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The financial viability of coal reserves within previously mined areas of the Witbank Coalfield.

by

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ABSTRACT

The future of coal mining in the Witbank Coalfield over the next 30 years and beyond depends on effective and responsible utilization of the remaining reserves, both within unmined and previously mined areas. Similar to all mineral resources, coal is also non-renewable and the current resources will not last forever. Unlike most other resources coal resources have to be considered in long term strategic planning for energy supply. It has therefore become very important to use the remaining resources and reserves to their full potential. This has prompted mining companies to re-mine or do secondary extraction of areas mined during the previous 50 years.

Reliable and internationally accepted valuation techniques and reporting standards are well established for virgin areas. The challenge is now to develop an equally robust and reliable system for remaining resources and reserves in previously mined areas. A number of established operations already exist in South Africa and internationally which are utilizing such reserves.

Due to numerous factors affecting the viability of this type of operation a system or matrix is proposed for defining such resources and reserves. This classification scheme caters for the obvious geological, mining and beneficiation factors, and also for the multitude of lesser known but equally important factors.

The effects of some of these factors on a future mining operation are demonstrated in a case study of such a previously mined area.

Factors affecting the Run of Mine (ROM) tons and saleable tons are:

a) derating percentage
b) percentage mining extraction
c) percentage dilution and contamination
d) percentage fines generated
e) spontaneous combustion

Numerous pitfalls are identified such as top coaled areas, water accumulations, no access to old areas to verify existing information and the time lapsed since previous mining occurred. Another complicating factor is the lack of a method for the quantification of the impact of spontaneous combustion on remaining reserves.

The financial viability of mining these areas are especially sensitive to the coal price, R/$ exchange rate, change in production and capital expenditure. The information generated during the investigation is processed in a financial model which is used to evaluate different scenarios and risk sensitivities.

It is demonstrated that in evaluating previously mined areas, it is not the obvious factors that often determine the financial viability of a project, but rather the not so obvious financial factors.

1. INTRODUCTION

1.1. LIST OF DEFINITIONS

In this treatise certain terms would be used which have a very specific meaning in the coal mining industry. To avoid misunderstanding these terms are defined here for the benefit of the reader. Commonly used abbreviations for some of these terms are also introduced here.

Arial extraction: is the percentage extraction in plan view, based on the dimensions of the remnant coal, pillars and mined out bord widths. This can usually be calculated relatively accurately from the existing survey plans.

Buffer zone: is a mining term used for the area blasted between the current mining cut and the unmined intact old bord and pillar workings. The main reasons for creating a buffer zone are to reduce the likelihood for spontaneous combustion, for additional highwall support and to create a stable platform for machinery to move on.

Capping: is the hard unblasted non-coal material that remain on top of the coal seam after the dragline has passed that specific area. It was left because it was too hard and solid for the dragline to remove. This is normally the result of too shallow drilling or an ineffective overburden or interburden blast.

Cladding: is the term used for covering blasted material with a thin ~0.5m to 1.0m layer of pre-strip soft material. The main reason for this is to prevent spontaneous combustion by limiting air movement and ingress of oxygen and creating a better surface to move machinery on.

Coaling: is the term used to describe the activity which extracts the coal from the mining pit and transports it to the tip.

Contamination: is extraneous coal and non-coal material unintentionally added to the planned mining horizon as a result of mining operations.

Derating: is the mechanism used to compensate for the percentage primary extraction that has already taken place. It entails derating or decreasing the different mining horizons by the percentage coal already extracted. This derating percentage is applied to the Gross tons *in situ* (GTIS) to get the Total tons *in situ* (TTIS) number.

Dilution: is non-coal or coal outside the theoretical mining height that is intentionally added as part of the planned mining section to obtain a practical mining horizon.

Geological loss: is a discount factor applied to the TTIS to take into account as yet unobserved geological features that may reasonably be expected to occur between points of observation. Examples of geological losses are localized thinning of the coal seam, weathering, faulting, dyke and sill intrusions, areas of burnt or devolatilised coal and others. (derived from SANS, 2003)

Gradecon: is an in-house developed software program that is used to apply mining and plant parameters to the theoretical geological model data. This is done to obtain a practical ROM tonnage and product yield.

Gross tons *in situ* (GTIS): is the tonnage contained in the full coal seam above the minimum thickness cut-off and relevant coal quality cut-offs. No loss factors have been applied yet. (derived from SANS, 2003)

Layout loss: This factor accounts for the expected loss of coal reserves due to actual mining activities not reaching the defined boundary of the mineable *in situ* coal resource block. It includes planned mining layout losses at cut extremities in the case of opencast mining operations.

Mineable tons *in situ* (MTIS): is the tonnage contained in coal seams or sections of seams, which are proposed to be mined at the theoretical mining height after the minimum and maximum mineable thickness cut-offs have been applied.

Mining extraction loss: is coal not extracted, by not reaching the planned theoretical mining limits or by not reaching the planned aerial mining extraction percentage. In the case of opencast mining operations this includes unplanned losses between adjacent mining cuts.

Mining loss: is a discount factor that is used to account for the net losses of coal reserves due to the mining method's inefficiency.

Primary extraction: this is the first pass mining operation which only partially extracted the reserve, leaving a remnant resource for possible secondary extraction.

Prodint: forms part of the Gradecon software program and is the product interpretation or plant wash simulation portion. The program will calculate the optimal cut-point density that a specific coal should be washed at to achieve a pre-determined product quality.

Reserve: is the economically mineable coal derived from a measured or indicated coal resource, or both. Coal reserves are sub-divided in order of increasing confidence into probable coal reserves and proved coal reserves. A coal reserve is based on an evaluation that demonstrates that the extraction of a coal resource is justified at the time of the valuation and that an economic mine plan has been defined. (derived from SANS, 2003 and SAMREC, 2000)

Resource: is an occurrence of coal of economic interest in such form, quality and quantity that there are reasonable and realistic prospects for eventual economic extraction. Coal resources are subdivided, in order of increasing geological confidence, into *inferred, indicated* and *measured* categories. (derived from SAMREC, 2000)

Run of Mine (ROM): is the as delivered tonnage of coal at a specific quality, mined from the *in situ* coal reserves. It is also the expected tonnage to be recovered after all geological losses, dilution, mining losses, contamination and moisture content factors have been applied to the mineable *in situ* coal reserves.

Safety factor: refers to the ratio of strength to load imposed on coal pillars. This ratio must be such that the underground pillars are large enough to stabilize the ground above them without being too large and unnecessarily sterilizing coal. (Van der Merwe, 1995)

Scalping: is when the dragline removes mineable coal together with the overburden. This is usually due to poor digging standards, poor visibility or too deep overburden drilling and blasting.

Secondary extraction: is the second pass mining operation either again partially or totally extracting the remnant reserve. There can be a significant time span between the primary and secondary operations.

Strip ratio: refers to the ratio between the volume of overburden and interburden required to be removed in order to access a ton of Run of Mine (ROM) coal. This can also be calculated on the basis of saleable coal tons. This term is used in open cast mining situations.

Top coaling: is the term used for the secondary mining operation that took place in the underground workings where an additional slice of the roof coal was extracted after the initial primary extraction.

Total tons *in situ* (TTIS): is the tonnage contained in the full coal seam above the minimum thickness cut-off and the relevant coal quality cut-offs. But now the geological loss factors are applied to the tonnage. (derived from SANS, 2003)

Volumetric extraction: is the percentage extraction in three dimensions and relies on both the aerial extraction and the mining heights of the extracted area. This is problematic because of the sparse information regarding mining heights. The derating numbers are very dependant on this number.

Yield: is the saleable product tons from the beneficiation plant divided by the ROM feed tons to the beneficiation plant, expressed as a percentage. The basis on which the product yield is calculated is quoted, and shall be preferably on an air-dried basis.

1.2. OVERVIEW

The South African coal industry has been growing over the last 130 years (Barker *et al*, 1999). In the Witbank Coalfield as shown in Figure 1.1 this has resulted in vast areas of partially extracted coal seams. Most of these mines used bord and pillar methods for underground coal extraction, resulting in coal left in the old workings and other remnants still remaining.

This treatise will focus on a future reserve block as a case study of Kleinkopje Colliery, an existing opencast mining operation. The case study area is known as the North West block and forms part of the Kleinkopje Colliery life of mine plan. It is planned to be mined from 2014 onwards to the end of life in 2025. This area was previously mined as part of the old Landau Colliery which was later incorporated into the current South African Coal Estates (SACE) which comprise 3 separate mines; Landau Colliery, Kleinkopje Colliery and Greenside Colliery as shown in the locality map in Figure 1.2 – South African Coal Estates Collieries. Mining started in the SACE area in 1895 when the Cassel Coal Company opened Landau Colliery to supply coal to the gold mines (SACE Brochure, 2000).

The area of interest is the old underground Landau I and II blocks where mining commenced in 1901 followed by opencast mining in 1942 (Kleinkopje Colliery, IWULA, 2002). Underground mining ceased in the Landau I and II areas in 1950, but continued in Landau III to the south until 1991. Kleinkopje Colliery was planned during 1976 and production commenced by 1978 (Barker *et al*, 1999). The mine was originally planned as a conventional opencast mine and only later the life of mine was extended by adding the old underground working reserves. The North West block falls within the old Landau II mined out area. During 1976 to 1979 and again from 1988 to 1989, mini pit operations took place on the northern boundary of this block. Although mining is currently only planned to commence in 2014 (Kleinkopje Colliery, FYFC, 2004), it was decided to evaluate this block in current terms. This is done for comparative strategic considerations and to serve as a template for future evaluations of similar resources remaining in areas after primary extraction.

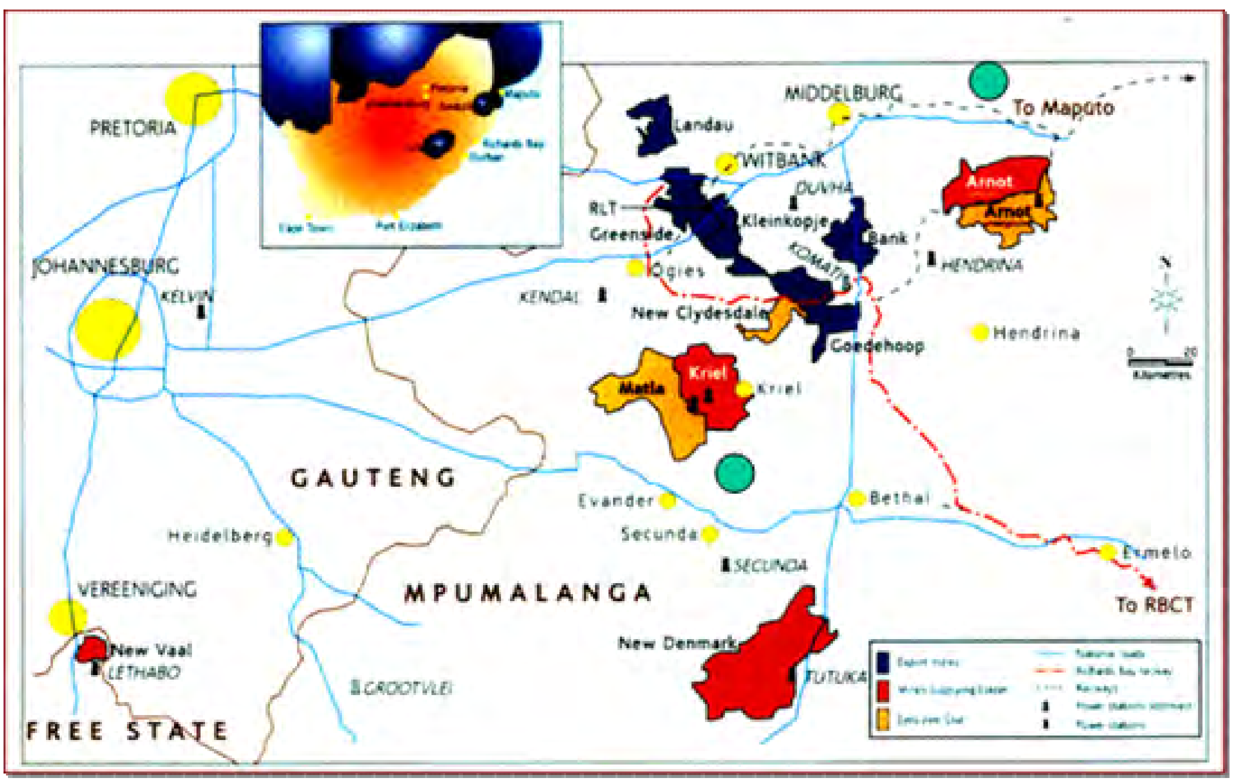

Figure 1.1 - Location

(Source: Kleinkopje Colliery, Geology Department)

In the shallower portions of the Landau I mining area shown in Figure 1.2 there are areas that have collapsed and are burning in places, sinkholes have also developed within the Landau II area including parts of the study area. An additional negative factor in this area was the secondary mining of roof coal (top coaling), which reduced the overall ground stability.

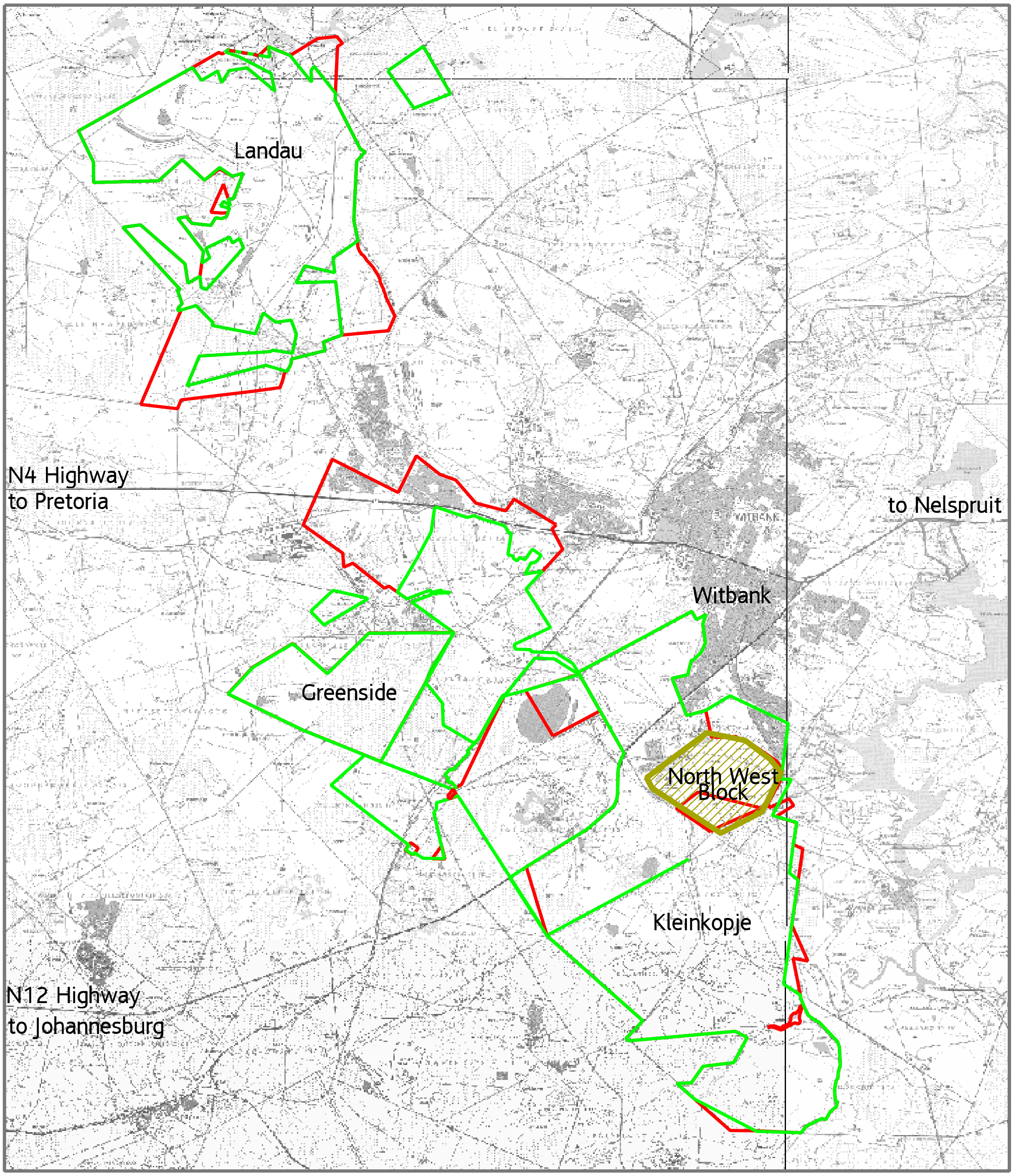

Figure 1.2 – South African Coal Estates Collieries

(Source: Kleinkopje Colliery Survey Department)

1.3. THE PROBLEM AND ITS SETTINGS

Due to coal reserves being depleted continuously through ongoing mining operations, new or additional coal resources are required. The previously mined areas, consisting of pillars, remnant coal left in the roof and floor, are now being utilized with secondary extraction methods utilizing opencast mining techniques. Prior to mining a “new” reserve block needs to be evaluated, first from a reserve perspective, then the mineability and finally the financial viability, which will also include aspects such as the marketing and possible beneficiation of the coal.

However this poses unique problems associated with secondary mining of old workings:

a) The arial distribution and shapes of the remnant pillars, coal in the roof and floor as well as the size and heights of the void make <u>resource and reserve estimation very difficult</u>. Old mine survey plans are not always compatible with current survey programs and grids, resulting in shifts or inaccuracies. To transfer the data from old cloth plans to an electronic format is time consuming and difficult. Due to the inaccessibility of these workings they cannot be re-surveyed. Mining heights, bord widths and roof and floor elevations are few and far between. Workings mined earlier than 1967 did not use the safety factor concept, and may have very small pillars, leaving less coal and eventually resulting in unstable roof conditions. The safety factor concept and its origin will be covered in section 3.6.2.1. Most of these areas were selectively mined for either sized coal for the inland market, or early exports of high grade low ash coals. These areas have better coal qualities and due to the depletion of the coal reserves have become very sought after.

b) The old workings have, over time, filled with water in the low lying areas. The water has to be pumped out before mining can take place. Unfortunately the quality of the water deteriorated and is now high in sulphates and has a low pH, preventing it from being added to clean surface water systems. Treatment is very costly but will soon become the only remaining option. A proper water management strategy is a crucial requirement for the mining of old areas.

c) Another problem associated with opencast mining of old workings is spontaneous combustion of the coal and carbonaceous shales. This is common in the Witbank area and causes environmental, mining and quality problems. In order to manage this, the mining method had to be changed to incorporate blasting of buffer zones, soil cladding, water cannons and sealing of hot holes and vents. There must also be good co-ordination between the dewatering program and the mining extraction plan as dewatered areas may become prone to spontaneous combustion.

d) As a result of the spontaneous combustion the mining operation creates a lot of dust from ash, steam and smog formed due to temperature differences and this causes safety, health and environmental concerns.

e) Environmental management also needs to be of a high standard to monitor the water, dust, vibration and noise impacts. The close proximity to Witbank's residential areas place additional pressure on operational standards and the monitoring required to comply with national and local standards.

f) The geological modelling and daily grade control in pit is crucial to the success of this type of operation. The effective beneficiation of this coal for the export market relies on the geological modeling, grade control and blending of the ROM coal. The mining operation causes extensive contamination of the ROM coal with overburden, highwall and low wall material. The mining practice has to guard against this and minimize it because of the detrimental effect on the overall yield. Reconciliation of the initial geological predicted coal reserve with the actual coal mined, hauled, tipped, beneficiated and final coal product tons and quality is essential. Only then can there be an effective system of estimation and utilization of secondary coal reserves.

