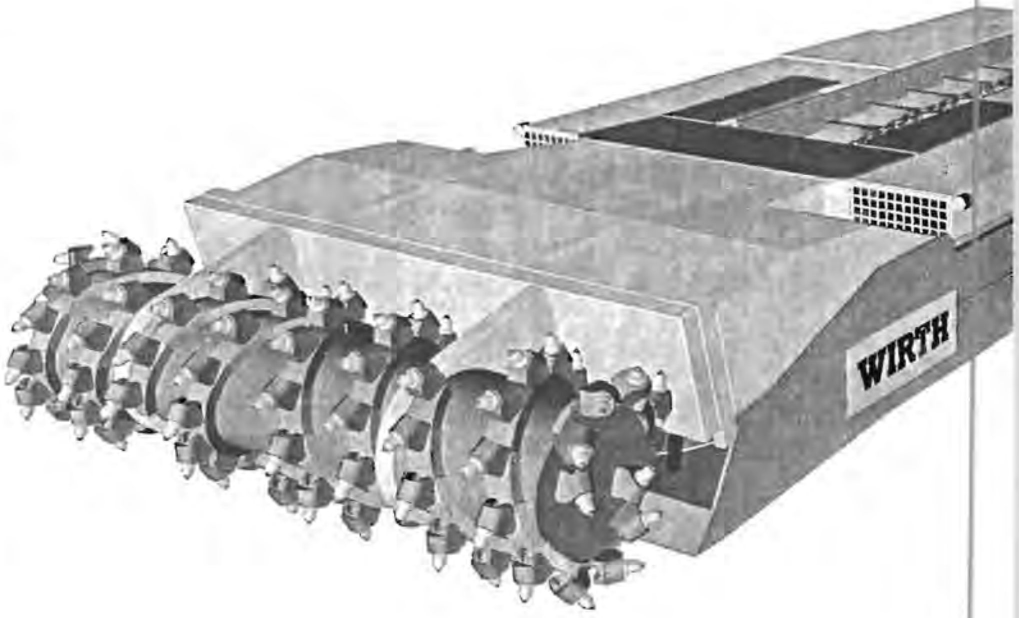




Low Seam Coal Header WIRTH PAURAT H4.30

1. The H4.30 Low Seam Coal Header combines the strength, robustness and versatility of WIRTH PAURAT's heavy-duty roadheader range with the ability to cut and load minerals, such as coal, potash, salt, etc. at a very high production rate.

   The Coal Header with a weight of approx. 50 t is designed to withstand the toughest of under ground conditions during long periods of use. It can deal with rock inclusions, washouts and undulations in the seam.



2. The H4.30 is capable of cutting and loading a cross-section up to 3.5 m wide and up to 2.80 m high from a single central position. With an overall height of only 1,000 mm the machine can operate in cross-sections only 1.1 m high on plain floor conditions.

3. The machine is equipped with a WIRTH PAURAT "Helix" cutting drum powered by two water-cooled and water-tight electric motors via epicyclic gearboxes. The cutting drum is divided into three sections - a centre drum and two outer drums.

   In operation the drum not only cuts but also crushes and conveys the material onto the loading apron. Cutting is carried out by tungsten carbide tipped point attack picks arranged in a double spiral around the drum. Wear resistant steel scrolls convey the cut material to the loading apron, and also protect the pick boxes and limit pick penetration. The loading apron behind the drum conveys the material by the two loading stars on the chain conveyor.

4. The main frame of the machine is constructed from solid cast steel components to give it the necessary mass to react the cutting forces within the compact overall dimensions. The individual components are bolted together for ease of assembly and transport as well as service inside the production area.

   The crawler tracks are integrated into the main frame, each track being independently driven by an electric, AC, motor, with variable speed by inverter control.

   The crawlers are fitted with 500 mm wide track plates. The crawlers have sufficient power to enable the machine to operate on tramming gradients up to +/- 18 degrees.

5. The machine is equipped with a high capacity roller chain conveyor powered by the two loading star

drives. The conveyor transports the cut and crushed material from the loading apron to the rear of the machine. The tail of the conveyor can be raised, lowered and slewed from side to side hydraulically enabling it to load almost any muck haulage system.

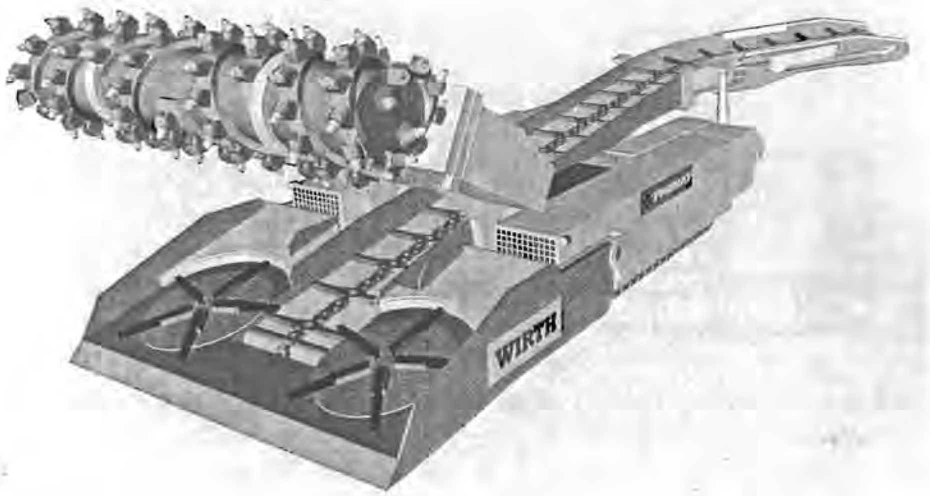
6. All drives, i.e. crawlers, conveyor, cutting drum and loading stars are electrically driven. All other functions of the machine are operated hydraulically. The power pack comprising tank, pumps with water electric motor, filters, coolers etc. is located on the right hand side of the machine. Preset level and temperature switches protect the system which is suitable for use with both normal mineral oil and HF-C fire resistant fluids, resp.

The main valves are operated by a radio remote control system.

7. As standard the machine is designed for use with an electrical power supply rated 1000 V / 50 Hz. The electrical system can also be modified for use with other voltages and 60 Hz frequency.

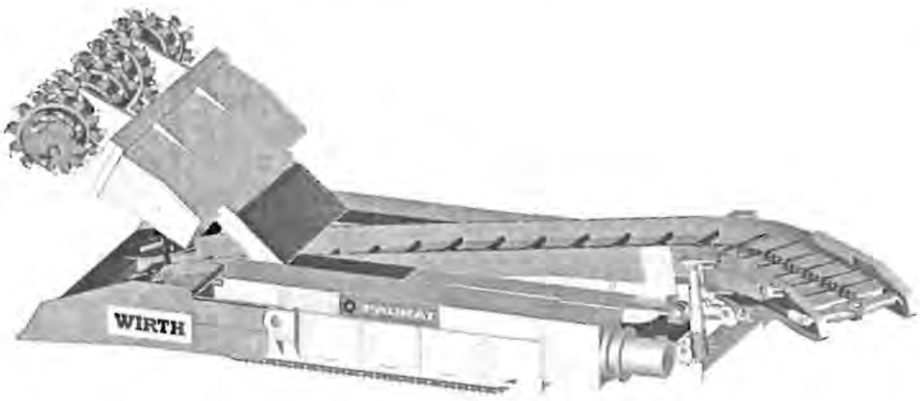
The machine can also be supplied for use in gassy mines in full compliance with the regulations of the relevant governing authorities.

The switchgear for all the motors on the machine and the main circuit breaker for the power supply, are all contained in one contactor case located on the right hand side of the machine. All motors are protected against both thermal and current overload as well as against earth leakage.



Start and stop buttons for all motors as well as ampmeters and fault indication lamps are located at the control panel. Emergency stop buttons are provided at several points around the machine.

8. On the left hand side of the machine is the wet dust collection system installed, which in combination with the unique water spray system offers excellent dust absorbtion for good visibility at low consumption of water to reduce mud spillage at the floor.



WIRTH

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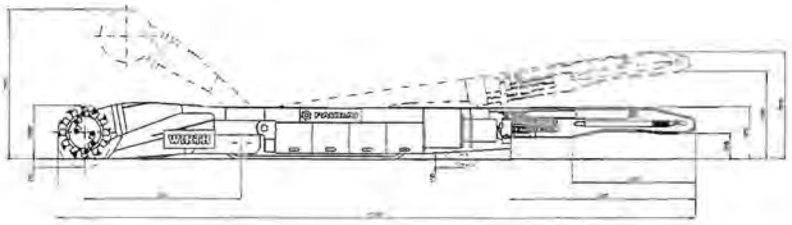
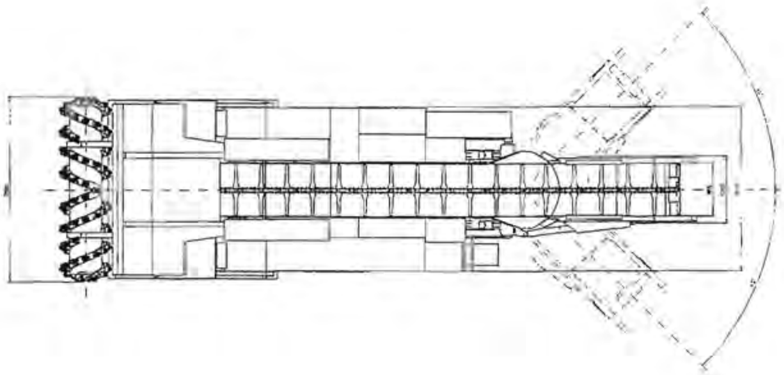
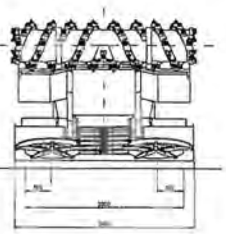
Low Seam Coal Header WIRTH PAURAT H4.30

Technical Data:







List of technical data for WIRTH PAURAT H4.30

Machine Overall		
Weight	50	t
Lenght	12100	mm
Height	1000	mm
Cutting heigth	1100-2800	mm
Cutting width	3500	mm
Crawler Tracks		
Speed	0-30	m/min
Drive	AC-motors	
Cutting Drum		
Installed power	2 x 200	kW
Diameter	1000	mm
Hydraulics		
Installed power	45	kW
Electrics (standard)		
Voltage	1000	V
Frequency	50	Hz

WIRTH

Maschinen- und Bohrgeräte Fabrik GmbH
P.O. Box 16 60
41806 Erkelenz
Germany

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Telefax: +49 24 31 83-267
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ANNEXURE 4

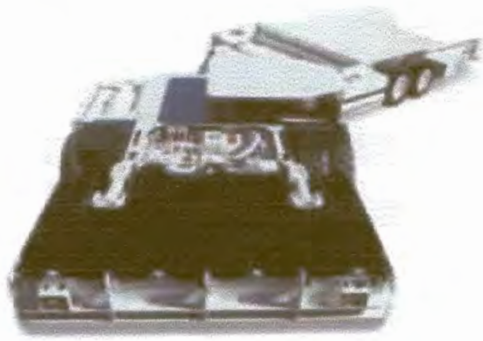
Stamler BH10 Battery Hauler

The Stamler BH10 Battery Hauler provides increased payload and low operating cost per ton. Compared to shuttle cars, the Stamler BH10 Battery Hauler offers a superior capacity, quiet and comfortable operation, and eliminates trailing cable problems.

High Performance Operation

- Substantial payload capacity in a variety of operating heights down to 890mm improves mine efficiency.
- Vertical articulation and patent pending "on demand" four wheel drive system allow efficient operation in difficult bottom conditions.
- Extended service life components are located in maintainance-friendly locations for easy access, even in low heights.
- Patented "Lift-from-grade" battery change system provides maximum uptime.
- Vertical articulation improves utility by allowing transport of components.
- Efficient, field-proven IGBT motor control system provides maximum shift battery life with the most efficient use of power.
- Ergonomically designed cabin offers good operator visibility.
- Quiet operation provides high operator comfort.

Features



Low Profile

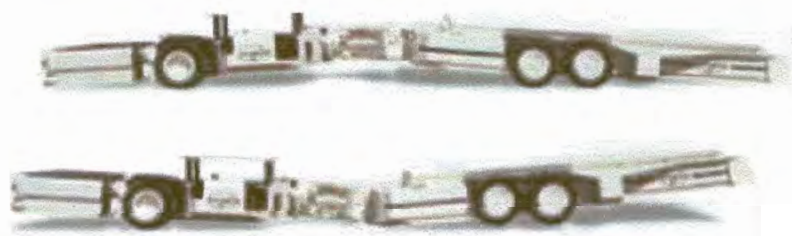
Low profile allows operation in seams as low as 890mm.

Efficient Hydraulic System

Reliable, high output system provides power for improved haulage cycle time.

Patented "Lift From Grade" Battery Change System

Allows fast and easy battery change for maximum uptime.



Vertical Articulation

Total vertical articulation of 25° (15° up, 10° down) allows efficient operation in difficult bottom conditions and enables loading and unloading of material or components.

OLDENBURG STAMLER

ANNEXURE 5

Development of a roof and floor classification applicable to collieries

35

Le développement d'une classification pour le toit et le mur en charbonnages

Die Entwicklung einer Klassifikation von Dach und Sohle, anwendbar für Kohlengruben

P. S. BUDDERY and D. C. OLDROYD, Genmin, Witbank, South Africa

Coal measures strata together with coal mining may be viewed very much as special cases with regard to rock engineering considerations. The strata are frequently laminated, generally weak and variable in character and thickness over relatively short distances. Coal mining is typically highly mechanized resulting in rapid geographical expansion and large areas of exposed roof, sides and floor. A roof and floor classification system for use by a major coal mining operation needs to be based on tests that enable large numbers of samples to be tested, including samples from the weakest strata, in ways that are related to the commonest forms of strata control problems.

Les terrains de charbon et l'exploitation de charbon peuvent être considérés comme des cas particuliers de mécanique de roches. Le terrain est souvent stratifié, normalement faible et variable en propriété et épaisseur sur des distances limitées. L'exploitation de charbon est bien mécanisée, ce qui a comme conséquence l'expansion geographique rapide et l'exposition de terrains dans lesquels le toit, le mur et les parois. Une classification du toit et du mur, appliquée par un charbonnage important, doit être basée sur des tests d'un grand nombre d'échantillons, y inclus des échantillons de roches faibles, d'une telle façon que les problèmes classiques de comportement de terrains en charbonnages sont abordés.

Das Kohlengebirge und der Abbau von Kohle können als Spezialfälle in der Gebirgsmechanik betrachtet werden. Die Schichten sind häufig laminiert, allgeimen wenig fest und über relativ kurze Entfernungen veränderlich in Gepräge und Mächtigkeit. Die mechanisierte Gewinnung von Kohle ergibt einen schnellen Abbaufortschritt mit grossen Flächen von freigelegtem Dach, Stoss und Sohle. Das System einer Dach-und Sohlenklassifikation für den Gebrauch in einer Kohlenindustrie-Gruppe muss auf Versuchen beruhen, die es ermöglichen eine grosse Anzahl von Proben zu nehmen, einschliesslich solchen von geringster Festigkeit, und das auf die alltäglichen Probleme der Gebirgsbeherrschung bezogen ist.

INTRODUCTION

The economic coal measures of South Africa occur predominantly in the Middle Ecca stage, and to a much lesser extent in the Upper Ecca and Molteno stages, of the Karoo system. The Karoo system is of Permian age thus making the South African coal measures somewhat younger than their European counterparts.

The coal bearing strata consist chiefly of sandstones with subordinate shales, carbonaceous shales, siltstones and mudstones.

Many of the coal measures strata are inherently weak while others are highly susceptible to weathering. Significant variation in the properties and thicknesses of a particular stratum over short horizontal distances is also a notable feature of many of the coalfields, as is the occurrence of dolerite intrusions in the form of both dykes and sills.

The majority of underground coal is extracted by means of mechanized bord and pillar methods from seams lying at shallow depths. Consequently, most mines experience a rapid rate of geographical expansion resulting in vast expanses of exposed roof, sides and floor being created, many of which have to be maintained for long periods of time, particularly if pillar extraction is contemplated.

THE NEED FOR A COAL MEASURES CLASSIFICATION

A number of tests are available for the determination of rock strengths and other properties such as durability and the potential for swell. The carrying out of these tests is governed by guidelines laid down by the ISRM (ref.1). Similarly, well-established rock mass classification systems exist which have proven themselves in numerous practical applications. When dealing with the soft rocks of typical coal measures strata, however, there are certain drawbacks with regard to the use of these tests and classification systems. These include:

i) The tests or classification parameters may not relate directly to actual strata behaviour in coal mine roadways.

ii) Sample preparation requirements and test procedures may make it impossible to test weak strata so that the behaviour of these strata has to be inferred from experience.

iii) The tests are typically costly, time consuming and can only be conducted in specialist laboratories. This presents significant difficulties when very large numbers of tests are required such as during the feasibility stage of a major coal mining project.

iv) Rock mass classification systems will frequently assign the same class to a wide range of coal mine roofs.