1.4. HYPOTHESES

Due to the long history of mining in the Witbank Coalfield most of the high quality reserves have already been partially extracted and only the pillars remain today. The secondary mining of these old workings will result in less coal being available for extraction compared to the coal available in similar unmined areas. The advantage of the old workings is the high quality coal still remaining in the pillars. The benefit from this higher quality coal in the remaining pillars makes this type of mining potentially viable. However there are certain factors that can impact negatively on this type of mining such as contamination, fines generation, coal losses, spontaneous combustion and water accumulations. The financial viability of this type of operation will therefore depend on the coal reserve, actual mining method employed, mine planning and the execution of this plan.

1.5. METHOD OF INVESTIGATION

The viability of secondary mining in the Witbank coalfield will be studied through an analysis of a case study. The case study area is a separate block of coal forming part of the current Kleinkopje Colliery life of mine plan, as indicated in Figure 1.3 (Kleinkopje Business Plan, 2006). However, this area is only planned to be mined from ~2014 to 2025. Due to its close proximity to the residential area and the ongoing risk of ground collapses due to the pillar dimensions and age, the decision should be reviewed and this block should be considered for earlier mining.

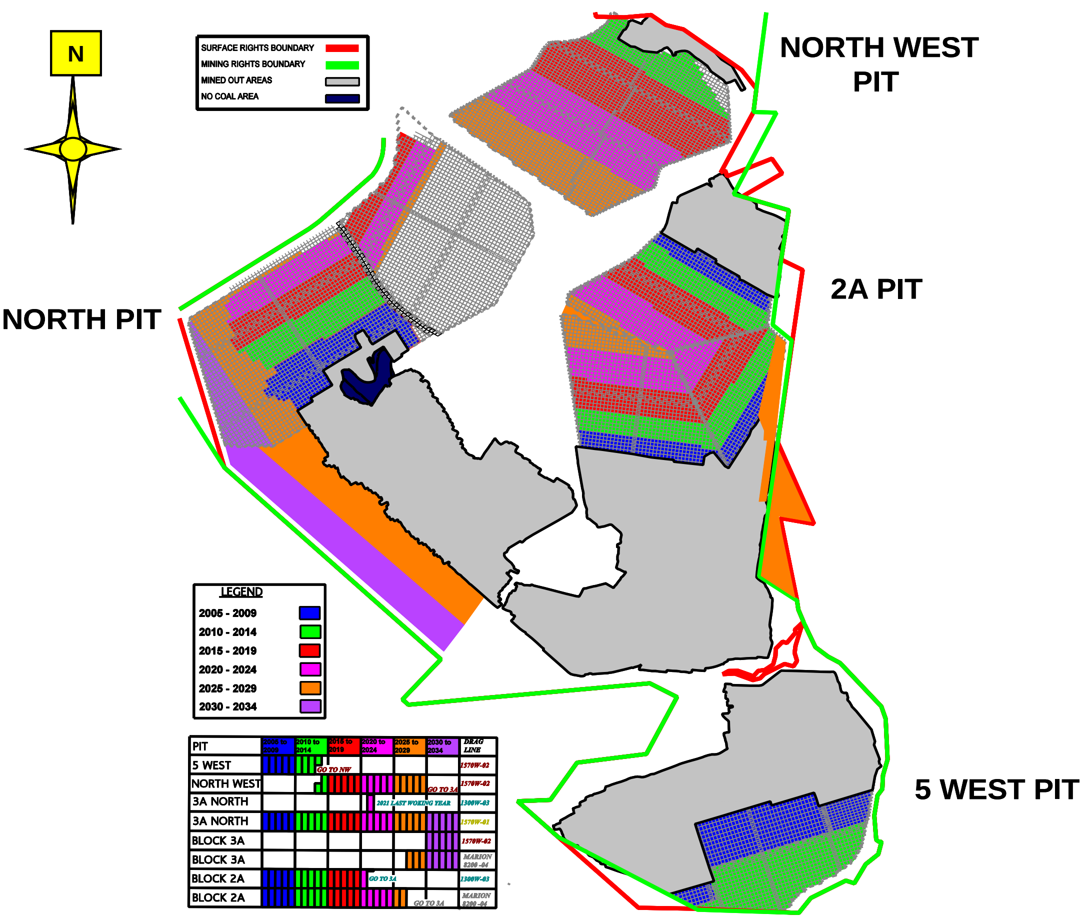

Figure 1.3 – The North West block within the Kleinkopje life of mine plan

(Source: Kleinkopje Colliery Survey Department)

1.5.1. RESEARCH METHODOLOGY

a) The *current geological model parameters* will be reviewed and adapted to ensure that it provides an accurate estimation of the size and quality of the resource.

b) The *derating* values used or applied in the study area will be verified: especially due to the extensive degree of top coaling in the past.

c) *Dilution and contamination* factors will be reviewed.

d) The *ground stability/geotechnical data* for this block will be briefly discussed, with reference to other areas, such as Landau I, II and III.

e) *Financial viability* at forecast and current coal prices will be done. The impact of various factors such as derating, contamination, extraction percentage and percentage fines will be evaluated.

f) *The risks associated with mining this specific area* will be identified/listed and addressed in a sensitivity analysis.

1.5.2. THE RISK MATRIX

One of the objectives of this study is to define a matrix of all the factors that could affect the viability of a previously mined area, and rating and weighting them based on historical experiences in other operations.

This matrix could become a useful tool for future application to any other similar resource evaluation exercise and will assist in standardizing the derating criteria and values being applied.

1.5.3. THE FINANCIAL MODEL

The financial viability of this area as a separate reserve block will be assessed using a discounted cash flow model. Strategic considerations are addressed by doing different scenarios and a sensitivity analysis is done on the base case scenario. The saleable ton strip ratio is the major deciding factor.

1.6. DELIMITATIONS

There might be different mining options available, but only the opencast methods will be discussed in this study. Depth to the top of the mineable coal horizon, the coal seam thickness, MTIS and coal qualities are the main criteria for determining the feasibility of a mining block. Other important considerations will be factors associated with the secondary mining of old workings such as, percentage derating and extraction, delineation of top coaled areas, higher contamination and spontaneous combustion.

From this it will be possible to determine

a) Reserve estimates

b) Mining conditions and constraints

c) Beneficiation parameters

d) Financial viability

This treatise will focus on the geology pertaining to this specific area and less on that of the surrounding or wider region. Similar criteria will be used in this assessment as are currently applied in the Kleinkopje Colliery's annual budget and five year forecast process. However additional factors will be highlighted and the effect of each individual impact will be assessed in more detail.

1.7. ASSUMPTIONS

This area will only be considered for opencast mining methods. The current Kleinkopje Colliery mining practices will be used as guideline to its estimation, extraction and beneficiation.

1.8. ACKNOWLEDGEMENTS

Professor H. Theart for his guidance through this treatise.

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A special word of thanks to Michèle for her continued support and valuable contribution and our children Bertus and Jeanne for their patience.

2. GEOLOGY AND MINING OF THE WITBANK COALFIELD

2.1. INTRODUCTION

The coal seams in the Witbank Coalfield form part of the Permian Vryheid Formation, which hosts the majority of the economic coal reserves in South Africa (Cairncross and Cadle, 1987).

The stratigraphy of the Vryheid Formation within the Witbank Coalfield is reasonably consistent. The distribution of the coal seams are closely associated with the original Pre-Karoo basement topography and the current surface topography, both of which affected the coal seam thickness, quality and lateral extent (Snyman, 1998).

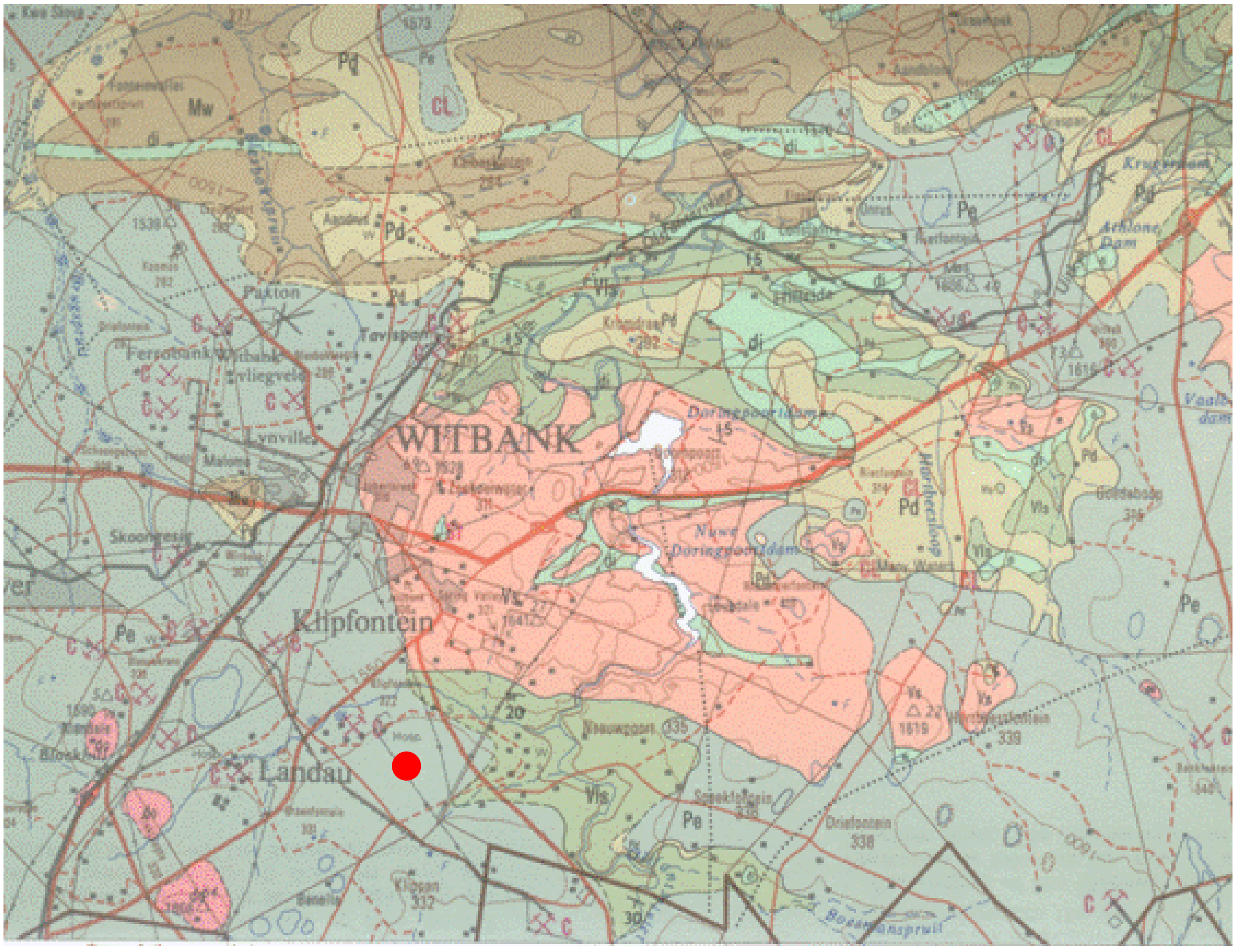
This coal field has been and still remains the main source of coal for both the export and inland market in South Africa. The average life of mines in this area is only 20 years, according to leaders in the industry (Mining Weekly Volume 11, No.21). It is thus critical to the country's economy that the remaining resources and reserves be utilized effectively and responsibly.

2.2. STRATIGRAPHY

An igneous Pre-Karoo basement, consisting of felsite and diabase associated with the Bushveld Igneous Complex, underlies the study area (refer to Study Area Figure 2.1 below). Glacial Dwyka tillite and the Vryheid Formation of the Karoo Supergroup rests unconformably on the basement.

The coal seams are numbered from the bottom upwards as in Figure 2.2, starting at No.1 Seam and ending with No.5 Seam. The No.1 Seam either lies directly above the Dwyka diamictite or above a fining upward sequence of glaciofluvial grit and sandstone. The Sandstone parting between the No.1 and No.2 seams is formed by a fining upward sequence of grit and sandstone with subordinate shale beds, typical of sediment formed within a braided fluvial environment. The No.2 Seam is overlain by a transgressive, generally upwards coarsening, sequence consisting of carbonaceous shale, bioturbated siltstone and fluvial sandstones. This sequence is overlain by the No.3 seam. Then follows another unit of fluvial sandstone overlain by the No.4 Seam which has a thin shale roof layer followed by further fluvial sediments.

The detailed coal seam zoning, yield and quality variants, will be addressed in section 2.6 as part of the grade control activities.



FORMATION

		Formation	
QUARTERNARY			Surface deposit; alluvian
JURASSIC	Karoo sequence	Letaba	Volcanic rocks, sandstone
TRIASSIC		Clarens	Fine-grained sandstone
PERMIAN		Irrigasie	Multi-coloured siltstone, sandstone, marl, mudstone, shale
		Ecca	Shale, shaly sandstone, grit, sandstone, conglomerate, coal in places near base and top
		Dwyka	Tillite, shale

Dolerite

Basalt, pyroclasts

Study Area

Figure 2.1 - Geological Map

(Adapted from the 1:250 000 Geological Map Series, 2528 Pretoria Sheet)

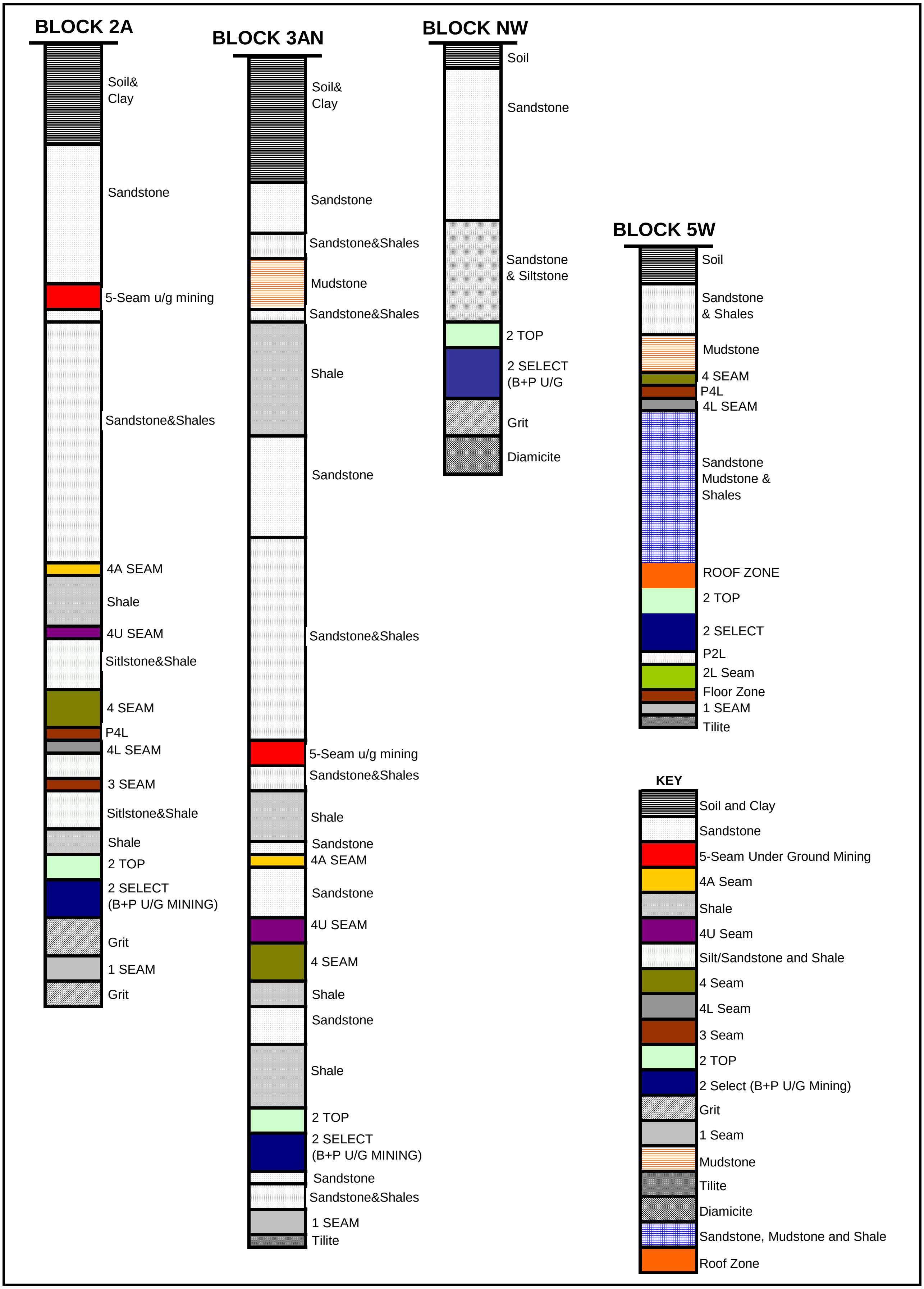


Figure 2.2 - Stratigraphic Column (Adapted from Kleinkopje Business Plan 2006)

2.3. EXPLORATION

The current exploration methods employed consist of fully cored and partially cored rotational drilling. This is followed by sampling of the coal seams, directed at the different coal quality zones within the seams. Borehole spacing is governed by the SAMREC code requirements for reserve and resource classification.

2.4. RESERVES AND RESOURCES

Numerous exercises have been done to accurately estimate the remaining coal reserves in South Africa. The first of these was done in 1913 with the more recent studies in 1998 (Barker *et al*, 1999). Although they vary significantly it still indicated clearly that the reserves are limited and that at the current rate of mining we are probably looking at between 130 to 150 years of remaining reserves from 2005 onwards.

This limitation in remaining resources highlights the importance for urgent exploration of new fields, especially lower quality coal resources, and research to develop new improved techniques to utilize the existing coal deposits.

During 2001/2002 the total number of collieries decreased from 60 to 55 (South Africa’s Mineral Industry 2001/2002), however this number has increased again to 64 currently (South Africa’s Mineral Industry 2004/2005). It is not evident whether this is due to the depletion of reserves or other constraints. At the same time the total ROM tons produced increased, but the saleable tonnage decreased. This is possibly an indication of a decrease in the quality of the remaining reserves.

The relationship between ROM tons and saleable tons is summarized in Table 2.1.

Table 2.1 - Relationship between ROM and Saleable tons

	2000	2001	2002	2003	2004	Variance
ROM	283,0 Mt	290,0 Mt	285,0 Mt	303,0 Mt	307,0 Mt	+ 4,0 Mt
Saleable	224,1 Mt	223,5 Mt	220,2 Mt	239,3 Mt	242,8 Mt	+ 5,0 Mt

(South Africa's Mineral Industry 2004/2005)

The current coal production per province is as given in Table 2.2:

Table 2.2 – Coal production per province in 2004

Province	Number of mines	Percentage of production
Mpumalanga	56	80%
Free State	2	7%
Limpopo	2	11%
KwaZulu Natal	4	1%
TOTAL	**64**	**100%**

(South Africa's Mineral Industry 2004/2005)

This demonstrates clearly that the Witbank Coalfield still produces most (>80%) of the export coal. Estimates show that this coalfield has approximately 11,3 Bt available which at current mining rates of approximately 156Mt per annum will last for another 72 years. However some experts only allow for 40 years life.

This re-emphasizes the importance of utilizing the current resources efficiently, and the need to do secondary extraction of financially viable previously mined areas. It is also important to maximize the extraction of high quality coal from existing mines, by pillar extraction methods, both in opencast and underground scenarios.

2.5. MINING METHODS

The mining methods employed generally depends on the depth of the coal deposits. When feasible, shallower deposits can be mined by opencast techniques. The deeper coal seams have historically been mined using underground bord and pillar and longwall mining methods.

Due to the emphasis on improved extraction and the depletion of good quality reserves, there has been an increased focus on pillar extraction using underground mining methods. However this method still raises concerns for safety and it has to prove itself first, current underground trials are underway at Goedehoop Colliery. There are only a few operations mining old underground workings by means of opencast mining methods. The best known operations are listed in Table 2.3 below.

Table 2.3 - Operations Mining Old Underground Workings

Date of Commencement	Operation	Company/Owner
1980	New Vaal Colliery	Anglo Coal
1995	Landau Colliery	Anglo Coal
1997	Kleinkopje Colliery	Anglo Coal
1999	Douglas Colliery (Boschmanskrans section)	Ingwe Coal

The Anglo Coal opencast operations make use of a method where the historic bord and pillar mined out area is collapsed, while the Ingwe operation only collapse the bords and keep the pillars intact. These methods are related to the mining equipment that will be used, either a dragline or a truck and shovel operation.

2.5.1. BORD AND PILLAR COLLAPSE METHOD

This surface mining method has been employed by Anglo Coal for more than 20 years. The main objective of this method is to fully collapse the pre-existing underground excavations in order to create a stable platform for the heavy dragline to operate on. In this case, a drilling pattern is designed to fit on the pre-existing underground excavations. Both the pillars and the bords are drilled, charged and blasted to collapse the total area. The pillar holes are positioned towards the centre of the pre-existing pillar and drilled to the base of the coal seam. The bord holes are drilled to just above the old roof elevation into the roof coal or shale, not holing into the old workings as seen in Figure 2.3.

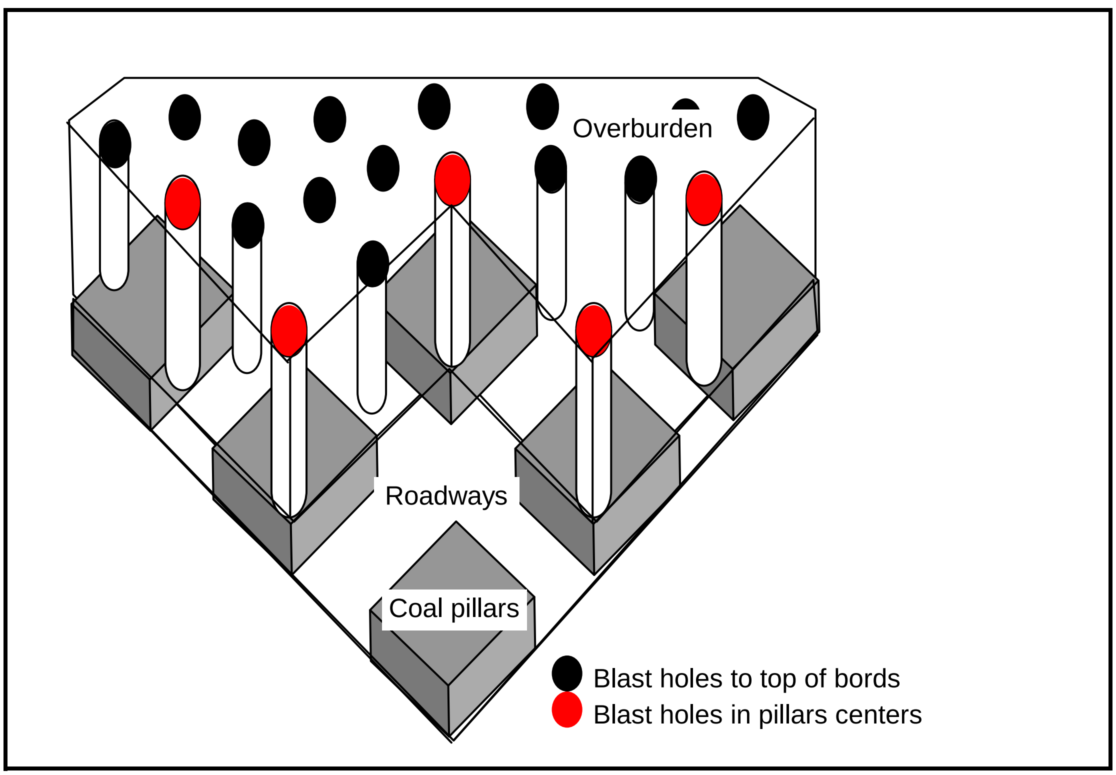


Figure 2.3 - Mining Method - Bord and Pillar Collapse Method

(Source: Eroglu and Moolman, 2003)

Depending on the accuracy of the old survey information the success rate is generally very high at 90% to 95% (CSIR Report, 19). However spontaneous combustion, rat holing, time related and blast induced collapses of the intersection roof, cause unplanned holing into the old workings. In the case of hot areas (due to underground combustion) the holed blastholes are plugged, abandoned and re-drilled. The temperature of each individual blasthole is measured and recorded, because the blasting standard prevents holes above 60^0C to be charged for safety reasons.

It is of utmost importance that both the pillar and bord is collapsed by the blast. This ensures that all the cavities are filled by the broken material and forms a good seal preventing air movement and limiting the chances of spontaneous combustion. The finer the material fragmentation after the blast, the better the seal to prevent air movement is (Crosby, 2000).

Current drilling and blasting practices are given in the table below (Table 2.4).

Table 2.4 - Current Drilling and Blasting Practices

Pre-split	Production blasting
(311mm diameter)	**(311mm hole diameter)**
Drilled to base of coal seam	Burden and spacing dependant on pillar centres
Spacing 5m	Hole in pillar down to top of P1
70kg Anfo per hole	Hole above bord drilled to shale
Instantaneous blast	Gas bag at coal contact in pillar hole
	4-5m stemming

(Kleinkopje Colliery report, Buffer blasting Pit 2A South)

After the overburden or interburden, (depending on whether No.4 Seam is mineable) have been blasted, a dragline is used to remove the broken material. The dragline will stand on the blasted material and dig down to the top of coal contact. It is important to note that the previously mined No.2 Seam coal surface is not a hard surface anymore as it was also shattered by the blast. This makes the digging to the correct surface or elevation difficult, remembering that the dragline is required to dig to the correct horizon in order to prevent a capping on the coal or scalping of the coal. Once the dragline removed the interburden material, the coaling operation can commence, using rope shovels or large front end loaders and haul trucks.

This mining method results in a relatively high degree of dilution due to the bord material being mixed with the coal. All this material is loaded out together and transported to the beneficiation plant. The plant hence needs to be designed to handle all the additional dilution or waste material. This will be covered in more detail in Chapter4, dealing with beneficiation.

2.5.1.1. BUFFER BLASTING

Buffer blasting is a technique employed to blast the material *in situ* without creating any movement or shift. This causes the material to heave upwards and forces the overlying material into the previously mined out voids between pillars. The coal pillars can be either shattered with the interburden or they can be left intact depending on the preferred mining method.

There are three main reasons for doing confined buffer blasting:

a) To stabilize area above old workings

b) Improve the highwall stability

c) To control spontaneous combustion, by limiting air movement

Figure 2.4 shows an example of what the highwall looked like before buffer blasting was implemented. It shows the intact coal pillar and bord and unblasted interburden along a vertical highwall. Also note the barrels of the pre-split holes on the highwall. The drilling and blasting of a pre-split down to the No.2 Seam old workings was stopped when buffer blasting was initiated at the Kleinkopje Block 2A Pit. This was to reduce the ingress of oxygen and air circulation from the highwall into the workings and up through the pre-split holes which would increase the likelihood for spontaneous combustion to occur.

Figure 2.5 illustrates an ideal buffered zone and battered back highwall. This is after the drilling and blasting of the interburden of the buffer and after the dragline exposed the No.2 Seam in the current mining cut. Following on from this point will be the cladding of the buffered zone and the coaling of the No.2 Seam.

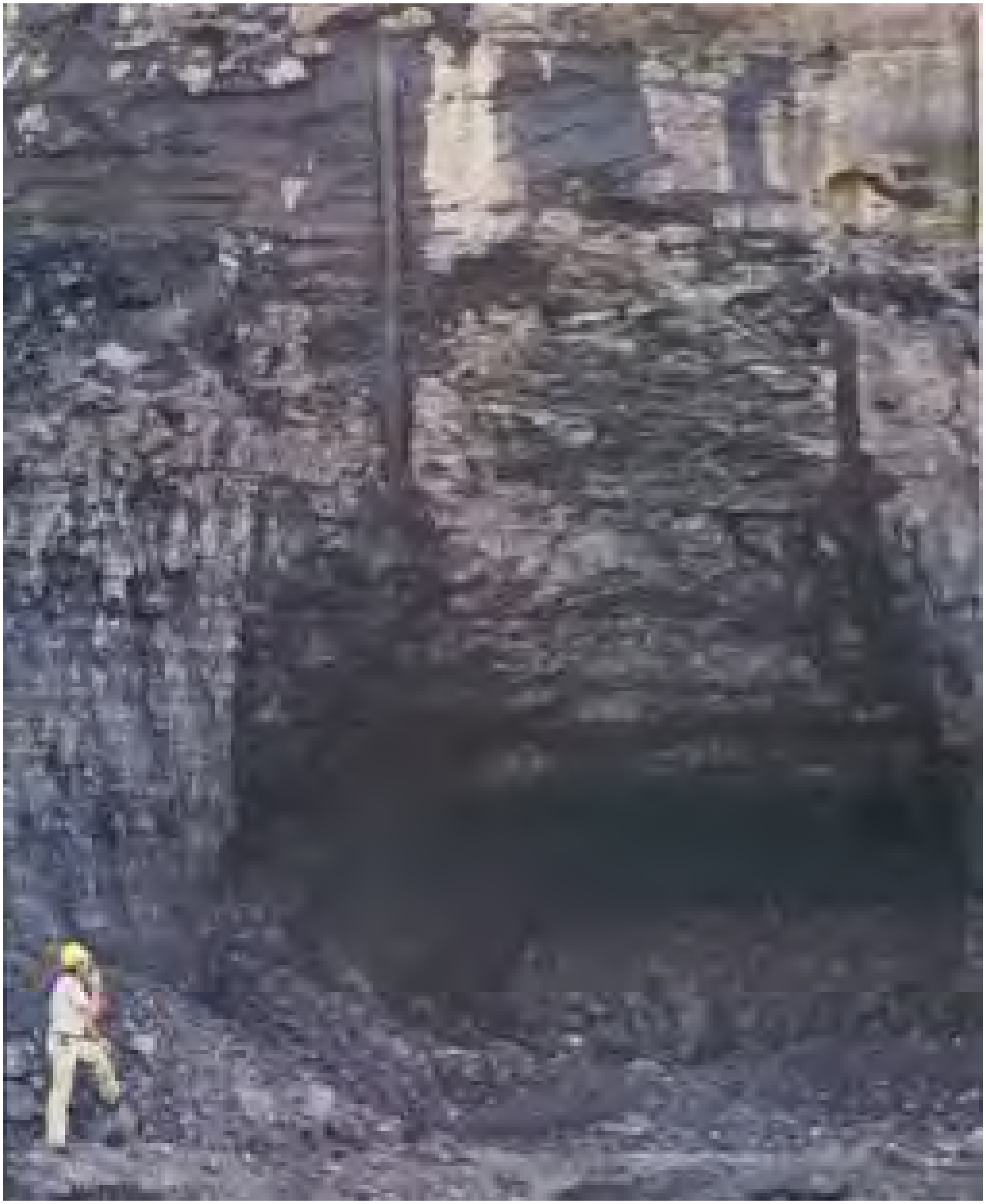


Figure 2.4 - Photo of Highwall pre-buffer blasting

(Source: AMCOAL Annual Report, 1996)

Figure 2.5 - Photo of Highwall post-buffer blasting

It is important to note the approximately 45 ^{0} angle (blue dashed line) of the battered highwall. This creates a stable highwall and improves the effectiveness of the cladding material.

The previous voids of the mined out areas are now closed up by the interburden and the coal pillars collapsed and blasted into the voids. The buffer blasted surface is levelled by a dozer and then cladded with pre-strip material using shovels, dumper trucks and dozer machines, to load, haul and level cladding material. This seals the surface and prevents spontaneous combustion of the buffered zone and the adjacent underground workings.

The end product is a relative safe surface for men and machine to move on and provides a good seal to manage spontaneous combustion. The buffered cut is created adjacent to the currently mined cut. This closes all the previously excavated openings, and thereby reducing the ingress of air and oxygen into the old workings. Buffer blasting comes at a cost to mining productivity and efficiency:

a) No throw on the interburden blast resulting in more material for the dragline to move
b) Results in tighter digging conditions, the diggability is reduced
c) This reduces the dragline productivity significantly
d) Less fragmentation of material resulting in reduced seal and difficult digging conditions

2.5.1.2. CLADDING

This is the procedure where soft pre-strip material is placed on top of the buffer zone to clad or seal it as illustrated in Figure 2.6. Cladding is also placed on the battered highwall to prevent air movement through the fractured material as shown in Figure 2.7. The main reason for cladding is to assist in the reduction and control of spontaneous combustion.

In order to prevent spontaneous combustion one may either remove the fuel (coal), or the heat, or the air (oxygen). In this case, the first option is not possible, because that is what is mined, the second is very difficult to do, because of the size of the affected area. Water cannons are used at the dragline face and at other hot spots, to reduce the heat.

Currently the best option is to do cladding, to try and reduce the airflow into or through the buffer and starve the fire of oxygen. This practice also confines the products of combustion, namely CO, CO_2 and other gasses inside this zone, which further inhibits combustion.

Figure 2.6 - Photo of buffer being cladded

Figure 2.7 - Photo of buffer post levelling and cladding

Negative aspects related to cladding include:

a) Additional material to load and haul

b) Need to be levelled by a dozer

c) Need to be applied at a correct thickness

d) It adds to the dragline re-handle volume

e) Also adds to contamination of the coal in the current mining cut

f) It is time consuming and impacts on the availability of equipment for the other mining activities and therefore needs to be integrated in the mining schedule

From the above it is obvious that cladding is an activity that requires proper planning and managing to ensure the benefit from it.

2.5.2. BORD ONLY COLLAPSE METHOD

The second mining method only collapse the bords and intersections leaving the coal pillars intact as in Figure 2.8. The blastholes are only drilled up to the roof of the coal seam. When this is blasted, it forces all the roof material into the bords or voids between the pillars creating a stable environment.

The interburden material is removed not by dragline, but instead using large truck and shovel equipment. Due to the thickness of the interburden it needs to be removed in benches compared to a single dragline operation in the Bord and Pillar Collapse Method.

The drill and blast method does not require the identification of the pillar position. A smaller blast pattern with closer spacing and smaller hole diameter is used.

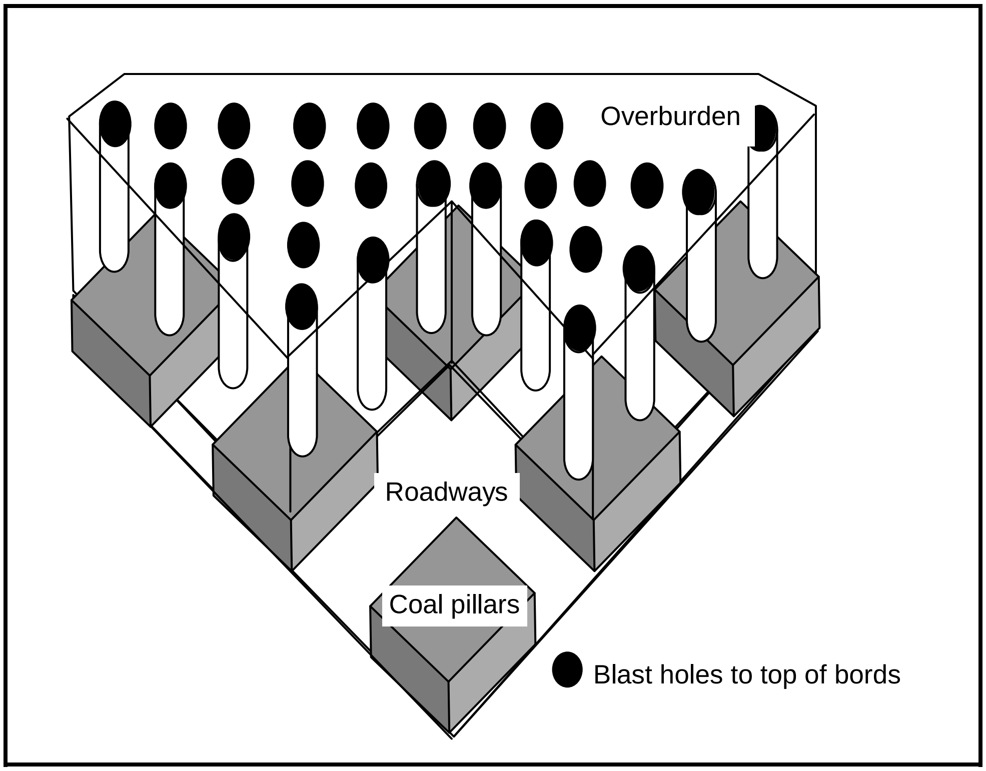


Figure 2.8 - Mining Method - Bord Only Collapse Method

(Source: Eroglu and Moolman, 2003)

Table 2.5 shows a comparison between the two mining methods.