TRANS-NATAL'S ROOF AND FLOOR CLASSIFICATION

The Coal Division of Genmin's Rock Engineering Department has always desired and striven to become more pro-active in order to anticipate strata control problems rather than to deal with them only once they become apparent. In order to do so it was essential to develop a means of classifying coal measures strata. The size of the department, budget constraints and the scope of work involved meant that the following philosophy had to be applied in devising a suitable classification system:

i) The tests should relate to the expected mode of failure of the strata.

ii) It should be possible to test even the weakest material.

iii) Large numbers of tests should be able to be conducted simply, quickly, at low cost and in-house.

The achievement of these aims was considered worth losing a degree of accuracy for.

Roof Classification

Roof failure in South African coal mines is predominantly governed by the frequency of laminations or bedding planes and their propensity to open and separate, and by the bord width. This is in accordance with the formula for tensile stresses in a fixed beam which gives the maximum tensile stress, P, developed in a beam of unit width as:

$$P = \frac{\rho g B^2}{2t} \qquad (1)$$

where: ρ = strata density
g = gravitational acceleration
B = bord width
t = beam thickness

Tests designed to indicate the potential for roof failure must therefore indicate the frequency of bedding planes and laminations, and their potential to open. During 1982 the introduction of a Coal Rock Structure Rating (CRSR) system was considered. This was based on three parameters; RQD, the results of impact splitting tests and a parameter related to joint condition and groundwater.

In coal measures strata it is impractical to satisfactorily distinguish between drilling induced and natural fractures in the rock. Therefore, the RQD was discarded from the system although it is still determined for all strata that are of interest and used, where necessary, to assist in interpretation.

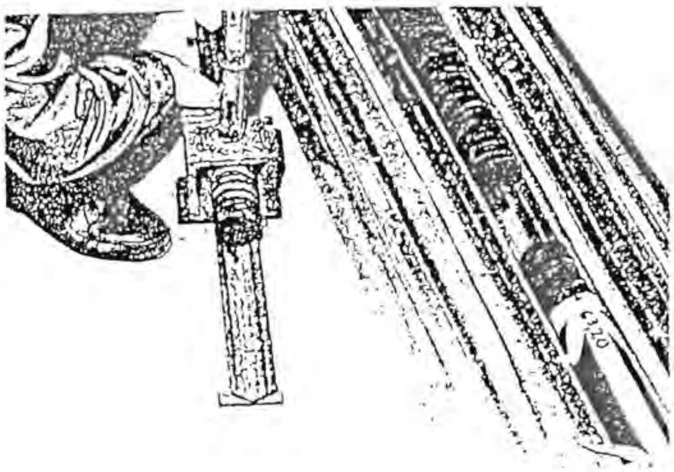


Fig.1. Impact splitting test

The third parameter proved difficult to determine. Furthermore, irrespective of the roof type, special support precautions are taken at all geological discontinuities exceeding 2m in length. Joints, therefore, unless they are exceptionally closely spaced have no influence on systematic roof support design. Consequently, in 1983 it was decided to confine the determination of roof ratings to the results of impact splitting tests.

Impact Splitting Tests

The impact splitting test involves imparting a constant impact to a length of core every 0,02m. The resulting fracture frequency is then used to determine a roof rating.

The instrument used is very simple (Fig. 1). It consists of an angle iron base which holds the core. Mounted on this is a tube containing a chisel with a mass of 1,5kg and a blade width of 25mm. The chisel is dropped onto the core from a constant height according to core size, 100mm for TNW (60mm dia.) and 64mm for NQ (48mm dia.).

The impact splitter causes weak or poorly cemented bedding planes and laminations to open under duress thus giving an indication of likely behaviour in situ when subjected to bending stresses, in some instances compounded by blasting.

When designing coal mine roof support, 2m of strata above the immediate roof are tested. If the roof horizon is in doubt then all strata from the lowest likely horizon to 2m above the highest likely horizon are tested so that all the potential horizons may be compared. For shaft boreholes the full length of strata is tested (ref. 2).

The strata is divided into geotechnical units which are very often shorter than the units described by the geologist. The RQD for each unit is determined and any geological discontinuities are noted. The units are then tested and a mean fracture spacing for each unit is obtained. Using either equation (2a) or (2b) an individual roof rating for each unit is determined.

$$\text{For } fs \leq 5 \quad \text{rating} = 4fs \qquad (2a)$$
$$\text{For } fs > 5 \quad \text{rating} = 2fs + 10 \qquad (2b)$$

where: fs = fracture spacing in cm

For example, a unit 1,2m long with 8 fractures will have a mean fracture spacing of 15cm and a unit rating of 40.

This value may be used to classify the individual strata units (Table 1) but for coal mine roofs the individual ratings are adjusted to obtain a roof rating for the first 2m of roof. The immediate roof unit will have a much greater influence on roof conditions than a unit 2m above the roof. Consequently, the unit ratings are weighted according to their position in the roof by using equation (3).

$$\text{Weighted rating} = \text{rating} \times 2(2 - h)t \qquad (3)$$

where: h = mean unit height above the roof (m)
t = thickness of unit (m)

The weighted ratings for all units are then totalled to give a final roof rating. For example, a coal mine roof has three units; 0-0.8m; 0.8-1.3m and 1.3-2.8m above the coal seam with ratings of 25, 32 and 8, respectively. For the purpose of determining a weighted rating the last unit is regarded as being from 1.3-2.0m above the coal seam. From equation 3 the weighted ratings at the mean heights of 0.4m, 1.05m and 1.65m are 64, 30.4 and 3.9, respectively. The final roof rating is therefore: 64+30.4+3.9 = 98.3.

After many years of experience and having collected data from numerous sites the classification given in Table 1 has been arrived at. Good agreement between expected and actual roof conditions has been found when using this rating system.

Table 1. Unit and coal roof classification system

Unit Rating	Rock Class	Roof Rating
<10	Very Poor	<39
11 - 17	Poor	40 - 69
18 - 27	Moderate	70 - 99
28 - 32	Good	100 - 129
>32	Very Good	>130

Floor Classification

The floor classification system was developed in late 1988/early 1989 for the feasibility study to the T-project which was investigating the extraction of torbanite and its conversion to syncrude. Torbanite is found in the N°5 coal seam of the Highveld coalfield (Fig. 6) which is notorious for poor floor conditions. Floor strata are liable to swell and degrade due to water. The mechanical action of mining equipment is also a major contributory factor to the degradation of the floor. In the light of the above it was decided to base the floor classification system on unconfined swelling strain and slake durability tests. In order to adhere to the aforementioned testing philosophy it has been necessary to modify the suggested methods as laid down by the ISRM. Only the modifications will be discussed here, for full details of the test methods the reader should refer to the ISRM suggested methods (ref.1).

Duncan Swell Test

The Duncan swell test measures the unconfined swelling strain in one or more directions when a sample of rock is immersed in water. When testing borehole cores from coal measures strata it is only necessary to measure the swelling strain perpendicular to the laminations since, in rocks liable to swell, the swelling strain perpendicular to the laminations will greatly exceed that in other directions.

Samples are not prepared but are chosen with their ends approximately parallel. This reduces the costs and time involved and, above all, allows the testing of weak samples that would otherwise break up during machining.

The test procedure requires that swelling displacement should continue to be recorded until it reaches a constant level or passes a peak. This can be extremely time consuming and, for practical purposes, is not necessary. For the vast majority of specimens, 90% or more of their final swell will have taken place by the time 30 minutes have elapsed. For this reason a 30 minute swelling strain is determined. A sample undergoing testing is shown in Fig.2.

The swelling strain, S_{30}, is calculated as follows:

$$S_{30} = \frac{d_{30}}{L} \times 100\% \qquad (4)$$

where: d_{30} = swelling displacement after 30 minutes
L = initial length of the sample.

At the end of the test the sample is immediately removed from the water. It is then assigned a rating from 1-6 according to its condition. A rating of 1 being assigned to an undisturbed sample and a rating of 6 to a totally degraded one (Fig.3). The swell index of the sample is then determined by multiplying the swelling strain by the condition rating.