Table 2.5 - Comparison between mining methods

Bord and pillar collapse	Bord only collapse
Stable buffer surface for dragline	Undulating and unstable surface not suitable for dragline
Use dragline to move overburden	Use truck and shovel equipment
Risk of capping	Less risk of capping
Risk of scalping	Less risk of scalping
High dilution with coal	Low dilution with coal
More waste hauled to plant	Less waste hauled to plant
Plant cost to handle waste	Less plant cost
More waste/discard generated for dump	Less discard
Less sensitive mining methods	More sensitive
Less small equipment used	More small equipment required
Single bench operation with dragline	Multiple bench operation without dragline
No hard pillars left	Hard pillars restrict movement
Fast coal exposure	Slower coal exposure

2.6. COAL YIELDS AND QUALITIES

This section addresses the importance of coal yield and quality data in the planning and mining of an operation. Due to the inherent variability of coal quality within a specific reserve area, coal seam, or coal zone, it requires selective mining and blending of coal to satisfy the marketing requirements.

The No.2 Seam is the main economically mineable horizon for this project. Other important coal seams are the No.4 Seam and No.1 Seam which are thinner but previously unmined. The No.2 Seam is generally 6 metres thick, but varies between 5 and 8 metres. The seam thickness is mostly affected by the pre-Karoo basement topography, localized floor undulations and in seam stone partings. Within the No.2

Seam we find vertical and horizontal quality variations which require separate sampling and analysis of the different in-seam quality zones.

2.6.1. NO.2 SEAM ZONING

Historically, geologists have recognized four distinct zones as shown in Figure 2.9, based on the visual difference in the amount of bright (vitrinite rich) coal (Ortlepp and Ackhurst, 1976). Zones 1 and 3 (Z1; Z3), containing most of the bright coal, have a low ash content and some coking properties. Zones 2 and 4 (Z2; Z4) consist of mainly dull-lustrous coal. These zones are generally regionally persistent and fairly uniform in grade.

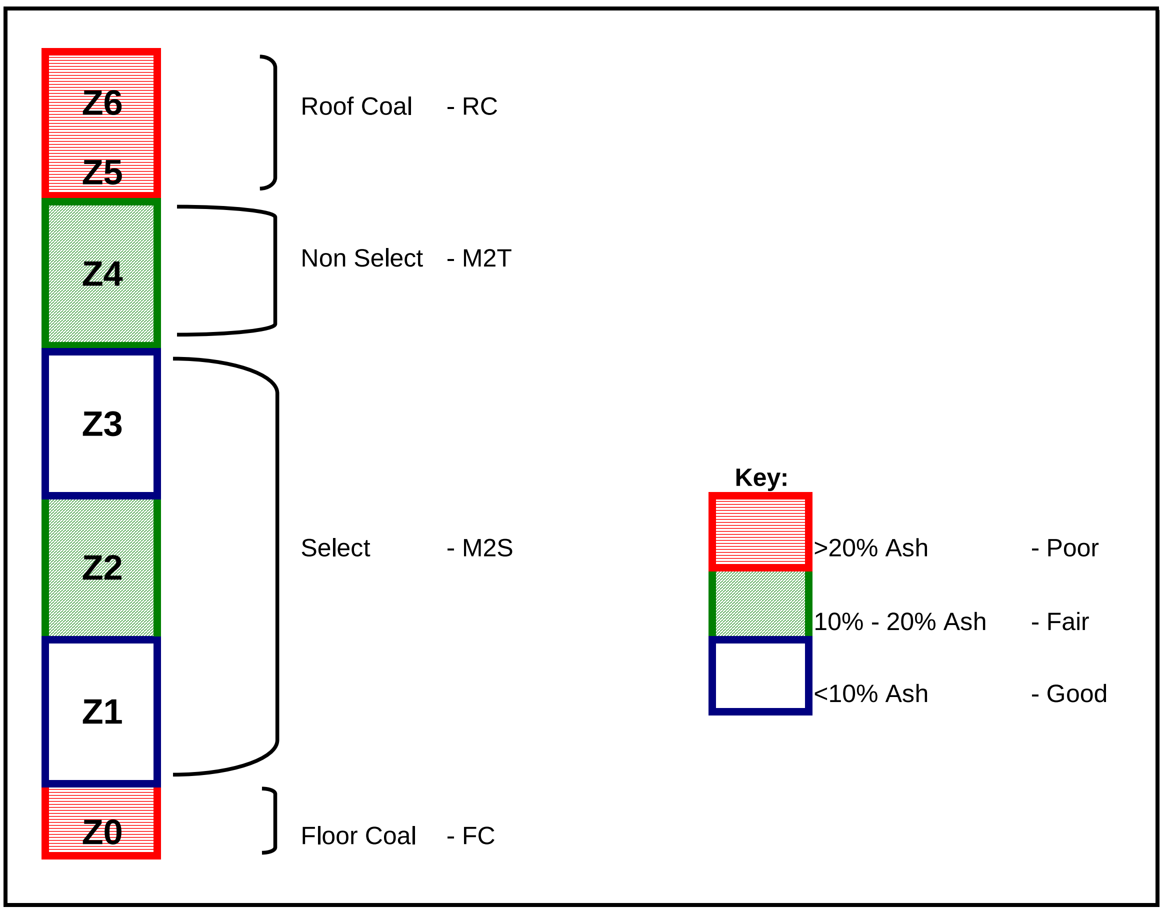


Figure 2.9 - Coal Quality Zones in the No.2 seam Kleinkopje Colliery

In some areas of the mine poor quality floor coal Zone 0 (Z0) and a similar poor quality roof coal, Zones 5 and 6 (Z5; Z6) are developed. To make this more user friendly the zones 1, 2 and 3 have been combined to form the No.2 Select (M2S) mining horizon with Zone 4 forming the No.2 Non Select (M2T) mining horizon. Both the floor coal (FC) and the roof coal (RC) gets removed and spoiled due to poor quality. These mining horizons can be mined separately and delivered to the tip as a M2S, select feed and a M2T, non select feed or combined as No.2 Full Seam (M2F). This depends on the product required and the quality of the different mining horizons. The different quality zones and mining horizons are illustrated in the Figure 2.10.

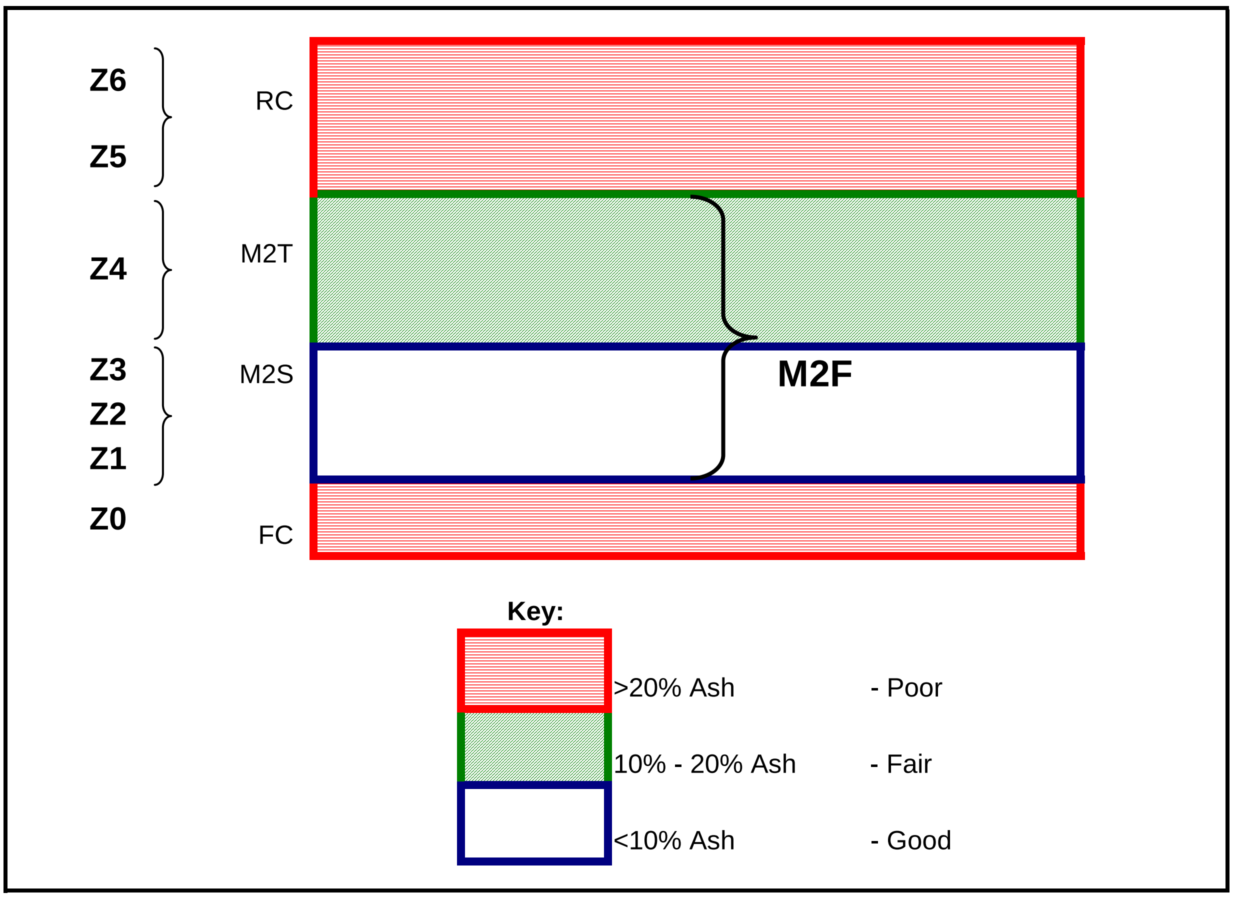


Figure 2.10 - No.2 Seam Zones

2.6.2. HISTORICAL UNDERGROUND GRADE CONTROL

The generally used term grade control refers to the procedure or method of selective control and monitoring of the mining horizon. The No.2 Seam, in-seam zoning forms the bases of this procedure, which was developed in the early to middle 70's to facilitate selective mining for the low ash, blend and coking coal export market to Japan. This was necessary to meet and maintain the stringent quality and quantity constraint written into the contract between the Transvaal Coal Owners Association (T.C.O.A.) and the Japanese (Barker *et al*, 1999).

This grade control system has since been significantly enhanced, but the same principles remain. It is based on the visual identification of the four main quality zones to guide and monitor the selective mining of the No.2 Seam. The higher quality coal (higher calorific value and lower ash content) is contained in the vitrinite or bright coal bands. Vitrinite occurs as very thin bands with a brilliant luster and are easily identifiable by sight (Ortlepp and Ackhurst, 1976). The concentrations of bright vitrinite rich bands increase in zones 1 and 3, while becoming sparsely distributed in zones 2 and 4. In the beginning this was used during the underground mining of the area and it is currently used during the secondary mining of the remnant coal in pillars, roof and floor. A benefit of this procedure is the geological prediction of the feed quality to the plant by combining the expected coal quality from the different mining faces. This allows the plant to set their densities to the correct level to optimize the product recovery and product quality control.

It is important to adapt the mining horizon and height, to optimize the coal quality and tonnages. The best quality coal is in zones 1 and 3, but this will have to be mined together with zone 2. The best mining height and horizon is from the base of zone 1 up to the top of zone 3, as shown in Figure 2.11.

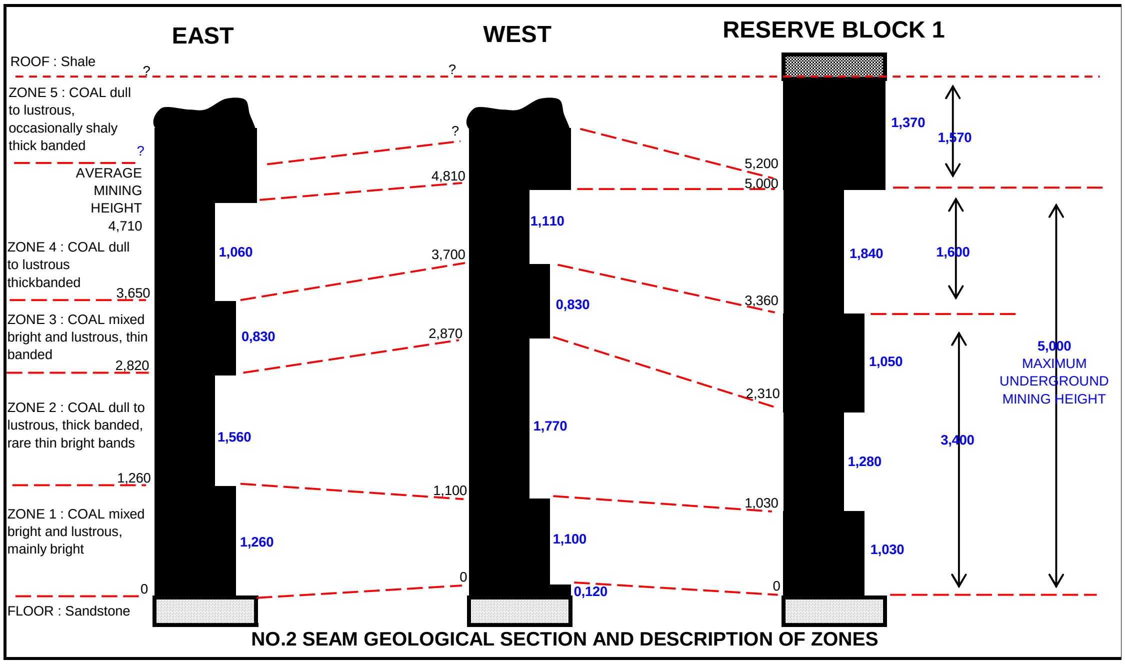


Figure 2.11 - No.2 Seam Geological Section and Description of Zones

(Adapted from Ortlepp and Ackhurst, 1976)

Historically, the mining height was from 3.45m to 3.75m which was about the limit of the mining equipment then available. Prior to the introduction of the routine grade control in October 1975, the best mining height was based on borehole data and a single target height was set for the whole mine. The borehole spacing was not as close as the current excepted standard. Figure 2.12 and Figure 2.13 illustrates the negative effect on the yield of not achieving the best mining height in two different mining areas (Examples a and b). In both cases the actual mining was less than the best mining height, resulting in less coal from the higher yielding Zone 3 and a greater proportion of the lower yielding Zone 2 coal in the feed to plant.

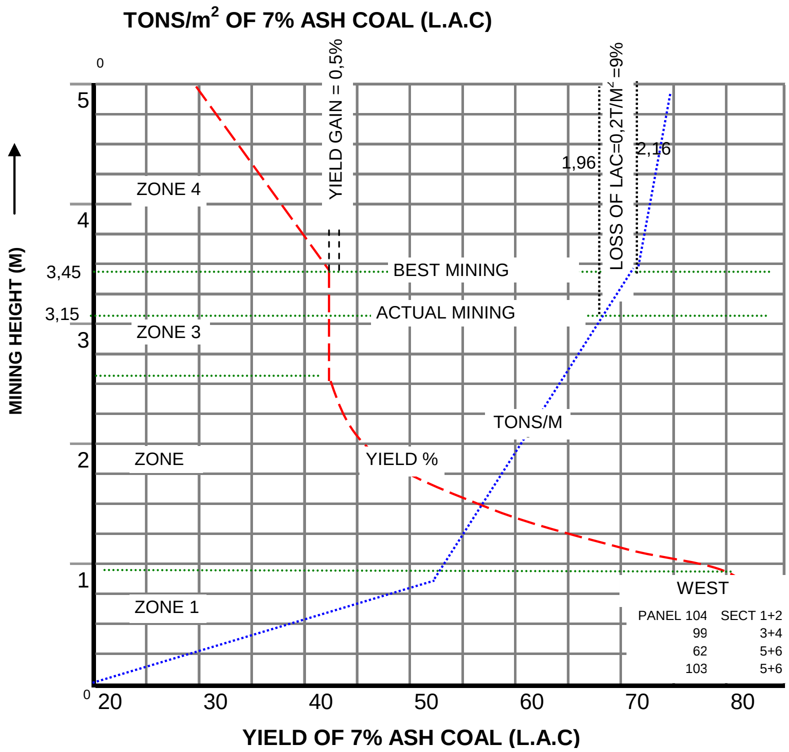


Figure 2.12 – Example a of No.2 Seam Mining Horizon

(Adapted from Ortlepp and Ackhurst. 1976)

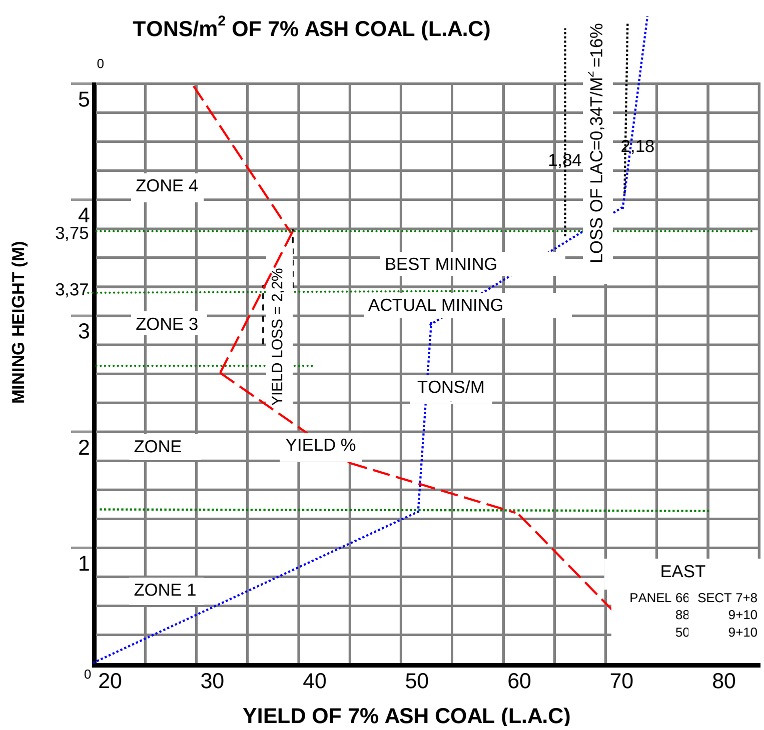


Figure 2.13 – Example b of No.2 Seam Mining Horizon

(Adapted from Ortlepp and Ackhurst. 1976)

The main reasons for not achieving the best height in the underground operations was due to the difficulties to mine at heights above 3m with the equipment available, and the roof conditions. It was equally important not to leave coal in the floor, as zone 1 contains the highest yielding coal. The floor has localized irregularities, undulations otherwise known as "stone rolls". This created difficulties in floor control and the extraction of zone 1.