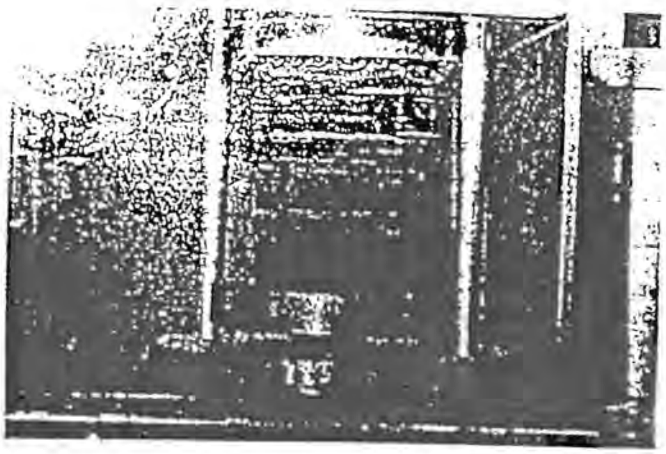


Fig.2. Duncan swell test

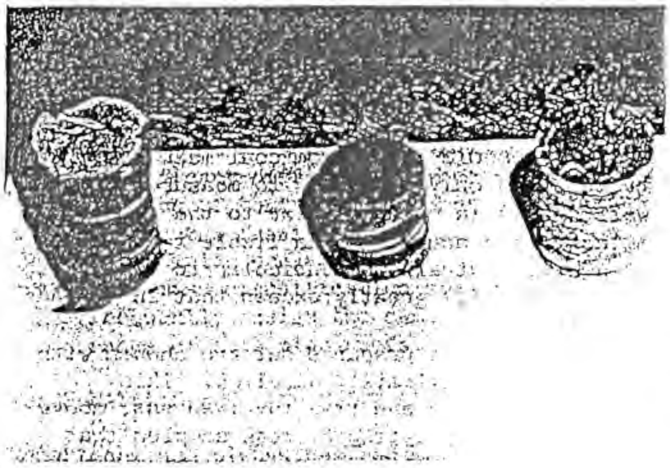


Fig.3. Samples after Duncan swell test. From left to right condition ratings are: 5; 3 and 1

Slake Durability Test

This test assesses the resistance offered by a rock sample to weakening and disintegration when subjected to two standard cycles of drying and wetting. The department had equipment manufactured - which conforms fully to the ISRM guidelines - with four drums thus allowing four samples to be tested at a time.

The slaking fluid used is in all instances water.

The International Standard calls for a representative sample comprising ten rock lumps, each weighing 40-60g. The size of core used by Trans-Natal means that 40-60g lumps can only be obtained from the more competent rock types. If only these rock are tested then the results would be biased towards good floor conditions. For this reason the lump requirement has been modified to 20-30g unprepared lumps (Fig.4).

The drying periods have been shortened from 2-6 hours to 1½-2 hours in order to speed up the procedure and because the lumps are smaller. Fig.5 shows the retained portions after the samples of Fig.4 had been tested.

The slake durability index (second cycle), I_{d2}, is calculated as follows:

$$I_{d2} = \frac{C}{A} \times 100\% \qquad (5)$$

where: A = dry mass prior to testing (g)
C = dry mass after two slaking cycles (g)

Treatment of Results

The brief from the T-Project management team was that the results should be descriptive and unambiguous.

Conventionally a high swell index implies a poor rock, conversely a high slake durability index implies a good rock. To avoid confusion it was decided to present the slake durability index as $100 - I_{d2}$. Both floor indices therefore increase as expected floor conditions get worse.

The more than 250 Duncan swell and slake durability indices were carefully compared. The approach was to rate the various lithologies with regard to their potential to swell or weather based on all availabe information. Ranges of the two indices, with appropriate descriptions, were then chosen to fit the majority of data. The remaining anomalies were then dealt with by fine adjustments to the ranges. The final ranges arrived at are given in Table 2.

Table 2. Swelling and slake durability floor classification.

Rating	Description	Swell index	Slake durability index
A	Good	< 1	< 14
B	Moderate	1 - 3	14 - 26
C	Poor	3.1 - 15	26.1 - 36
D	Very Poor	> 15	> 36

There is not always complete correlation between the two indices. In these circumstances the index suggesting poorer floor conditions dictates the rating.

Each floor is then described to a depth of 0.6m according to the rating and thickness of each unit, e.g. Borehole BN14:A(0.32)/C(0.25)/A. The last layer is usually not given a thickness because it goes beyond 0.6m. Finally the condition of the immediate floor is classified according to Table 3.

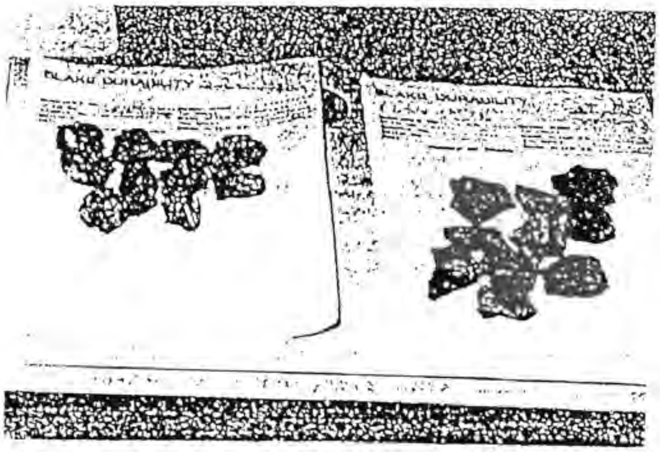
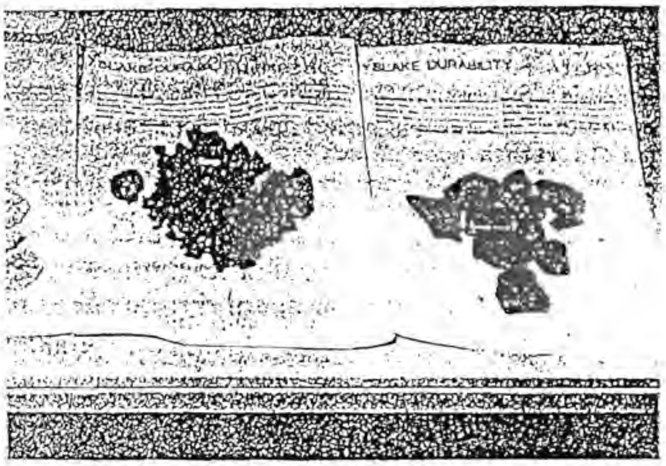


Fig.4. A weathering dolerite and shale prior to slake durability testing



.5. Samples from Fig.4. after testing

Table 3 Floor classification system

Description	Basis of classification
Good	A/B to a depth $\geq$ 0.4m
Possibly Poor	A/B to a depth $<$ 0.4m The first figure in the bracket refers to the thickness of A/B and the second figure to the underlying C/D. e.g. BN14(0.32/0.25)
Poor	C/D in the immediate floor. The figure in the bracket refers to the thickness of C/D. e.g. BN37(0.12)

T-PROJECT

Sadly the T-Project never got off the ground. Had it done so it would have required massive capital investment. Consequently, the feasibility study had to be conducted to a high degree of confidence. Previous experience with the 5 seam floor and to a lesser extent the 5 seam roof meant that rock engineering considerations would play a major part in mine design, equipment selection and contamination.

Since the classification approach used by the Rock Engineering Department (RED) was novel and untried the project management decided to test the classification system against known conditions. Three holes were drilled at the nearby Matla Colliery. The location of these holes was not made known to the RED. Neither was a plan of the location of the exploration boreholes made available. When given the results for the individual boreholes the project management team expressed themselves happy with the classifications assigned to those boreholes with known conditions. Furthermore, the classifications assigned to the exploration holes correlated to other available geological data and made sense when plotted on a plan of the reserves.

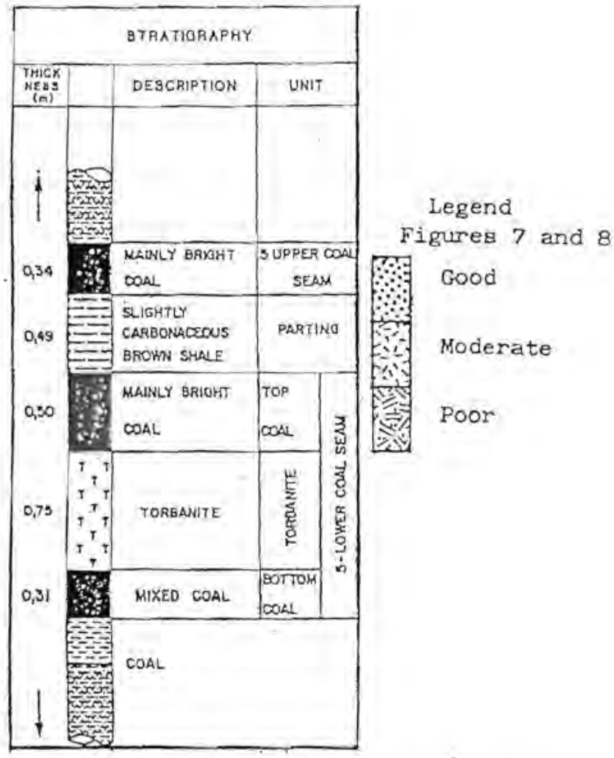


Fig.6. Generalised stratigraphic section, N°5 coal seam, T-Project and legend for figures 7 and 8

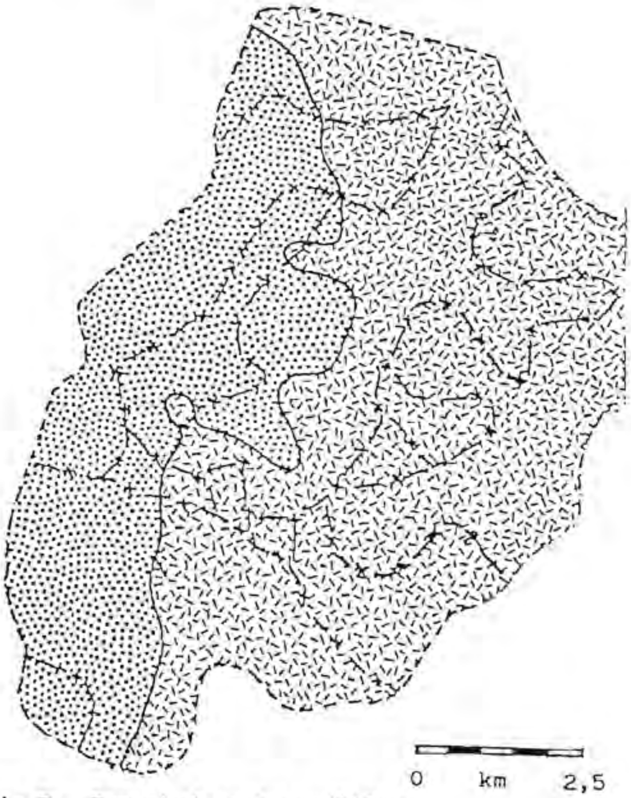


Fig.7. Expected roof conditions, T-Project

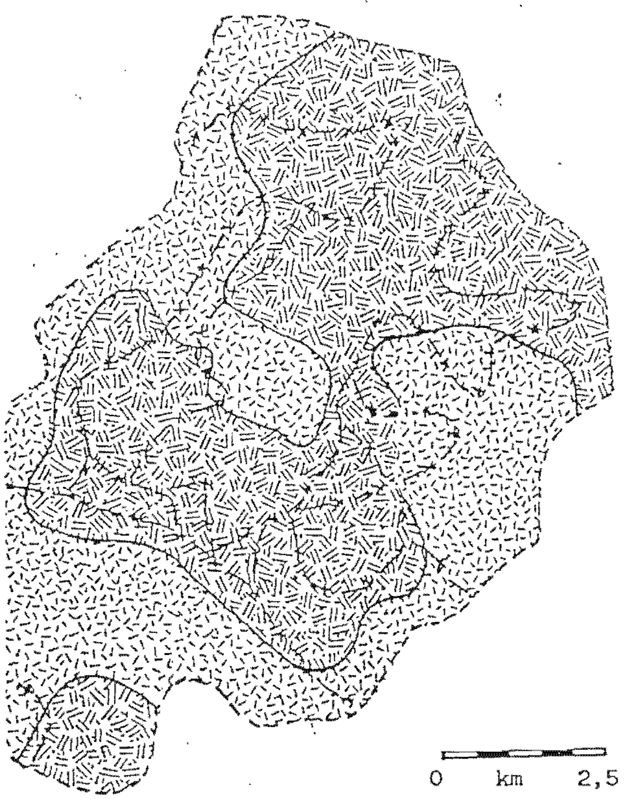


Fig.8. Expected floor conditions, T-Project

The management team chose to reduce the number of classes to two for each of the roof and floor plans. Thus the roof was rated moderate or good and the floor poor or moderate (Figs. 7 and 8).

This information was then applied in a number of ways. After considering all potential mining methods it was decided that longwall mining would be applied in areas where poor coal seam roof and floor conditions existed and the possibility of geological disturbances was minimal. Ribpillar mining would be applied in geologically complex areas where the coal seam roof and floor conditions were manageable. Mechanised bord and pillar mining would be applied in main and secondary entries, areas where surface structures needed to be protected and in mining panels not suited to longwall and ribpillar mining and where the coal seam roof and floor conditions were manageable.

Using the information it was possible to determine expected levels of contamination for the roof and floor. For example, no roof contamination was expected from the longwalls irrespective of the nature of the roof whereas for bord and pillar panels contamination was expected to be 10cm for good roofs and 20cm for moderate roofs. Floor contamination was expected to vary from 5cm for a longwall with a moderate floor to 50cm for bord and pillar panels with a poor floor.

Longwall panels were expected to have a consistent in-panel extraction factor of 92%. For ribpillar panels this was 82% with a moderate floor dropping to 78% with a poor floor.

Although the T-Project was shelved the project management team consider that from a technical perspective the feasibility study was a complete success.

CONCLUSIONS

Since its successful contribution to the T-project the Trans-Natal roof and floor classification system has been applied to further major projects as well as on a much smaller scale. Trans-Natal's mine and project management both view it as an essential tool in the investigation of greenfield sites and mine extensions. It has proven particularly valuable in shaft design (ref.2). The manner of the presentation of the results means that mine and project management are able to envisage the expected conditions in terms of their own experience.

The authors make no claims regarding their classification system other than that it successfully meets the needs of a rock engineering department which is endeavouring t provide a meaningful service to a major coal producer. It is not generally applicable to other minerals and strata types.

REFERENCES

1. ISRM. Suggested methods for determining swelling and slake-durability index properties. Committee on Laboratory Tests document 2, part 2, final draft, Nov 1972.
2. OLDROYD D.C. and BUDDERY P.S. The design and support of inclined shafts through coa measures strata, the use of rock classification.ibid.

ANNEXURE 6

TOTAL COAL HOLDINGS SOUTH AFRICA (PTY) LTD

DORSTFONTEIN COAL MINE

TESTS ON BOREHOLE SAMPLES OF THE PARTING BETWEEN THE 2 LOWER AND UPPER SEAMS

1. INTRODUCTION

Tests on the borehole samples of the parting between the 2 Lower and Upper seams are required to determine:

(i) Whether the parting can be supported using conventional roofbolting methods to allow the safe mining of the lower seam only.
(ii) The feasibility of mining the parting with a continuous miner if it cannot be safely supported.

The following tests were conducted:

(i) Rock mechanics – impact splitter
compressive strength

(ii) Mining – "J" factor (cutting)
"W" factor (wear)

2. ROCK MECHANICS

2.1 Impact Splitter Tests

This test was devised by rock engineering practitioners of the then Genmin group in 1982 and is used throughout the industry and particularly by the Ingwe Group. Roof failure in predominantly governed by the frequency of lamination or bedding planes, their propensity to open and the bord width. The impact splitter causes weak or poorly cemented bedding planes and laminations to open up under duress, thus giving an indication of likely in situ behaviour when subjected to bending stresses.

The rating system requires 2m of strata above the immediate roof to be tested. The borehole core is tested in geotechnical units preferably of about a half a metre in length. A mean fracture spacing for each unit is obtained and an equation used to determine the unit rating and the roof rating.

These were in-house tests. The results of the 3 borehole cores that were tested are:

UNIT POSITION (m)	DF 326		DF 327		DF 322	
	RATING	CLASSIFICATION	RATING	CLASSIFICATION	RATING	CLASSIFICATION
1.5 – 2.0	112.0	Very Good	110.0	Very Good	110.0	Very Good
1.0 – 1.5	110.0	Very Good	110.0	Very Good	110.0	Very Good
0.5 – 1.0	60.0	Very good	24.3	Moderate	21.1	Moderate
0 – 0.5	24.8	Moderate	20.0	Moderate	22.5	Moderate
2m Roof	227	Very Good	175	Very Good	175	Very Good

Despite the weighting of the individual units according to their position in the roof, the very competent upper units results in the overall classifications of the roof being "Very Good". The classification of the lower units that form the first 0,5 to 1,0m of the roof is of greater significance and this zone is classified as "Moderate".

2.2 Uniaxial Compressive Strength Tests

As the name uniaxial compressive strength (UCS) implies, in these tests a load is applied in one direction only with no lateral confinement. In this case the load was applied at right angles or near right angles to the laminae. The results of these tests therefore rather reflect the intrinsic strength of the material and not the strength of the roof when subjected to bending stresses that result in the de-lamination of the roof beam.

These tests were done at CSIR Mingtek. Three specimens from each of two borehole cores were tested. The results are tabulated below.