Both the zone 1 coal left in the floor and the zone 3 coal remaining in the roof were closely monitored and measured by the grade controller so that corrective action could be taken to retrieve this coal. This was before computer modeling, such as Gradecon and Prodint which could combine zones and simulate the actual washing process, was available. Today we are mining the old workings by means of opencast methods and the grade control procedures have been adapted accordingly.

2.6.3. CURRENT OPENCAST GRADECONTROL

The principle reason for having a grade control system in place has not changed from the early TCOA and Japanese contract days. It has become even more onerous and crucial to the survival of the mine. Historically a previously unmined coal seam was mined using underground mining methods. Currently the same previously underground mined areas are mined using large scale opencast equipment.

The main difference is that the No.4, 2 and 1 seams are currently being exploited, producing a select blend product and a non-select steam coal product. Previously, the coal was only sourced from the No.2 seam, and was supplied from 7 sections, with a total of 84 headings. The current operation has 13 ramps, but only mine 2 to 3 faces at a time, which means that large volumes is extracted from a few areas. This results in less opportunity for mixing and blending of the different coal yields and qualities. Added to this is the fact that currently the previously mined No.2 seam pillars and roof coal with large amounts of dilution are included in the ROM due to the mining method.

The geological model still forms the basis of the mine planning and grade control systems. Both mine planning and geology use the same model information and the results are therefore comparable and reconcilable. The total reserve is subdivided in 50m x 50m mining blocks. The current modeling package used is the Mincom, Miner2 software, until the conversion to Stratmodel has been completed. These blocks are then processed in the geological model, and tonnage, quality and yield values attached to each. This is downloaded into a mine planning scheduling package, Xpac, from which a weekly, monthly, annual, and life of mine plan is constructed. This takes into account drill depths, soft and hard overburden, interburden volumes, coal tonnages, yields and ultimately saleable tons for the various products.

Geology use this model data to predict and reconcile the coal mined and beneficiated. The grade controller’s function is that of monitoring the coal exposed and available for loading, as in Figure 2.17, determining which coal faces to send to the tip as select, and which as non-select coal. In pit cut-offs and limits are indicated on the highwall to assist mining, shown in Figure 2.14.

Figure 2.14 – In Pit Highwall Markings

Daily pit visits are done by the grade controller to record the coal face position. This information is then related to the specific gradecon blocks being mined, to do checks as shown in the flow diagram, Figure 2.15.

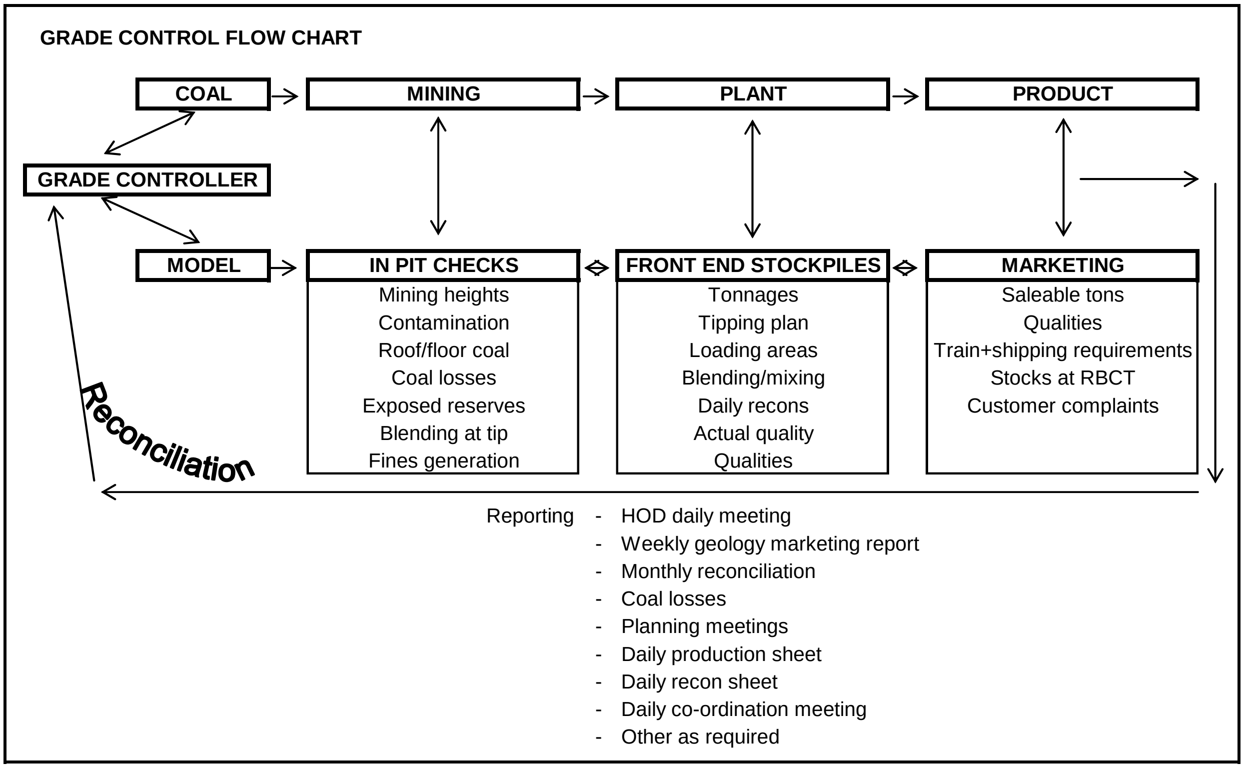


Figure 2.15 - Gradecontrol Flowchart

The next role player in the grade control process is the plant, for the ROM feed coal to be beneficiated to produce the required product tons at a pre-determined quality. Mining delivers the ROM coal in the designated tips according to the tipping plan which is drawn up daily by the grade controller after mutual consultation between mining, geology and the plant. From the three tips, the ROM feed is placed on either the A (non-select) or B (select) stockpile. Each stockpile is further subdivided into roughly four blocks to allow for different coals to be kept separate and independent stacking and reclaiming to take place.

The individual stockpiles are then visually sized and compared to the recorded tonnages from the tip control system and the plant weightometers. Using this information, each stockpile is then assigned a tonnage and quality by the grade controller. The plant is provided daily with the expected yield, calorific value, ash, and volatile percentage at a predicted cutpoint density of each individual stockpile as shown in Figure 2.18. This data is generated using an in-house developed program, Gradecon, which simulates the beneficiation of coal by using individual wash curves, based on analytical model data and applying plant discount factors. This then allows the plant to set their equipment at the correct densities to handle the feed and produce the required product qualities.

The hourly plant figures are recorded and made available to the grade controller daily. This actual plant data is used by the grade controller to simulate a wash and determine a theoretical yield, which can be reconciled to the actual yield produced, seen in Figure 2.19. The deviations or discrepancies are investigated and related back to the geological model, actual mining conditions, plant factors and efficiencies, so that continuous monitoring and corrective adjustment can take place. This continuous mining cycle with all the role players is illustrated in Figure 2.16.

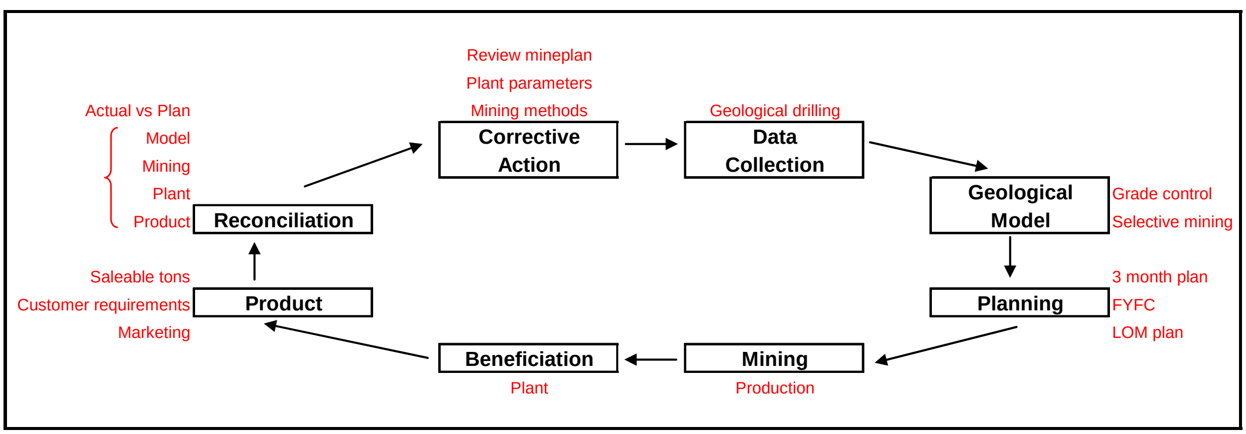


Figure 2.16 - Mining Cycle

In the bigger scheme, the daily, weekly and monthly production must tie in with the annual mine budget and saleable tons produced. This ensures financial viability and customer satisfaction. The effectiveness of daily reconciliation is demonstrated in Figure 2.20.

LOADING PLAN : ***22-Feb-05*** **STEAM COAL**

COMMENTS	TONS	RAMP	SEAM	BLOCK	YIELD	DENSITY	ASH	C.V.	VOL/ROGA	EXPOSED	PARTING
		2A-STH									
		10R	M1	3007A	69.9	1.64	16.2	27.45	25.3	15000	120000
		3A-NTH									
	4000	11R	M4	0534A	53.2	1.67	14.3	27.45	24.1	25000	
		5-WEST									
	9000	15R	M2	3148A	45.2	1.61	15.0	27.45	23.0	55000	
		16R	M1	3133A	50.1	1.59	16.3	27.45	24.4	-	5000
WATER		17L	M1	3828A	47.4	1.55	15.5	27.45	24.8	35000	15000
			ROMN	-	51.3	1.63	15.0	27.45	22.5	10000	
									TOTAL NONSELECT RESERVES:	140,000	140000
					BLEND						
		2A-STH									
WATER		07R	M2S	2934B	57.1	1.58	13.5	28.20	22.9	20000	
		07R	M1	2934B	74.7	1.80	13.8	28.46	25.4	5000	>1.75
SPONCOM	9000	09R	M2F	3018B	51.6	1.57	13.6	28.20	23.5	20000	1.70 - 1.75
											1.65 - 1.70
											1.60 - 1.65
		5-WEST									1.55 - 1.60
MOOLMAN		18R	M2	2610B	64.0	1.75	13.5	28.41	24.6	1000	<1.55
		2A-NTH									
CAPPING		21R	M4	2272b	59.7	1.60	13.8	28.20	23.8	25000	
									TOTAL BLEND RESERVES:	71,000	tons
									TOTAL RESERVES:	**211,000**	**tons**

Current mining faces

General mining comments

Exposed reserves

Figure 2.17 – Available In-pit Reserves

Figure 2.18 – Plant Front End Stockpiles

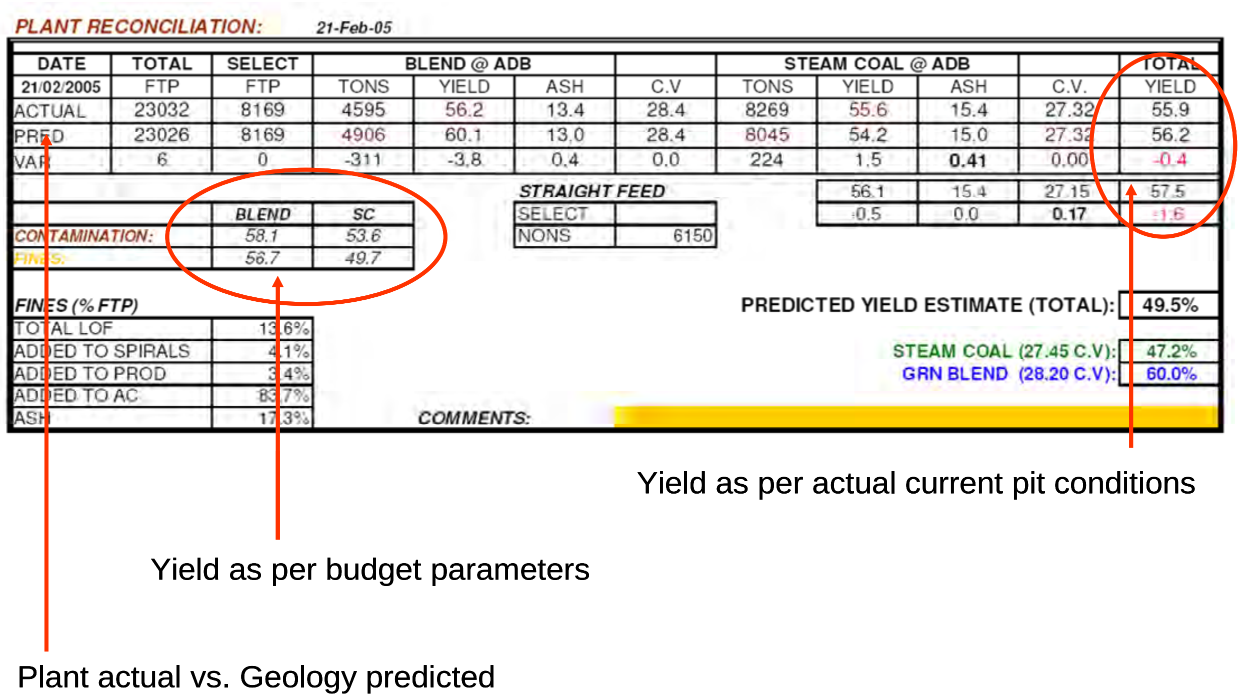


Figure 2.19 – Daily Yield Reconciliation

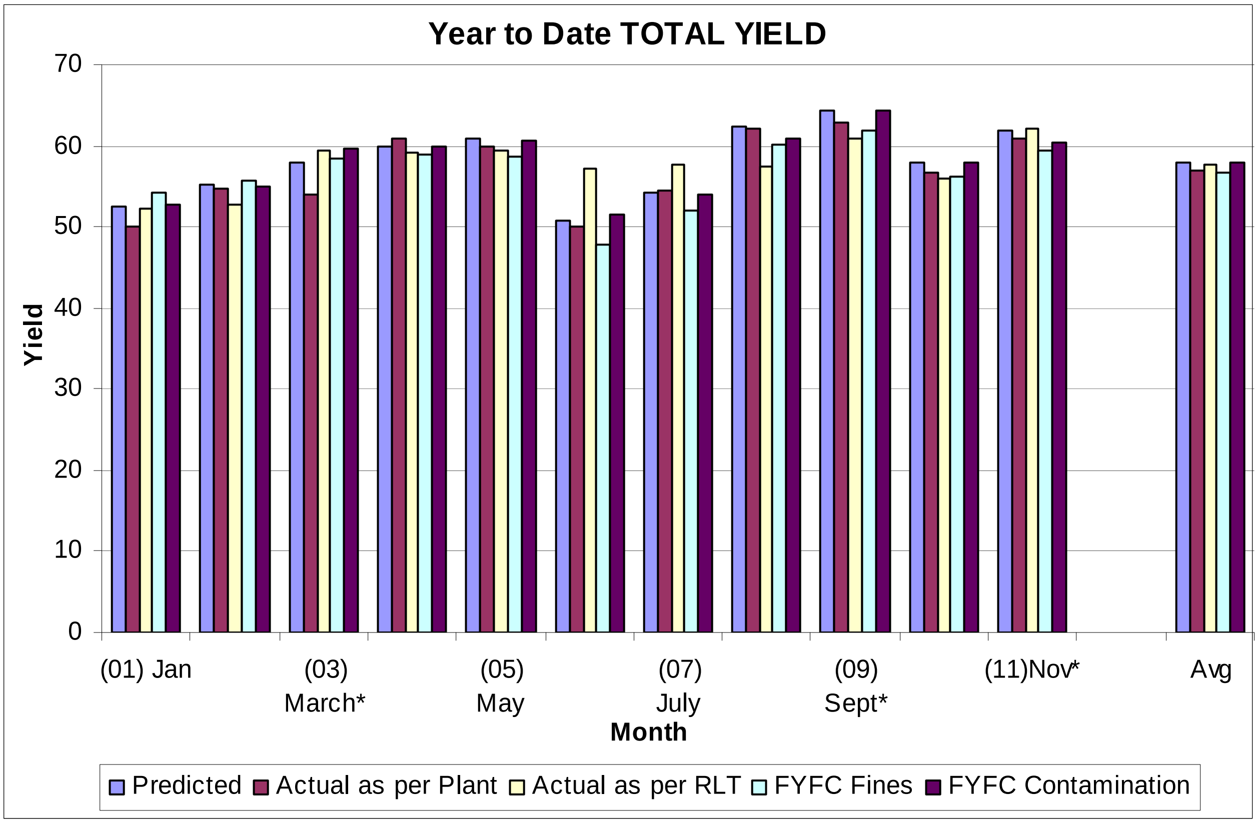


Figure 2.20 – Annual Yield Reconciliation

3. RESOURCE ESTIMATION AND UTILIZATION

3.1. INTRODUCTION

Resource or reserve estimation is done with a geological modeling software package known as Mincom's Miner 2. This uses the geological borehole information to estimate the coal seam area, thickness, and coal qualities within a delineated block. In this application, each borehole has an area of influence or polygon around it, the model uses this to extrapolate and to estimate overall seam area, thickness, and coal qualities within the delineated block.

The number of boreholes per block provides an indication of the level of confidence or accuracy of the reserve estimation for that block. This is expressed as the number of boreholes per 100 ha. The following definitions, based on the SAMREC code are used at Kleinkopje Colliery for resource estimation purposes.

a) Resources

Inferred Coal Resource

It is that part of a coal resource for which the tonnage and coal quality can only be estimated with a low level of confidence. A sampling density of less than four surface boreholes per 100 ha defines inferred resources.

Indicated Coal Resource

It is that part of the coal resource for which tonnage, densities, shape, physical characteristics and coal quality can be estimated with a moderate level of confidence. A sampling density of 4 to 8 surface boreholes per 100 ha defines an indicated resource.

Measured Coal Resource

It is the part of a coal resource for which tonnage, densities, shape, physical characteristics and coal quality can be estimated with a high level of confidence. A sampling density of greater than 8 surface boreholes per 100 ha defines a measured resource.

b) Reserves

Probable Coal Reserve

A probable coal reserve identifies the economically mineable coal derived from a measured coal resource and/or indicated coal resource. It is estimated with a lower level of confidence than that applying to a proved coal reserve. Access to the reserve still needs to be established.

Proved Coal Reserve

A proved coal reserve is the economically mineable coal derived from a measured coal resource. It is estimated with a high level of confidence and access to the reserve is established.

Figure 3.1 illustrates the change in resource and reserve category with an increase in geological information.