SPECIMEN PARTICULARS		SPECIMEN DIMENSIONS		TEST RESULTS	
CSIR SPECIMEN No. 2339-	CLIENT No.	DIAMETER (mm)	DENSITY (kg/m^3)	UCS (MPa)	MODE OF FAILURE
UCS – 01	DF 329	60,7	2450	99,2	XA
UCS – 04	DF 329	60,6	2380	95,0	XA
UCS – 05	DF 329	60,6	2450	98,8	XA
UCS – 02	DF 331	60,3	2450	97,3	XA
UCS – 03	DF 331	60,2	2510	110,3	XA
UCS - 06	DF 331	60,2	2470	111,7	XA

XA: Partial cone development

3. MINING

3.1 "J" Factor Tests

This test is used extensively by Joy Mining Machinery to predict the cutting rate of a machine such as a continuous miner. The "J" factor is determined by the controlled drilling of a specimen of the material that is to be cut. The "J" factor is the average depth of 5 holes in millimetres multiplied by 10. Material with "J" factors above 500 can be cut and becomes easier to cut as the number gets larger.

These tests were doneat CSIR Miningtek. Four specimens from one borehole were tested. The results of the tests are tabulated below.

SPECIMEN PARTICULARS		"J" FACTOR			
CSIR No.	CLIENT No.	TEST 1	TEST 2	TEST 3	AVER. + STD.
01 Top	DF328	220	210	204	211,1±8,0
01 Bot.	DF328	282	245	257	261,1±19,2
02 Top	DF328	176	194	188	186,1±9,3
02 Bot.	DF328	186	237	215	212,3±25,6

3.2 "W" Factor Tests

The "W" factor or wear factor is some indication of the pick wear that will result from both the abrasive material in the rock and the manner in which the material is found in the matrix. Bit wear takes place when drilling the holes to determine the "J" factor. This wear expressed in thousands of an inch is the "W" factor. "W" factors range from 0,000 to 0,018, with long life having a "W" factor under 0,003.

These tests were done at CSIR Miningtek. The results of the tests are tabulated below.

SPECIMEN PARTICULARS		"W" FACTOR			
CSIR No.	CLIENT No.	TEST 1	TEST 2	TEST 3	AVER. + STD.
01 Top	DF328	0,0034	0,0039	0,0046	0,0040±0,0006
01 Bot.	DF328	0,0032	0,0034	0,0032	0,0033±0,0001
02 Top	DF328	0,0025	0,0027	0,0027	0,0026±0,0001
02 Bot.	DF328	0,0039	0,0032	0,0049	0,0040±0,0009

4. CONCLUSIONS

4.1 Impact Splitter Tests

The classification of the lower units of the roof as "Moderate" indicates that the roof can be supported provided about 0,4m of the rockbolt can be anchored in the competent sandstone above the 2 Upper seam. This means that a 0,9m long bolt can only be used if the parting and the upper seam together are not more than 0,5m thick. Rows of 4 full column anchored rockbolts every 1,5 m will be required. A reduction in the density may be possible but that will depend on observations of favourable roof behaviour over a period of time.

4.2 Uniaxial Compressive Strength Tests

The UCS of the specimens tested varied between 95.0 and 111,7 MPa. As stated in section 2.1, this test does indicate the ability of the parting to withstand the de-laminating bending stresses that occur in the roof. What it does indicate is that this material can be transformed into a competent beam if de-lamination is prevented by clamping the layers together.

4.3 "J" Factor Tests

"J" factors of between 282 and 176 indicate that the parting will be difficult to cut. Rock with "J" factors below 500 can be cut if the rock is highly laminated or fractured and provided high operating costs can be tolerated.

4.4 "W" Factor Tests

The results show that pick life will be greatly reduced as the majority of the "W" factors are greater than 0,003.

M G SPENGLER

ANNEXURE 7

Sensitivity

Operating Costs	0.00%
Selling Price (Export)	0.00%
Selling Price (Domestic)	0.00%
Yield	0.00%
Production	0.00%
Capital Expenditure	0.00%

			Dorstfontein									
Period		**Total**	**2,003**	**2,004**	**2,005**	**2,006**	**2,007**	**2,008**	**2,009**	**2,010**	**2,011**	**2,012**
Production	**T000's**											
Run of mine		3,514	314	377	377	377	377	377	377	314	314	314
Yield		85.2%	85.2%	85.2%	85.2%	85.2%	85.2%	85.2%	85.2%	85.2%	85.2%	85.2%
Coal produced		2,993	267	321	321	321	321	321	321	267	267	267
Operating costs	**R/ton**											
Run of mine		104.37	82.08	82.91	87.89	93.16	98.75	104.68	110.96	123.41	130.82	138.67
Produced		122.54	96.36	97.35	103.19	109.38	115.94	122.90	130.27	144.90	153.59	162.81
Distribution costs	**R/ton**											
Railage costs		50.56	38.62	40.94	43.40	46.00	48.76	51.69	54.79	58.08	61.56	65.26
Port charges		9.33	7.13	7.56	8.01	8.49	9.00	9.54	10.11	10.72	11.36	12.05
Transport costs		20.71	15.82	16.77	17.78	18.85	19.98	21.18	22.45	23.79	25.22	26.74
Total sales	**T000's**	**2,993**	**267**	**321**	**321**	**321**	**321**	**321**	**321**	**267**	**267**	**267**
Export		1,773	158	190	190	190	190	190	190	158	158	158
Inland sales		1,220	109	131	131	131	131	131	131	109	109	109
Export selling price												
Export selling price	$/ton	$31.64	$27.77	$27.77	$27.77	$27.77	$27.77	$27.77	$27.77	$27.77	$27.77	$27.77
Exchange rate	$/R	8.5000	8.5000	8.7500	9.0074	9.2723	9.5450	9.8257	10.1147	10.4122	10.7184	11.0337
Export selling price	R/ton	268.98	236.05	242.99	250.13	257.49	265.06	272.86	280.89	289.15	297.65	306.41
Inland selling price	**R/ton**	**216.01**	**165.03**	**174.93**	**185.43**	**196.55**	**208.35**	**220.85**	**234.10**	**248.15**	**263.03**	**278.82**
Turnover	**R000's**	**740,441**	**55,345**	**69,027**	**71,757**	**74,608**	**77,588**	**80,703**	**83,960**	**72,804**	**75,772**	**78,876**
Export revenue		476,990	37,374	46,168	47,526	48,924	50,362	51,844	53,368	45,782	47,128	48,514
Inland revenue		263,450	17,971	22,859	24,231	25,685	27,226	28,859	30,591	27,022	28,643	30,362
Distribution costs		**163,579**	**11,328**	**14,361**	**15,173**	**16,033**	**16,942**	**17,905**	**18,924**	**16,668**	**17,619**	**18,626**
Railage costs		89,652	6,116	7,779	8,246	8,740	9,265	9,821	10,410	9,196	9,747	10,332
Port costs		16,549	1,129	1,436	1,522	1,613	1,710	1,813	1,922	1,697	1,799	1,907
Transport costs		36,732	2,506	3,187	3,378	3,581	3,796	4,024	4,265	3,768	3,994	4,233
Marketing fee - Inland	1.5%	3,952	270	343	363	385	408	433	459	405	430	455
Marketing fee - Export	3.5%	16,695	1,308	1,616	1,663	1,712	1,763	1,815	1,868	1,602	1,649	1,698
FOR revenue		**576,862**	**44,017**	**54,666**	**56,584**	**58,576**	**60,646**	**62,798**	**65,036**	**56,136**	**58,152**	**60,250**
Operating costs		**365,681**	**25,343**	**31,203**	**33,030**	**35,012**	**37,112**	**39,339**	**41,699**	**38,577**	**40,954**	**43,412**
Operating costs		366,769	25,751	31,217	33,090	35,075	37,180	39,410	41,775	38,721	41,044	43,506
Stock Movement		-1,087	-408	-14	-59	-64	-67	-71	-76	-143	-90	-94
Operating profit		**211,180**	**18,674**	**23,463**	**23,553**	**23,564**	**23,534**	**23,459**	**23,337**	**17,558**	**17,198**	**18,838**
Taxation		28,983	139	4,534	5,084	2,190	4,662	5,146	4,091	0	0	3,138
Profit after taxation		**182,197**	**18,535**	**18,930**	**18,469**	**21,374**	**18,872**	**18,313**	**19,246**	**17,558**	**17,198**	**13,700**
Working capital incr/(decr)		4,627	3,382	623	135	140	146	151	157	-424	156	161
Capital expenditure		114,569	18,210	8,351	6,607	16,265	7,994	6,306	9,700	23,844	11,737	5,554
Nominal Cash flow		**63,001**	**-3,057**	**9,956**	**11,728**	**4,969**	**10,732**	**11,856**	**9,388**	**-5,862**	**5,305**	**7,985**
Real Cash Flow		**46,302**	**-2,884**	**8,861**	**9,847**	**3,936**	**8,020**	**8,358**	**6,244**	**-3,678**	**3,140**	**4,459**

IRR:	**305.2%**		**-2,884**	**8,861**	**9,847**	**3,936**	**8,020**	**8,358**	**6,244**	**-3,678**	**3,140**	**4,459**
Discount rate		0.0%	2.5%	5.0%	7.5%	10.0%	12.5%	15.0%	17.5%	20.0%	22.5%	25.0%
Net Present Value		46,302	41,878	38,078	34,790	31,926	29,417	27,206	25,246	23,501	21,939	20,536

			Dorstfontein									
Period		Total	2,003	2,004	2,005	2,006	2,007	2,008	2,009	2,010	2,011	2,012
Tax Computation												
Tax loss		0	0	0	0	0	0	0	0	0	-6,285	-824
Operating profit		211,180	18,674	23,463	23,553	23,564	23,534	23,459	23,337	17,558	17,198	16,838
Capital expenditure		114,569	18,210	8,351	6,607	16,265	7,994	6,306	9,700	23,844	11,737	5,554
Taxable Profit		96,611	464	15,112	16,947	7,299	15,540	17,153	13,636	-6,285	-824	10,459
Tax payable	30%	28,983	139	4,534	5,084	2,190	4,662	5,146	4,091	0	0	3,138
<u>Working Capital</u>												
Stocks			987	1,000	1,060	1,124	1,191	1,262	1,338	1,482	1,571	1,666
Stores (4 weeks op costs)	2		990	1,201	1,273	1,349	1,430	1,516	1,607	1,489	1,579	1,673
Debtor (6 weeks)	4		4,257	5,310	5,520	5,739	5,968	6,208	6,458	5,600	5,829	6,067
Creditor (4 weeks all costs)	4		2,852	3,506	3,713	3,931	4,163	4,409	4,669	4,261	4,513	4,779
Net Current Asset/(Liabilities)			3,382	4,005	4,140	4,280	4,426	4,577	4,734	4,310	4,466	4,627
Opening Balance			0	3,382	4,005	4,140	4,280	4,426	4,577	4,734	4,310	4,466
Yearly Movement			3,382	623	135	140	146	151	157	-424	156	161

ANNEXURE 8

12-Aug-03			Dorstfontein thin seam									
Year			**2003**	**2004**	**2005**	**2006**	**2007**	**2008**	**2009**	**2010**	**2011**	**2012**
Period			**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
Escalated capital expenditure												
Underground												
Wirth Machine			15,000,000	-	-	-	-	-	-	-	-	-
Stamler Hauler			-	4,400,000	-	-	-	-	-	-	-	-
Ventilation			15,000	16,500	18,150	19,965	21,962	48,315	53,147	58,462	64,308	70,738
Telemetrey			120,000	-	-	-	-	-	-	-	-	-
Concrete roads			240,000	-	-	-	-	-	-	-	-	-
Extraordinary support			100,000	110,000	121,000	133,100	146,410	161,051	177,156	194,872	214,359	235,795
Conveyor belt and structure			900,000	1,100,000	1,210,000	1,331,000	1,464,100	1,610,510	1,771,561	1,948,717	2,143,589	2,357,948
Pumps and accessories			78,000	85,800	94,380	103,818	114,200	125,620	138,182	152,000	167,200	183,920
Roof Brushing			250,000	275,000	302,500	332,750	366,025	402,628	442,890	487,179	535,897	589,487
Electrical distribution			105,000	115,500	127,050	139,755	153,731	169,104	186,014	204,615	225,077	247,585
CM Overhaul			-	-	-	10,648,000	-	-	-	15,589,737	-	-
Equipment overall			-	1,155,000	3,630,000	1,996,500	4,392,300	2,415,765	5,314,683	2,923,076	6,430,766	-
Sub total - underground			**16,808,000**	**7,257,800**	**5,503,080**	**14,704,888**	**6,658,727**	**4,932,992**	**8,083,633**	**21,558,657**	**9,781,196**	**3,685,472**
Surface												
Overland conveyor			90,000	99,000	108,900	119,790	131,769	144,946	159,440	175,385	192,923	212,215
Infrastructure			90,000	99,000	108,900	119,790	131,769	144,946	159,440	175,385	192,923	212,215
Environmental			60,000	66,000	72,600	79,860	87,846	96,631	106,294	116,923	128,615	141,477
Strategic spares			90,000	99,000	108,900	119,790	131,769	144,946	159,440	175,385	192,923	212,215
Sub total - surface			**330,000**	**363,000**	**399,300**	**439,230**	**483,153**	**531,468**	**584,615**	**643,077**	**707,384**	**778,123**
Processing												
Plant & Laboratory modifcation			66,000	72,600	79,860	87,846	96,631	106,294	116,923	128,615	141,477	155,625
Discard dump			111,000	122,100	134,310	147,741	162,515	178,767	196,643	216,308	237,938	261,732
Slurry pond			45,000	49,500	54,450	59,895	65,885	72,473	79,720	87,692	96,461	106,108
Strategic spares			150,000	165,000	181,500	199,650	219,615	241,577	265,734	292,308	321,538	353,692
Sub total - processing			**372,000**	**409,200**	**450,120**	**495,132**	**544,645**	**599,110**	**659,021**	**724,923**	**797,415**	**877,157**
Sub total			**17,510,000**	**8,030,000**	**6,352,500**	**15,639,250**	**7,686,525**	**6,063,570**	**9,327,269**	**22,926,657**	**11,285,995**	**5,340,752**
Capex fees @ 4%			700,400	321,200	254,100	625,570	307,461	242,543	373,091	917,066	451,440	213,630
Total capital expenditure			**18,210,400**	**8,351,200**	**6,606,600**	**16,264,820**	**7,993,986**	**6,306,113**	**9,700,359**	**23,843,723**	**11,737,435**	**5,554,382**

12-Aug-03			Dorstfontein thin seam									
Year			2003	2004	2005	2006	2007	2008	2009	2010	2011	2012
Period			1	2	3	4	5	6	7	8	9	10
Capital Expenditure	**R000's**											
Unescalated capital expenditure												
Underground												
Wirth Machine			15,000,000	-	-	-	-	-	-	-	-	-
Stamler Hauler			-	4,000,000	-	-	-	-	-	-	-	-
Ventilation			15,000	15,000	15,000	15,000	15,000	30,000	30,000	30,000	30,000	30,000
Telemetrey			120,000	-	-	-	-	-	-	-	-	-
Concrete roads			240,000	-	-	-	-	-	-	-	-	-
Extraordinary support			100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000
Conveyor belt and structure			900,000	1,000,000	1,000,000	1,000,000	1,000,000	1,000,000	1,000,000	1,000,000	1,000,000	1,000,000
Pumps and accessories			78,000	78,000	78,000	78,000	78,000	78,000	78,000	78,000	78,000	78,000
Roof Brushing			250,000	250,000	250,000	250,000	250,000	250,000	250,000	250,000	250,000	250,000
Electrical distribution			105,000	105,000	105,000	105,000	105,000	105,000	105,000	105,000	105,000	105.000
CM Overhaul			-	-	-	8,000,000	-	-	-	8,000,000	-	-
Equipment overall			-	1,050,000	3,000,000	1,500,000	3,000,000	1,500,000	3,000,000	1,500,000	3,000,000	-
Sub total - underground			**16,808,000**	**6,598,000**	**4,548,000**	**11,048,000**	**4,548,000**	**3,063,000**	**4,563,000**	**11,063,000**	**4,563,000**	**1,563,000**
Surface												
Overland conveyor			90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000
Infrastructure			90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000
Environmental			60,000	60,000	60,000	60,000	60,000	60,000	60,000	60,000	60,000	60,000
Strategic spares			90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000
Sub total - surface			**330,000**	**330,000**	**330,000**	**330,000**	**330,000**	**330,000**	**330,000**	**330,000**	**330,000**	**330,000**
Processing												
Plant & Laboratory modifcation			66,000	66,000	66,000	66,000	66,000	66,000	66,000	66,000	66,000	66,000
Discard dump			111,000	111,000	111,000	111,000	111,000	111,000	111,000	111,000	111,000	111,000
Slurry pond			45,000	45,000	45,000	45,000	45,000	45,000	45,000	45,000	45,000	45,000
Strategic spares			150,000	150,000	150,000	150,000	150,000	150,000	150,000	150,000	150,000	150,000
Sub total - processing			**372,000**	**372,000**	**372,000**	**372,000**	**372,000**	**372,000**	**372,000**	**372,000**	**372,000**	**372,000**
Sub total			**17,510,000**	**7,300,000**	**5,250,000**	**11,750,000**	**5,250,000**	**3,765,000**	**5,265,000**	**11,765,000**	**5,265,000**	**2,265,000**
Capex fees @ 4%			700,400	292,000	210,000	470,000	210,000	150,600	210,600	470,600	210,600	90,600
Total capital expenditure			**18,210,400**	**7,592,000**	**5,460,000**	**12,220,000**	**5,460,000**	**3,915,600**	**5,475,600**	**12,235,600**	**5,475,600**	**2,355,600**