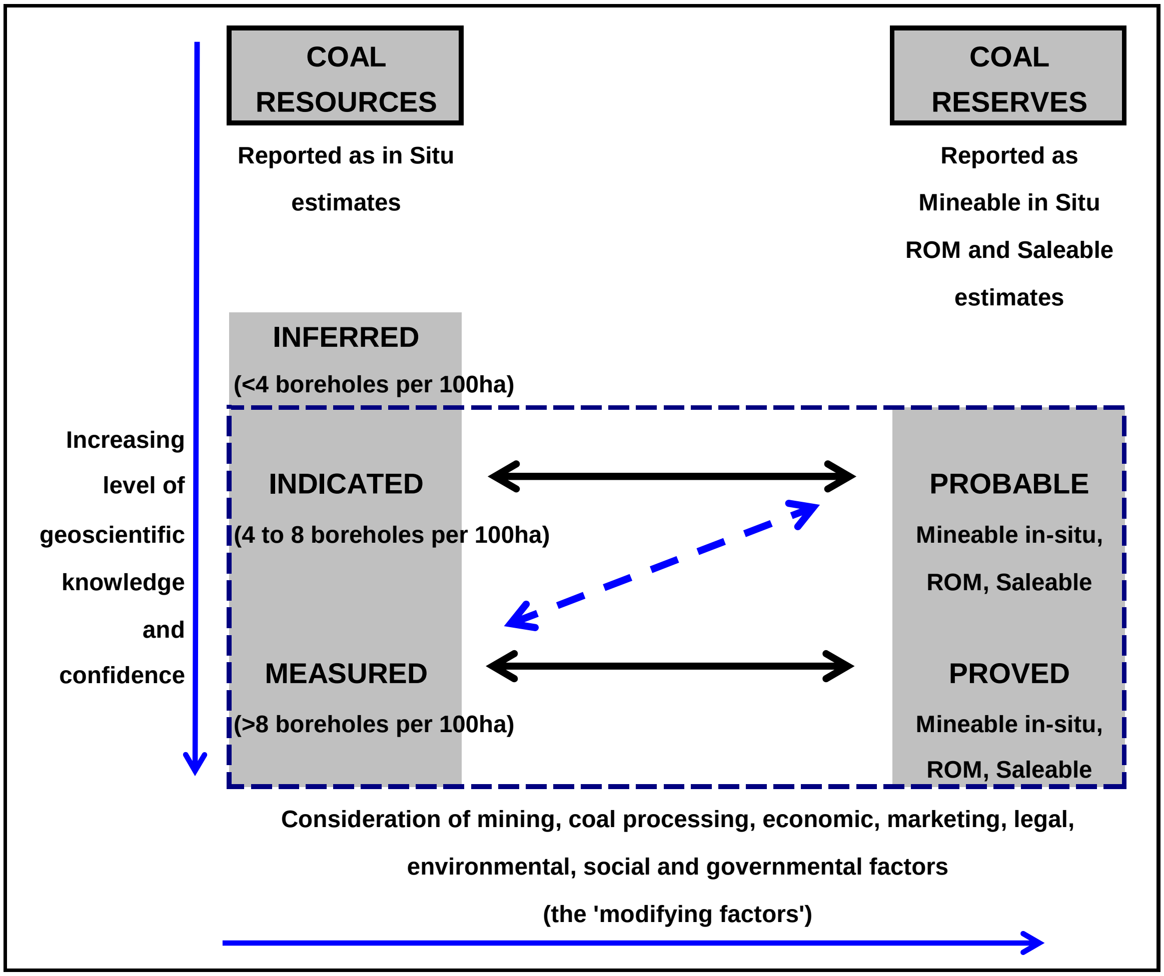


Figure 3.1 - Relationship between Coal Resources and Reserves

(Adapted from the South African code for reporting of mineral resources and mineral reserves, March 2000)

3.2. Resource Statement

The study area considered here has more than 8 boreholes per 100ha and could be classified as a measured resource and probable reserve.

The 2005 resource estimate for the “North West” block is provided in Table 3.1. and the reserve estimates are tabulated in Table 3.2.

Table 3.1 – Resource in Case Study Area

BLOCK	MINING HORIZON	AREA	SEAM THICK	R.D	GROSS TONS IN SITU	TONS PREV MINED	TOTAL TONS IN SITU	GEOL LOSS	MINEABLE TONS IN SITU	(MTIS/ GTIS)
		Ha	M		Mt	%	Mt	%	Mt	%
PROBABLE RESERVES	M5	41.6	1.30	1.47	0.778	75.0	0.195	7.0	0.181	23.2%
	M4	94.7	2.23	1.61	3.414	0.0	3.414	7.0	3.175	93.0%
	M2FS	482.7	4.22	1.53	45.582	32.0	30.864	9.0	28.086	61.6%
NORTHWEST	M1	109.0	1.52	1.57	2.598	0.0	2.598	7.0	2.416	93.0%
	TOTAL	728.0	3.39	1.54	52.371	27.0	37.070	8.7	33.858	64.6%
MEASURED RESOURCES	M5	86.7	1.30	1.45	1.638	75.0	0.409	7.0	0.381	23.3%
	M4	52.2	1.96	1.62	1.657	0.0	1.657	7.0	1.541	93.0%
NORTHWEST DEEP	M2FS	107.8	5.06	1.48	10.542	22.0	8.072	9.0	7.345	69.7%
(>70 M TOP S2)	M1	65.8	1.71	1.56	1.757	0.0	1.757	7.0	1.634	93.0%
	TOTAL	312.5	2.79	1.51	15.594	17.5	11.895	8.4	10.901	69.9%
TOTAL		1040.5	3.21	1.53	67.965	24.18	48.965	8.57	44.759	66.23%

(Adapted from Kleinkopje FYFC 2005)

3.3. Reserve Statement

Table 3.2 – Reserve Statement in Case Study Area

BLOCK	MINING HORIZON	MINING LOSS	MINING EXTRACT	ECOVER ROM ADB / MTIS	R.O.M TONS ADB	CONT.	R.O.M TONS ADC	INH H2O	TOTAL H2O	R.O.M TONS AS DEL	R.O.M. AS DEL/ MTIS
		%	%	%	Mt	%	Mt	%	%	Mt	%
PROBABLE RESERVES	M5	4.0	90.0	86.4	0.156	40.0	0.261	2.3	6.5	0.272	150.5
	M4	5.0	94.0	89.3	2.835	2.5	2.908	2.1	6.5	3.044	95.9
	M2FS	5.0	82.0	77.9	21.879	20.0	27.349	2.2	6.5	28.607	101.9
NORTHWEST	M1	5.0	76.0	72.2	1.744	3.5	1.807	1.8	6.5	1.898	78.6
	TOTAL	5.0	82.7	78.6	26.615	17.7	32.3	2.2	6.5	33.822	99.9
MEASURED RESOURCES	M5	4.0	90.0	86.4	0.329	40.0	0.548	2.3	6.5	0.573	150.4
	M4	4.0	93.0	89.3	1.375	2.5	1.411	2.1	6.5	1.477	95.9
NORTHWEST DEEP	M2FS	5.0	82.0	77.9	5.722	20.0	7.152	2.3	6.5	7.478	101.8
(>70 M TOP S2)	M1	5.0	76.0	72.2	1.180	3.5	1.223	1.9	6.5	1.283	78.5
	TOTAL	4.8	82.9	79.0	8.606	15.7	10.334	2.2	6.5	10.811	98.8
TOTAL		4.94	82.80	78.71	35.221	17.4	42.659	2.17	6.50	44.632	99.56

(Adapted from Kleinkopje FYFC 2005)

The method of estimation works as follows:

The geological modeling software generates total seam area per block in hectares, with a total seam thickness in meters, a raw relative density (rd), and a tonnage called gross tons in situ (GTIS).

The GTIS will contain all the coal within the delineated block. Therefore if any portion needs to be excluded due to thickness or quality cut-offs, this should have been done when the block outline was determined. It is therefore assumed that everything within this block complies to the set constraints.

In the case of previously mined out areas a derating factor has to be applied to account for the mined out tonnage.

3.4. UNMINED AREAS

By using the SAMREC code and the SANS guideline it is possible to do proper resource estimation.

a) Resource utilization is directly related to the mining method i.e. opencast versus underground, and the total extraction versus selective mining
b) This will be determined by the depth, thickness, quality and locality of the resource
c) The main driver is the market or offset area for the product, which will determine whether it's a raw or washed product that is required

3.5. PREVIOUSLY MINED AREAS

The SAMREC, 2000 (see clause 6.4 and sub clause 6.4.1) code and SANS, 2003 (see code 6.5.2.8 Previously mined areas) guidelines are used to ensure that the resource and reserve evaluation is internationally acceptable.

Previously mined areas are slightly more complex as far as estimation of resources is concerned. It is necessary to bring into account the previously extracted or mined out areas and derate the resource for that.

Also very important is the expected or predicted losses applied, relating to secondary or tertiary mining operations.

In order to accurately reconcile between planned tonnages to be mined and actual tonnages extracted a proper survey needs to be done. However due to the safety risk associated with working on the blasted and collapsed old workings it is not possible to physically survey the top of coal. Most of these areas have some degree of heat buildup due to spontaneous combustion as well as the presence of uncollapsed voids not immediately evident.

It is only possible to accurately measure the final floor of coal and cut width. The top of coal exposed by the dragline is estimated as well as the highwall and lowall edges of coal, which results in errors in estimation. Losses due to scalping by the dragline and minor changes in seam thickness can not be measured accurately.

In order to understand all the factors associated with previously mined areas we need to consider the following:

a) Mining design dimensions
b) Percentage mining extraction
c) ROM quality and product requirements

3.6. PROBLEMS ASSOCIATED WITH OLD WORKINGS

3.6.1. DERATING

This is one of the biggest uncertainties within the current reserve and resource estimation process. It is mainly due to sparse information and the unavailability of access to old workings. In order to determine the derating percentage, two components are required, the arial extraction and the mining height which is then used to calculate the volumetric extraction.

Arial extraction, shown in Figure 3.2, is reasonably easy to determine from the old survey plans giving the mapped pillar positions and shapes. However, very old areas mined in the pre-Salamon (1967) era were not off-set which increases the uncertainty and decreases the level of accuracy. Off-setting is the actual surveying of the pillar dimensions or shape, which was only started after 1967.

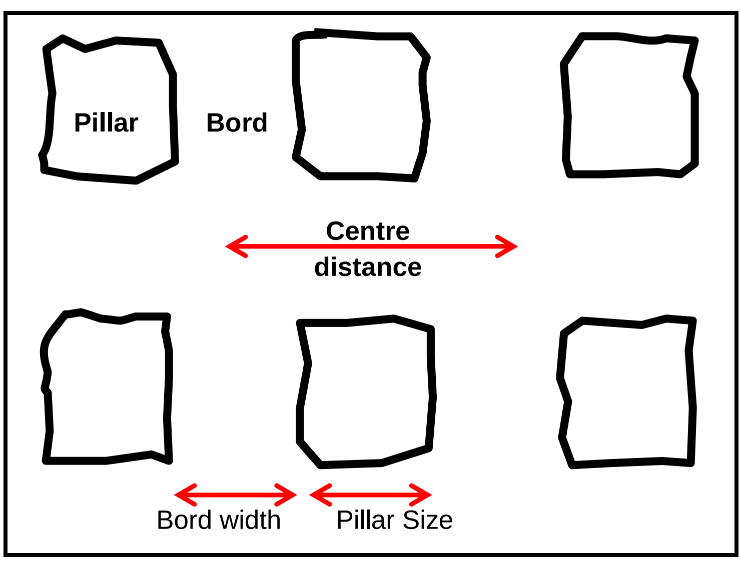


Figure 3.2 - Arial Extraction Percentage

The calculation or estimation of the volumetric extraction is more problematic. It is based on the area and the old mining heights (see Figure 3.3 below). The frequency of mining height measurements is very low and it is required to estimate these heights.

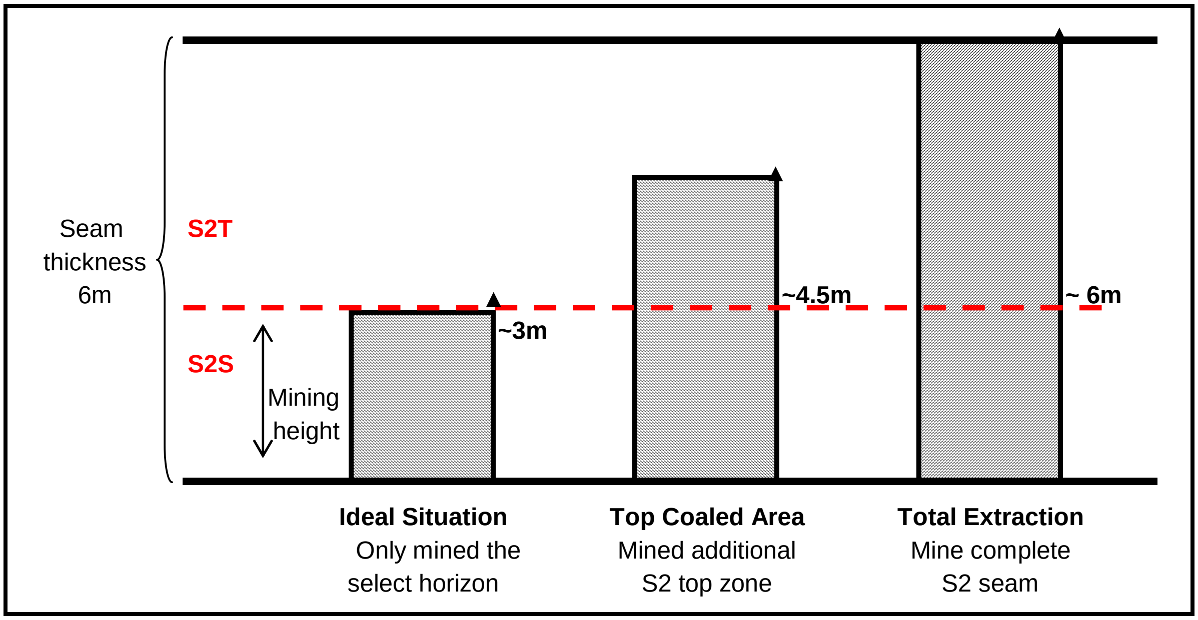
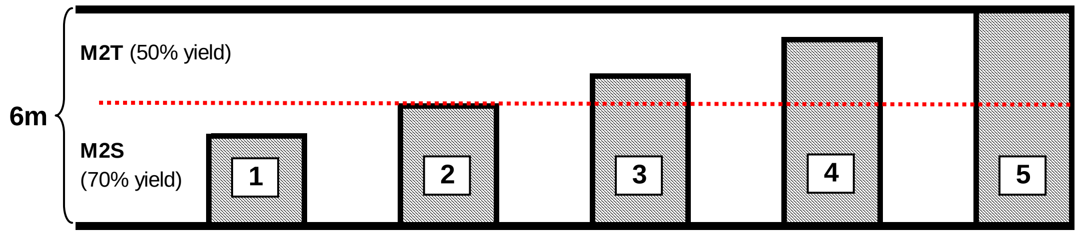


Figure 3.3 - Volumetric Extraction

It is assumed that the primary mining was conducted in the M2S (M2 select zone) portion of the seam. Additional top coaling included mining parts of the M2T (M2 top zone). Even where the top coaled areas are indicated on plan, very few mining height measurements are shown.

The estimation process of tonnage, yield, and coal qualities are more complicated in derated areas. The select (M2S) and top (M2T) portions of the seam have different relative densities, yields and qualities. Resource estimation of derated areas depends on where the primary mining horizon was and this determines the remaining reserve. This could vary significantly between actual and estimated, as may be seen in Figure 3.4.



	Weighting of the % yield will give the following results
1. Left select coal in roof	1. 70%
2. Ideal case, only mined the select horizon	2. 70%
3. Increased mining height including top zone	3. 65%
4. Increased height further by doing top coaling	4. 62%
5. Total seam height extracted	5. 60%

Figure 3.4 - Determination of Remaining Reserve

Although the yield is reduced by including the top zone, it should be remembered that the overall saleable tonnage increases which improves the saleable strip ratio. It is therefore a trade-off between saleable tonnage and yield. Also important is the change in mining dimensions, for example, as the mining height increases larger pillars need to be left to maintain the overall safety factor. However this was not the norm before the use of Salamon's equation and areas mined before 1967 had smaller pillars. A larger derating factor needs to be applied to these areas.

It is evident from this that derating plays a crucial role in the financial viability of the reserve. The case study area in particular is a very good example of relatively small pillars and extensive top coaling. This will again have an effect on the amount of contamination, percentage extraction, spontaneous combustion, percentage fines generation and the overall ground stability.

3.6.2. MINING EXTRACTION

3.6.2.1. GROUND STABILITY

Rock mechanics in opencast mines is mostly related to slope stability of the highwall and low wall. Additional complications arise when previously mined bord and pillar areas are considered for secondary extraction by opencast methods. In this case the stability of the previously mined areas is of great importance. Because this treatise deals with opencast mining methods applied to previously underground mined areas both sets of conditions are considered.

Typical ground stability issues on an opencast operation relate to the different types or mechanisms of slope failure, namely:

a) Plane failure – single joint or plane
b) Wedge failure – multiple or intersecting joints and planes
c) Toppling failure – steep joint or failure along pre-split in Figure 3.5
d) Circular failure – sloughing in soft and unconsolidated material in Figure 3.6

Figure 3.5 - Photo of Toppling Failure

Figure 3.6 - Photo of Circular Failure

The actual joint or plane surface condition plays a significant role in the probability of failure and movement occurring. If the surface is wet or very smooth the likelihood of failure increases. Slope angle is also important for overall stability, especially in soft or unconsolidated material.

The natural angle of repose of the soft and unconsolidated material needs to be determined. This material must then be battered back or sloped to an angle or gradient less than the angle of the repose. Figure 3.7 illustrates the highwall and low wall angles.