ANNEXURE 9

			Dorstfontein thin seam									
Year			2003	2004	2005	2006	2007	2008	2009	2010	2011	2012
Period			1	2	3	4	5	6	7	8	9	10
Operating Costs	R000's											
Cash costs - Unescalated			25,753	29,451	29,451	29,451	29,451	29,451	29,451	25,753	25,753	25,753
Mining contractor costs	R/ton		50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00
Mining contractor costs	R000's		15,688	18,825	18,825	18,825	18,825	18,825	18,825	15,688	15,688	15,688
Outbye costs	R000's		300	300	300	300	300	300	300	300	300	300
Repair and maintenance	R000's		2,241	2,241	2,241	2,241	2,241	2,241	2,241	2,241	2,241	2,241
Other underground costs	R000's		570	570	570	570	570	570	570	570	570	570
Plant costs	R/ton		7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50
Plant costs	R000's		2,353	2,824	2,824	2,824	2,824	2,824	2,824	2,353	2,353	2,353
Laboratory & Weighbridge	R000's		436	436	436	436	436	436	436	436	436	436
ROM stockpile	R000's		189	189	189	189	189	189	189	189	189	189
Product stockpile	R000's		486	486	486	486	486	486	486	486	486	486
Service costs	R000's		609	609	609	609	609	609	609	609	609	609
Safety and training	R000's		159	159	159	159	159	159	159	159	159	159
Utility costs	R000's		1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200
Other costs	R000's		893	893	893	893	893	893	893	893	893	893
Operating fee 2,5%	R000's		630	720	720	720	720	720	720	630	630	630
Cash costs - Escalated	R000's		25,751	31,217	33,090	35,075	37,180	39,410	41,775	38,721	41,044	43,506
Mining cost			15,688	19,955	21,152	22,421	23,766	25,192	26,704	23,588	25,003	26,504
Outbye costs			300	318	337	357	379	401	426	451	478	507
Repair and maintenance			2,241	2,375	2,518	2,669	2,829	2,999	3,179	3,370	3,572	3,786
Other underground costs			570	604	640	679	720	763	809	857	908	963
Plant costs			2,353	2,993	3,173	3,363	3,565	3,779	4,006	3,538	3,751	3,976
Laboratory & Weighbridge			436	462	490	520	551	584	619	656	695	737
ROM stockpile			189	200	212	225	239	253	268	284	301	319
Product stockpile			486	515	546	579	614	650	689	731	775	821
Service costs			609	645	684	725	768	815	863	915	970	1,028
Safety and training			159	169	179	189	201	213	226	239	253	269
Utility costs			1,200	1,272	1,348	1,429	1,515	1,606	1,702	1,804	1,913	2,027
Other costs			893	946	1,003	1,063	1,127	1,195	1,266	1,342	1,423	1,508
Operating fee 2,5%			628	761	807	855	907	961	1,019	944	1,001	1,061

			Dorstfontein thin seam									
Year			2003	2004	2005	2006	2007	2008	2009	2010	2011	2012
Period			1	2	3	4	5	6	7	8	9	10
Cash costs - R/ton ROM			**82.08**	**82.91**	**87.89**	**93.16**	**98.75**	**104.68**	**110.96**	**123.41**	**130.82**	**138.67**
Mining cost			50.00	53.00	56.18	59.55	63.12	86.91	70.93	75.18	79.69	84.47
Outbye costs			0.96	0.84	0.90	0.95	1.01	1.07	1.13	1.44	1.52	1.62
Repair and maintenance			7.14	6.31	6.69	7.09	7.51	7.97	8.44	10.74	11.38	12.07
Other underground costs			1.82	1.60	1.70	1.80	1.91	2.03	2.15	2.73	2.90	3.07
Plant costs			7.50	7.95	8.43	8.93	9.47	10.04	10.64	11.28	11.95	12.67
Laboratory & Weighbridge			1.39	1.23	1.30	1.38	1.46	1.55	1.64	2.09	2.22	2.35
ROM stockpile			0.60	0.53	0.56	0.60	0.63	0.67	0.71	0.91	0.96	1.02
Product stockpile			1.55	1.37	1.45	1.54	1.63	1.73	1.83	2.33	2.47	2.62
Service costs			1.94	1.71	1.82	1.93	2.04	2.16	2.29	2.92	3.09	3.28
Safety and training			0.51	0.45	0.47	0.50	0.53	0.57	0.60	0.76	0.81	0.86
Utility costs			3.82	3.38	3.58	3.80	4.02	4.27	4.52	5.75	6.10	6.46
Other costs			2.85	2.51	2.66	2.82	2.99	3.17	3.36	4.28	4.54	4.81
Operating fee 2,5%			2.00	2.02	2.14	2.27	2.41	2.55	2.71	3.01	3.19	3.38
Cash costs - R/ton produced			**96.36**	**97.35**	**103.19**	**109.38**	**115.94**	**122.90**	**130.27**	**144.90**	**153.59**	**162.81**
Mining cost			58.70	62.23	65.96	69.92	74.11	78.56	83.27	88.27	93.57	99.18
Outbye costs			1.12	0.99	1.05	1.11	1.18	1.25	1.33	1.69	1.79	1.90
Repair and maintenance			8.39	7.41	7.85	8.32	8.82	9.35	9.91	12.61	13.37	14.17
Other underground costs			2.13	1.88	2.00	2.12	2.24	2.38	2.52	3.21	3.40	3.60
Plant costs			8.81	9.33	9.89	10.49	11.12	11.78	12.49	13.24	14.03	14.88
Laboratory & Weighbridge			1.63	1.44	1.53	1.62	1.72	1.82	1.93	2.45	2.60	2.76
ROM stockpile			0.71	0.62	0.66	0.70	0.74	0.79	0.84	1.06	1.13	1.19
Product stockpile			1.82	1.61	1.70	1.81	1.91	2.03	2.15	2.73	2.90	3.07
Service costs			2.28	2.01	2.13	2.26	2.40	2.54	2.69	3.42	3.63	3.85
Safety and training			0.59	0.53	0.56	0.59	0.63	0.66	0.70	0.89	0.95	1.01
Utility costs			4.49	3.97	4.20	4.46	4.72	5.01	5.31	6.75	7.16	7.59
Other costs			3.34	2.95	3.13	3.32	3.51	3.73	3.95	5.02	5.32	5.64
Operating fee 2,5%			2.35	2.37	2.52	2.67	2.83	3.00	3.18	3.53	3.75	3.97

ANNEXURE 10

			Dorstfontein thin seam									
Year			**2003**	**2004**	**2005**	**2006**	**2007**	**2008**	**2009**	**2010**	**2011**	**2012**
Period			**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
Escalation Rates												
Deflator			6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%
Deflator factor			1.060	1.124	1.191	1.262	1.338	1.419	1.504	1.594	1.689	1.791
S.A. PPI	%		0.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
S.A. PPI Growth Factor			1.000	1.050	1.103	1.158	1.216	1.276	1.340	1.407	1.477	1.551
US CPI	%		0.00%	2.00%	2.00%	2.00%	2.00%	2.00%	2.00%	2.00%	2.00%	2.00%
US CPI Growth Factor			1.000	1.020	1.040	1.061	1.082	1.104	1.126	1.149	1.172	1.195
Dollar Selling price	%		0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Dollar Selling price Growth Factor			1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Inland Selling price	%		0.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%
Inland Selling price Growth Factor			1.000	1.060	1.124	1.191	1.262	1.338	1.419	1.504	1.594	1.689
ESKOM	%		0.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%
ESKOM			1.000	1.060	1.124	1.191	1.262	1.338	1.419	1.504	1.594	1.689
Operating Costs	%		0.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%
Operating Costs Growth Factor			1.000	1.060	1.124	1.191	1.262	1.338	1.419	1.504	1.594	1.689
Railage Costs	%		0.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%
Railage Costs Growth Factor			1.000	1.060	1.124	1.191	1.262	1.338	1.419	1.504	1.594	1.689
Port charges	%		0.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%
Port charges Growth Factor			1.000	1.060	1.124	1.191	1.262	1.338	1.419	1.504	1.594	1.689
Transport costs	%		0.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%
Transport costs Growth Factor			1.000	1.060	1.124	1.191	1.262	1.338	1.419	1.504	1.594	1.689
Capital Expenditure	%		0.00%	10.00%	10.00%	10.00%	10.00%	10.00%	10.00%	10.00%	10.00%	10.00%
Capex Growth Factor			1.000	1.100	1.210	1.331	1.464	1.611	1.772	1.949	2.144	2.358