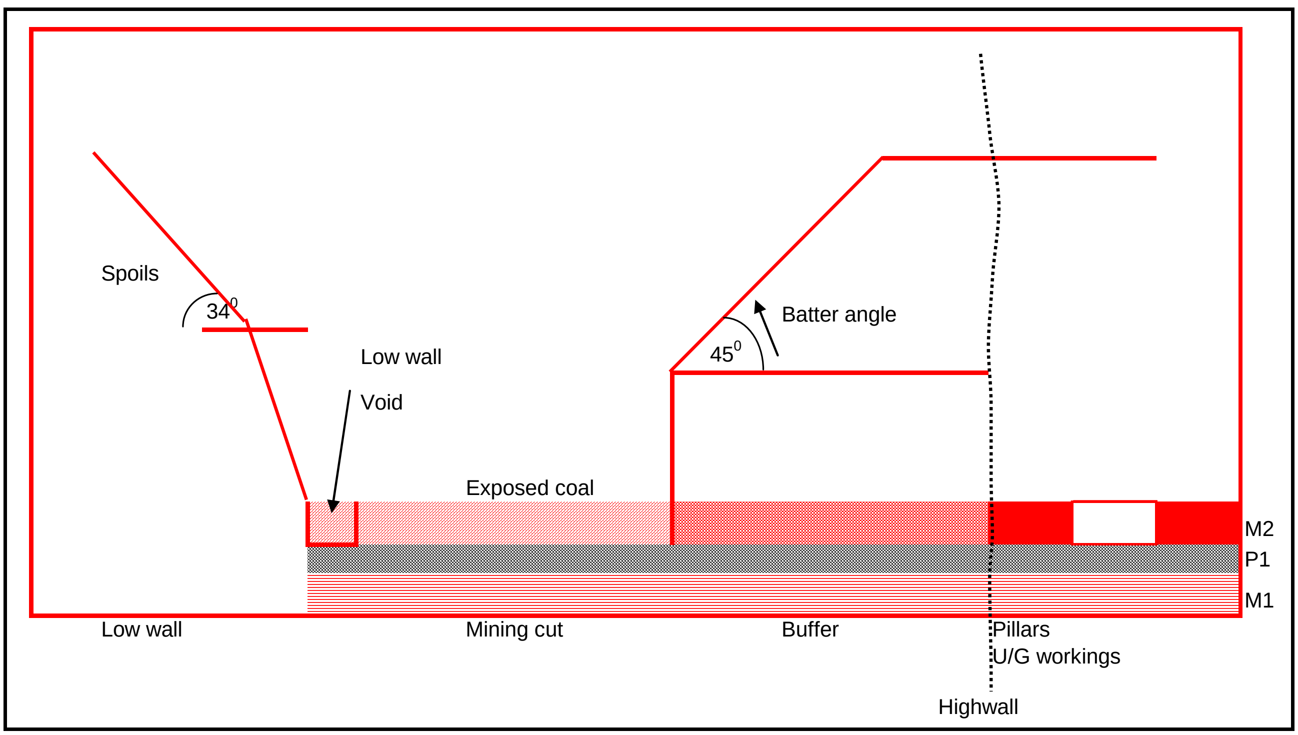


Figure 3.7 - Diagram of Highwall and Low Wall angles

The main risk of unstable ground and slope failure is to people and equipment, and then also to coal reserves and productivity.

The common hazards are as follows:

a) Weathered overburden

b) Water

c) Coal seam and strata undulations

d) Joints

e) Faults

f) Dolerite dykes

g) Burning and spontaneous combustion

h) Underground workings

i) Top coaled areas

j) Deviations from mining standards (placing softs at base of spoils, undercutting of spoils, spoiling on water, lack of voids, not battering back, not spoiling far enough, highwall loading, water accumulations and poor drainage, lack of slot dozing)

All the above can lead to slope failure which will have a negative impact on the coal extraction and eventually culminate in a financial loss. The ideal situation would be to have the correct strata control system in place to handle the mine or pit specific conditions. This will reduce the risk to people, equipment, coal reserve and financial viability.

Ground stability in an underground environment is affected by other factors. The starting point is the initial pillar design for a bord and pillar mine layout. This is currently done by using a formula which is based on the coal pillar safety factor concept. Before 1960, there was no method of calculating the coal pillar strength to assist with mine design (Van der Merwe, 1995). This lead to pillars being too small to support the overlying strata or conversely to pillars being too large, resulting in coal losses. In 1960, the Coalbrook Colliery disaster resulted in a massive loss of 437 underground workers lives due to the failure of an estimated 7 500 pillars (Madden, 2000). This triggered intensive research to prevent any similar occurrences from happening. In 1967, professor M.D.G Salamon and A.H Munro published a paper in which they proposed a formula to predict the strength of coal pillars which is discussed below.

Coal pillar strength:

The crux of this formula states that the strength of a coal pillar is dependant on three parameters namely,

a) The inherent strength of the coal material

b) The width of the pillar and

c) The height of the pillar.

Ơ' = pillar strenght (MPa)
w = pillar width (m)
h = pillar height (m)

$$\sigma' = 7.2 \frac{w^{0.46}}{h^{0.66}} \text{ MPa}$$

A constant inherent strength of the coal material of 7.2 MPa was derived statistically from analysis of underground pillars (Van der Merwe, 1995).

Coal load:

This needs to be calculated to determine how much load a coal pillar is subjected to.

L = pillar stress (MPa)
H = mining depth (m)
W = pillar width (m)
B = bord width (m)
C = pillar centre (m)

$$\mathbf{L} = 0.025\ H\ \frac{c^2}{w^2}\ \text{MPa}$$

Safety factor:

The combination then of the coal pillar strength and coal pillar load will give an indication of the overall ground stability of a specific area. This is indicated by what is called the safety factor (SF).

$$\textbf{Safety Factor} = \frac{\text{Coal Pillar Strength}}{\text{Coal Pillar Load}}$$

$$\mathbf{SF} = \frac{\mathbf{O'}}{\mathbf{L}}$$

The safety factor can also be calculated by using a combination of the above formulas:

$$\mathbf{SF} = \frac{288\ w^{2.46}}{h^{0.66}\ H\ (w + B)^2}$$

Salomon's work as based on empirical data of 27 collapsed and 98 intact pillars. The back analysis of this data showed a probability of stable geometry where at a safety factor of 1.0 the probability of a stable state may be expressed as 0.9947 or 5 300 in 1 million, or 0.5% probability of pillar failure (Madden, 2000). This can be seen in Table 3.3.

Table 3.3 – Probability of failure for a given safety factor

Safety Factor	Probability of pillar failure (%)
2.0	0.0006
1.9	0.003
1.8	0.01
1.7	0.04
1.6	0.15
1.5	0.53
1.4	1.70
1.3	4.92
1.2	12.52
1.1	27.41
1.0	50.00
0.9	74.66
0.8	92.01
0.7	99.34
0.6	99.40

(Source: Van der Merwe, 1995)

This showed clearly that as the safety factor increased the probability of failure decreased. The probability of pillar failure is drastically reduced at safety factors above 1.6.

Generally there are four recommendations to adhere to in order to maintain acceptable geotechnical ground stability in an underground bord and pillar mine:

a) Maintain a pillar width to height ration of >2 for underground bord and pillar design.

b) Ensure that the percentage arial extraction is <75%

c) Maintain a minimum pillar width of more than 5,0 metres.

d) Maintain a factor of safety higher than 1.6.

An application of the concept is demonstrated in Figure 3.8 and the paragraph below.

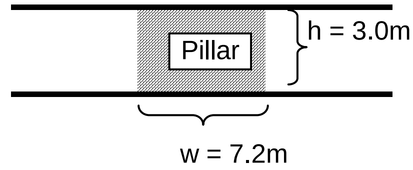
Pillar Strength (σ') :

$$\sigma' = 7.2 \, \frac{w^{0.46}}{h^{0.66}}$$

$$= 7.2 \, \frac{7.2^{0.46}}{3.0^{0.66}}$$

$$= 7.2 \, (1.2008)$$

$$= 8.65 \text{ MPa} \rightarrow$$

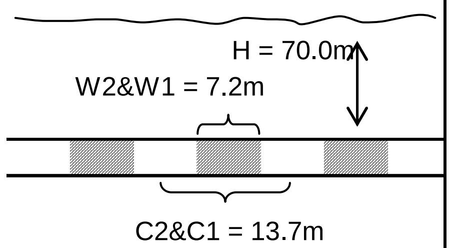


Pillar Load (L) :

$$L = \frac{0.025 \, H \, (C1 \; C2)}{(W1 \; W2)}$$

$$= \frac{0.025 \, (70.0)(13.7 \times 13.7)}{(7.2 \times 7.2)}$$

$$= 6.34 \text{ MPa} \rightarrow$$



Safety Factor: $= \dfrac{\text{Strength}}{\text{Load}}$

$$= \frac{8.65 \text{ MPa}}{6.34 \text{ MPa}}$$

$$= 1.36 \rightarrow$$

Figure 3.8 – An example of Safety Factor Calculations

This is less than the recommended 1.6 factor of safety and will therefore have a higher probability of failure.

Risk = Probability of failure x Consequence of failure

Assuming this is a working environment, then the consequence of failure can be loss of life, which is unacceptable.

3.6.2.2. WATER

This is a very extensive subject and relates to the degradation of water qualities within the old workings. Most of the old areas have a certain amount of recharge from the groundwater through boreholes, faults, fissures, dykes and other forms of natural or man made conduits. The water will then gradually deteriorate due to the mineral matter (predominantly pyrite and marcasite) inherent to the coal seam, which leads to the formation of sulphuric acid in oxidizing environments. Shallow, mined out areas have a higher risk of decanting acid mine drainage along the coal sub outcrop and outcrops. This acid water will then enter the surface water drainage systems and pollute them.

When a previously mined area is considered for secondary mining, water and the quality of the water needs to be included as one of the key factors influencing the viability of the project. Water accumulation in old workings is common and this will have several impacts on the future utilization of that remaining resource. A few examples of the possible negative impacts of water are listed below:

a) The deterioration in water quality over time prevents it from being easily returned to the natural surface water system.

b) High cost of treatment of poor quality water to acceptable quality.

c) The complex and onerous legislative process associated with water transfer, storage and usage.

d) The cost of pumping and piping of large volumes of water.

e) The risk of flooding and seepage from large underground and surface water bodies.

f) The time required to dewater areas.

g) The negative impact of water accumulation on mining operations.

h) Continuous recharge to existing water bodies.

i) Closure cost of mines.

These are only the main aspects to be considered when embarking on an evaluation exercise of previously mined areas. In each of the above instances there will be an associated cost implication. The cost can be a once off initial expense, or continuous throughout the mining period or even at closure or all of the above.

3.6.3. DILUTION AND CONTAMINATION

3.6.3.1. CONTAMINATION

The definition of contamination according to the South African National Standard (2003) is the following:

> Contamination
>
> It is extraneous coal and non-coal material <u>unintentionally added</u> to the practical mining horizon as a result of mining operations.

The other very important term is dilution which goes hand in hand with contamination and should also be clearly understood.

> Dilution
>
> It is non-coal or coal outside the theoretical mining height that is <u>intentionally added</u> in as part of the planned mining section to obtain a practical mining horizon.

Figure 3.9 shows an example of contamination and dilution in coal face.. This is a classic example of a previously mined area. The dilution material in the old mining board and the coal pillar is visible.

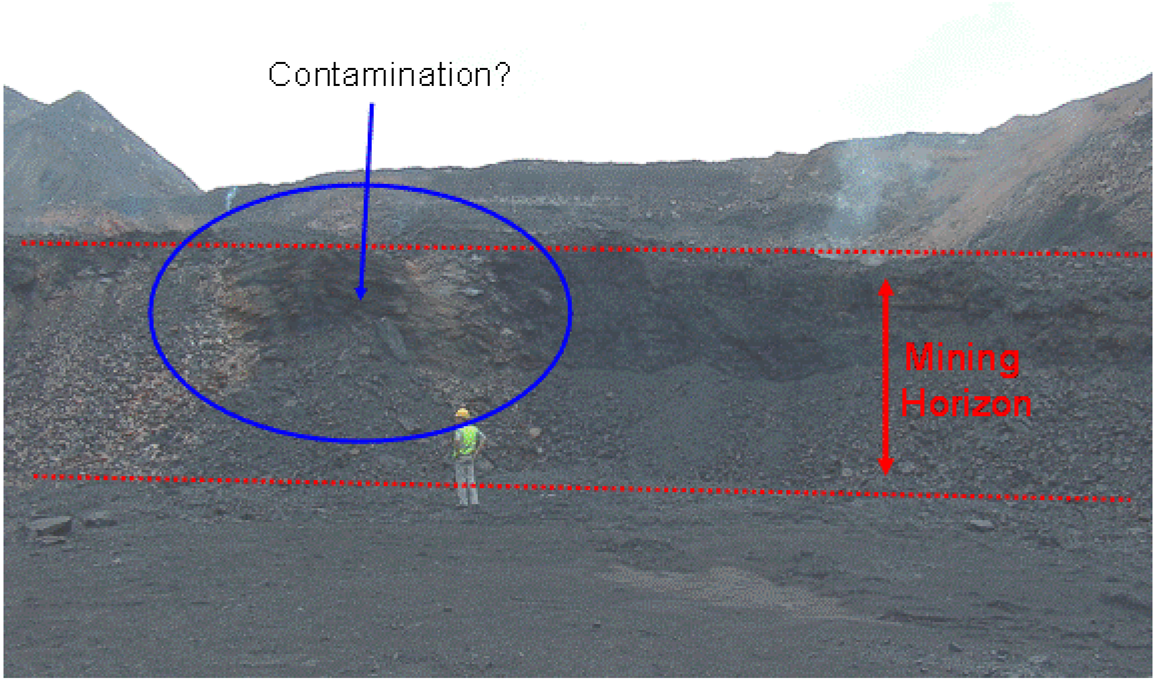


Figure 3.9 - Example of Contamination and Dilution in Coal Face

Figure 3.10 illustrates contamination on top of the coal seam in a previously unmined area. This contamination can be removed relatively easily.

Figure 3.10 – Example of Contamination and Dilution on Top of Coal Seam

The key phrases from the above are contamination – unintentionally added and dilution – intentionally added. Due to the size of the equipment used in opencast mining operations, it is inevitable that a certain amount of contaminant will be mined together with the coal seam. This can originate from the roof, floor and highwall or low wall edges. In order to apply realistic factors to the geological predictions and reconciliations this unavoidable percentage of contaminant is provided for in the estimates, and is then called dilution. Anything in excess of the planned dilution percentage that gets added during the mining process is called contamination because that was unintentionally added.

The degree of contamination at Kleinkopje Colliery is increasing, due to the increasing mining depths, the higher proportion of production sourced from previously mined areas and the higher amount of contamination in areas where previous top coaling took place. This is more prevalent in the NW block where up to 65-70% of the area falls within this category. To ensure future viability new methods to reduce the degree of contamination need to be devised and implemented.

As contamination increases as a percentage of the total ROM feed to the plant it reduces the overall yield.

Total ROM feed = Contaminant + ROM coal

There are various sources of contamination and they are related to the different mining activities. The overall mining process follows a sequence of activities listed in Table 3.4.

Table 3.4 – Sequential activities of the mining process

Ore body	Step	Activity
	1.	Top soil removal
No.4 Seam	2.	Pre-strip removal
No.4 Seam	3.	Overburden drilling (pre-split and grid separately)
No.4 Seam	4.	Overburden blasting (pre-split and grid separately)
No.4 Seam	5.	Dragline removing overburden
No.4 Seam	6.	Drilling of No.4 Seam coal
No.4 Seam	7.	Blasting of No.4 Seam coal
No.4 Seam	8.	Loading and hauling of No.4 Seam coal to tip/plant
No.2 Seam	9.	Drilling of interburden
No.2 Seam	10.	Blasting of interburden
No.2 Seam	11.	Dragline removing interburden
No.2 Seam	12.	Drilling of No.2 Seam coal
No.2 Seam	13.	Blasting of No.2 Seam coal
No.2 Seam	14.	Loading and hauling of No.2 Seam coal to tip/plant
No.1 Seam	15.	Drilling of P1 parting
No.1 Seam	16.	Blasting of P1 parting
No.1 Seam	17.	Removal/spoiling of P1 parting by truck and PC/shovel
No.1 Seam	18.	Drilling of No.1 Seam coal
No.1 Seam	19.	Blasting of No.1 Seam coal
No.1 Seam	20.	Loading and hauling of No.1 Seam coal to tip/plant
	21.	Levelling of spoils
	22.	Rehabilitate spoils

Measures to deduce the impact of contamination related to specific activities are listed in Table 3.5.

Table 3.5 –Measures to reduce the Impact of Mining Activities on Contamination

Activity succession →			
Drilling	**Blasting**	**Dragline**	**Coaling**
Accurate depth control: - Pre-split drilling - Overburden drilling - Interburden drilling - Parting drilling - Coal drilling Affective hothole drilling Feedback on actual vs planned drill depths Missed holes to be reported and reconciled Condition specific planning of drilling Poor drilling result in coal losses	Effective and accurate blasting Adhere to correct mining horizon depths Blast overburden and coal separately Create optimal digging surface for draglines Over-blasting and under-blasting are detrimental to contamination Typical problems: - capping - hard drilling conditions - poor highwall conditions - mixing of coal and rock - hard pillars Limited spontaneous combustion control due to poor fragmentation, hard pillars and banks Coal losses	Dig on line and to correct surface Dig low-wall void to: - prevent berm losses - prevent contamination -improve water pumping Prevent scalping Clean coal surfaces reducing contamination Clean highwall properly Remove roof coal where possible No undercutting of spoils Do not put coal on highwall	Clean coal surfaces, reducing contamination Mine according to planned mining horizons Selective mining Reduce mixing Water management Reduce fines generation Avoid intermediate stockpiles Manage stockpiles effectively Emphasise accurate parting drilling and blasting Loading in water not good practice: - poor visibility - addition of sand and silt - building of temporary access roads and dams increase contamination and result in coal losses - wet coal needs to be stacked out to dry - rehandling, fines generation, contamination

Each of these activities influences the next, and if a problem or deviation occurs during the activity, this influence is negative. This results in loss of production time, additional cost of production, and eventually leads to a loss of coal or increases contamination which results in loss of saleable yield. Overall contamination may therefore be seen as the sum total of contamination related to each individual activity.

Overall contamination may be sub-divided into a fixed portion related to the geological conditions and the mining method, and a variable portion, which depends on the actual implementation of mining standards and efficiency measures.

The challenge is to minimize the variable portion and reduce the fixed portion by continual improvement of mining method design through a better understanding of the geological model.

3.6.3.2. SOURCES OF CONTAMINATION

This can be explained by looking at each activity in detail and identifying the possible generation of contamination.

<u>Overburden and interburden drilling:</u>

Drilling the blast holes too shallow will result in leaving a hard capping on top of the coal seam, which the dragline will not be able to remove. This will necessitate secondary drilling and blasting of the capping and removal by truck and shovel. The capping can also be drilled and blasted with the coal seam, which will increase contamination.

Drilling the holes too deep will result in blasting the top of coal with the overburden. This creates a mixing of roof contaminant with the coal seam. The dragline can also scalp this contaminated coal off with the overburden removal, resulting in coal losses. Drill spacing is also important because it will impact on the blasting with regard to the amount of charging required to break the rock sufficiently for the dragline to remove it.

In order to minimize the risk of contamination due to overburden and interburden drilling it is crucial to maintain accurate drilling depth control. The fixed portion will be determined by the accuracy and reliability of the current depth measuring equipment being used on the drills, which is a mechanical system rather than the latest available electronic systems. The variable portion will be determined by the operators adherence to the depths on the drilling plan and the maintenance of the depth measuring device. An online, real time, drilling monitoring system such as the Aquila system, is able to utilize the geological model information for drilling planning and will show and measure the compliance at all times. The best available system for the current operation needs to be identified and implemented to ameliorate both the fixed and variable portions of the contamination risk.

Parting drilling:

Drilling the holes too short will again result in a hard capping due to ineffective blasting resulting from it. The additional problem is the type of material that the partings normally consist of, namely a medium to coarse grained sandstone. The uniaxial compressive strength of these sandstones are in the range of ~100 MPa while coal is ~30 MPa. This hard material requires additional charge or closer spaced holes and this higher energy results in the creation of more coal fines from the seam below.

Drilling the holes too deep will result in mixing of parting and coal increasing contamination, or coal losses depending on the loading. If only the top of the seam was blasted with the parting then secondary drilling and blasting of seam will be required. It is not possible to drill on an uneven surface therefore the drilling surface, whether parting (hard capping) or coal (remainder of seam) first needs to be cleared and prepared.

The fixed portion of contamination risk can be reduced by optimal planning of the drilling spacing, pattern, and depths based on the available geological model. The variable portion depends on the strict adherence to the plan and drilling standards.

Hot hole drilling:

Due to spontaneous combustion of the coal and shales in the old workings, drilling can be very difficult in these areas. If the holes are too deep or misplaced they go into the hot old workings, causing them to be abandoned and requiring a re-drill to replace the faulty hole. Hot holes also cause problems for charging and blasting, sometimes resulting in ineffective blasts leaving hard banks which the dragline cannot remove. Figure 3.11 illustrates the problems associated with hot areas. The difference here is that no secondary drilling and blasting can be done on hot "sponcom" areas.

Figure 3.11 - Effect of Hot Drilling Areas

The amelioration of the risk at this stage is not possible. The solution to this problem lies in the root cause, which is the prevention of the heating in the first place through timeous and effective cladding.

Overburden and interburden blasting:

This activity is intended to break the overburden material in such a way that the dragline, which is the next activity, will be able to dig and remove it. The effectiveness or success of the blasting plays a major role in the productivity of the dragline operation. If the material is not broken sufficiently by the blast, it leads to hard banks, large size material and odd shapes which the dragline cannot effectively move. An example of an un-blasted sandstone capping is shown in Figure 3.12.

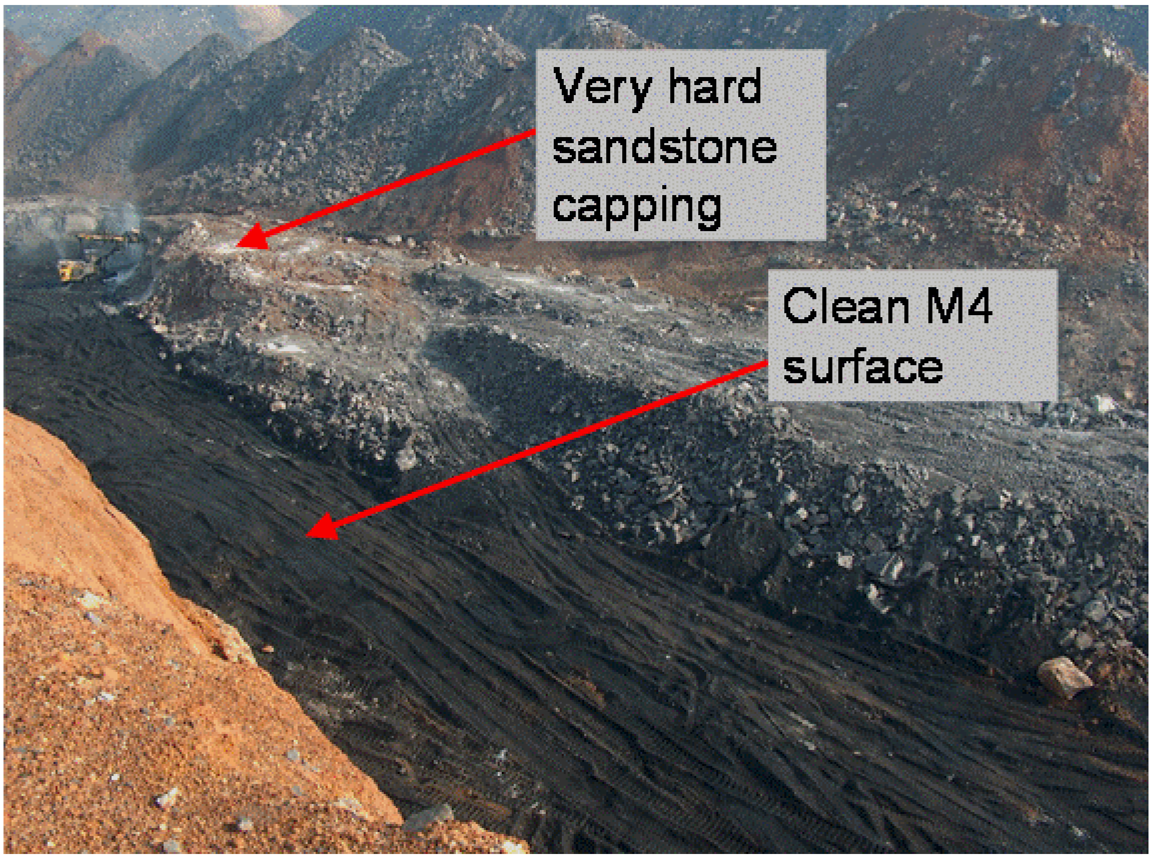


Figure 3.12 - Sandstone Capping on Coal Seam

The resulting hard digging, hard banks, hard capping and slow dragline digging will increase the risk of damaging the dragline bucket, which is very costly. In the event that the dragline cannot dig properly due to large boulders not blasted, it results in this material being left behind which results in coal losses. Secondary drilling and blasting of the boulders after the dragline has moved, will result in additional contamination added to the coal and increased costs.

Due to the old workings and spontaneous combustion a buffer zone needs to be blasted. The coal pillars must therefore also get blasted. The idea is to shatter the coal pillars and level, or spread out the blasted coal seam into the mined out void, creating an undulating surface. This improves the effectiveness of the dragline in cleaning off the overburden material rather than trying to dig out the waste between hard pillars. In order to reduce the risk of contamination it is important to create an optimal digging surface for the dragline.

Figure 3.13 shows an example of a hard coal pillar which was not blasted. This pillar is still solid and too hard for the coal loading machine to break and load. It will therefore require secondary drilling and blasting of this pillar.

Figure 3.13 - Example of Hard pillar Remaining

Dragline (overburden and interburden removal):

The draglines are the main equipment used to remove overburden and interburden material. This is due to their capacity to move large volumes of material. This is calculated in terms of Bank cubic meters (BCM's) and is a volumetric measure. The bucket size and capabilities of these machines range from 21 to 58 cubic metres which relates to a monthly capacity (budget) of ~450 000 to 1 300 000 BCM's.

Due to the size of the equipment they have certain limitations as far as the sensitivity or accuracy of the surface that they work to. This means that variances of approximately 0.5m are acceptable for elevation accuracy of the digging surface. But the No.4 Seam is only 2.0m thick, therefore a big loss can be incurred if the operator does not work carefully. The opposite will cause a capping of contamination.

In previously unmined areas a hard top of coal surface is created, which the dragline can dig down to. However any overdrilling and overblasting will break up the coal, making scalping possible. Underdrilling or blasting will again cause a hard capping which the dragline will not be able to remove.

In areas of poor roof coal the drilling and blasting go below the top of coal contact to the base of roof coal or top of mineable coal horizon. The material now looks very similar and this makes it difficult for visual identification by the dragline operator. Remembering that he sits in a cab anything from 30 to 50m away from the actual surface he is cleaning or working to. This makes drilling and blasting very crucial so that the correct "hard surface" for the dragline to dig down to is created. Since visual differentiation will not be possible for the operator.

The alternative will be to have a "spotter" or geologist on shift to guide the operator. Another option is to investigate the use of a survey instrument fixed to the dragline bucket to indicate to the operator what elevation to "dig" to.

The two edges of the cut or excavation are very important to ensure that all the coal is exposed and to prevent contamination from the edges and coal losses under spoils.

The general practice is to dig a void on the low wall side down to the base of the mining horizon. This also delineates the coal edge of the previously mined cut.

If the void is absent it could lead to low wall berm losses, which is when coal is left under spoils. It also leads to additional contamination from material "sloughing" on the coal edge from the low wall spoils.

Therefore, good practice on drilling, blasting and draglines can prevent unnecessary coal losses and additional contamination.

<u>Coaling:</u>

The coaling operation is at the end of the activity cycle or mining sequence and therefore suffers under the accumulated errors of poor practice. However, this is the operation that reaps the benefits for all the previous efforts, namely the coal. Therefore it is critical to take out everything that was planned for (mining extraction) with as little as possible contamination added.

The main impact of contamination on coaling is the unnecessary hauling of non-paying material (contamination or dilution). This reduces the overall yield, and increases the loading, hauling, beneficiation, waste management and rehabilitation cost.

ROM = Coal + contamination and moisture

Saleable Coal = ROM – (contamination + waste coal)

Figure 3.14 shows the weekly variation in the percentage contamination for 2005. It is clear from this that there are large changes from week to week, which is a function of the area mined. The contamination will be low when mining previously unmined areas and high when mining previously mined areas. Variations will occur relating to the mix and blending of coal from different areas and coal seams. Even with all the weekly fluctuations during the year, the annual average planned and actual achieved contamination for 2005 are very close to the predicted budget value of 8.0 percent.

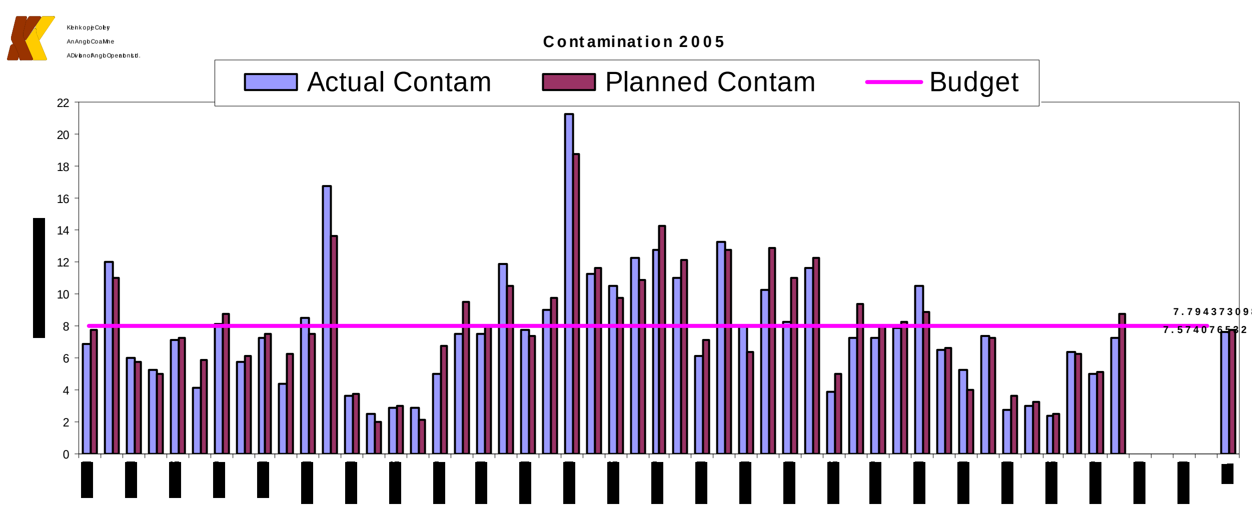


Figure 3.14 – Contamination Percentages in 2005

3.6.4. FINES GENERATION

Fines generation is a function of the material mined and mechanisms employed to do so (Scott, 1997).

The highest fines generation is attributed to the blasting activity. Factors that can result in increased fines generation are:

a) Increased blasthole diameter

b) Water in blasthole

c) Increased burden distance

d) Increased sub-drilling

e) Decreased stemming height

f) Decreased blasthole spacing

Factors that reduce fines are "V"-type initiation patterns rather than "in-line" firings and the use of decked charges. The use of decoupled low density, and low velocity of detonation packaged products in dry blastholes may decrease fines generation.

There is a direct relation between the increase in fines and increased powder factors, as illustrated in Figure 3.15 (pg28, figure 6, Lordford Darkwah report).

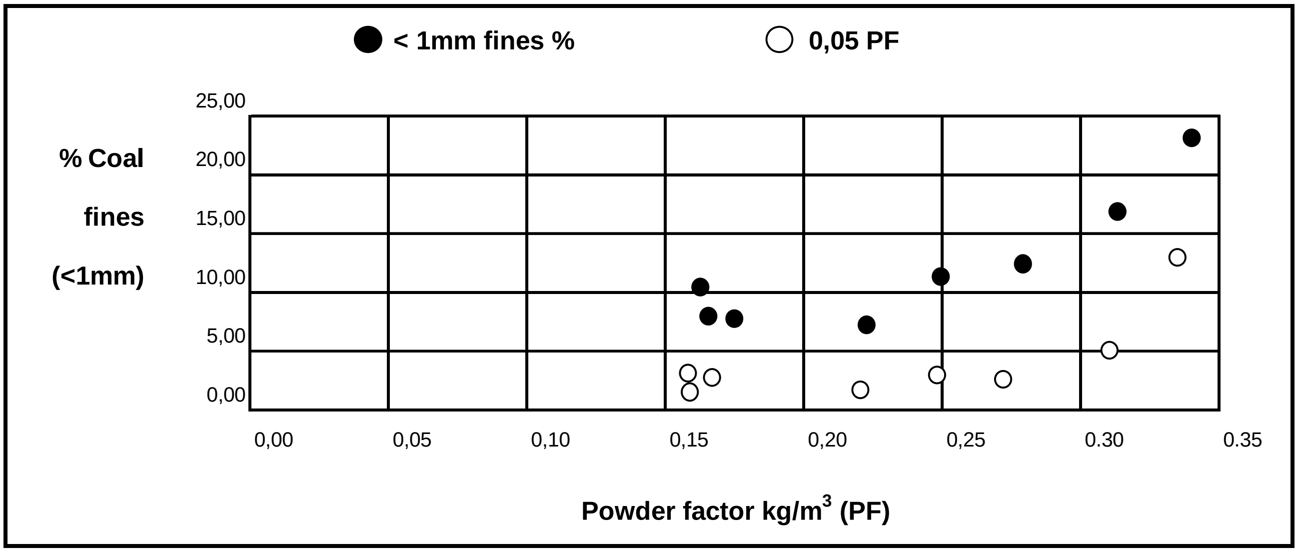


Figure 3.15 - Fines Generation and Powder Factors

(Modified after Slaughter et al, 1992)

During the mining process fines are also lost due to the difficulty of recovery in pit, and losses of coal dust due to wind action. Wet in-pit conditions result in fine coal being washed away by water.

The beneficiation of fine coal is less efficient and more expensive. Fine coal is difficult to separate due to the failure of gravity methods for small low mass particles. Fines also retain high moisture and reduce the overall heat value.

Possible fines generation points and activities are shown in Table 3.6: Generation of fines through coal handling is shown in Figure 3.16.

Table 3.6 - Fines generation points and activities

Section	Activity	Influence
Mining	Overburden blasting	- explosives
	Overburden removal	- dragline
	Coal blasting	- cut-off, total seam
	Parting blasting	- competent sandstone
	Coal blasting	-
	Coal loading	-
	Coal hauling	-
Stockpiles/ Tips	Intermediate stockpiles	- wet coal
	ROM stockpile at tip	- wet coal, tip problem
	Directly into tip	- pushed through grizzly
Plant	Breakers	-
	ROM stockpiles A&B	- automatic stacker
	Reclaiming	-
	Silo's	-
	Screening	-
	Conveyor belt tipping	-
	Crusher	-

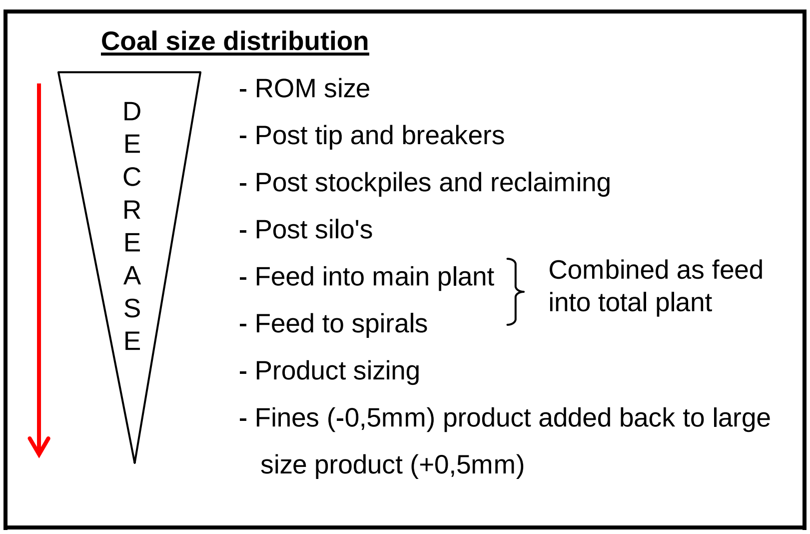


Figure 3.16 – Coal size distribution

The cost implications are as follows:

a) Ratio of 3:1: for every 3 percent fines generated only 1 percent is recovered as product.
b) Total product moisture increases as the fines percentage increases.
c) Handling of excessive fine coal is problematic, due to dust suppression and equipment blockages.

In order to manage the problem of fines generation it is necessary to measure it accurately. A modeling tool is required to identify and quantify the generation and model the processes to reduce fines (Scott, 1997)

3.6.5. SPONTANEOUS COMBUSTION

Spontaneous combustion can develop in oxygen concentrations around 10% and active heatings may be sustained at levels as low as 6% oxygen. Smouldering combustion can be maintained at a concentration as low as 2%. At any time or season the underground workings at Kleinkopje have atmospheric oxygen levels ranging between 20% and 22% (CSIR Report, 1999).

The orientation of the opencast mining cut in relation to the old working panel layout also plays a role. Where old barrier pillars exist parallel to the highwall direction it helps to prevent the air movement into and out of old workings. Cracks in the overburden and interburden due to pre-split and grid pattern blasting also increases the chances of air movement and can lead to a “chimney effect”. Open boreholes and drill holes also create airflow, propagating spontaneous combustion.

This is a very large topic in its own right and this treatise would only concentrate on the aspects affecting the current neighboring pits and future planned mining in this specific area.

Spontaneous combustion of coal occurs when the rate of heat generation is greater than the rate at which the heat is dissipated or removed. The result is an increase in temperature until it reaches a point at which sufficient heat is available for self combustion. Combustion requires three pre-requisites namely heat, fuel and oxygen. Spontaneous combustion cannot occur without one of these factors as per Figure 3.17.

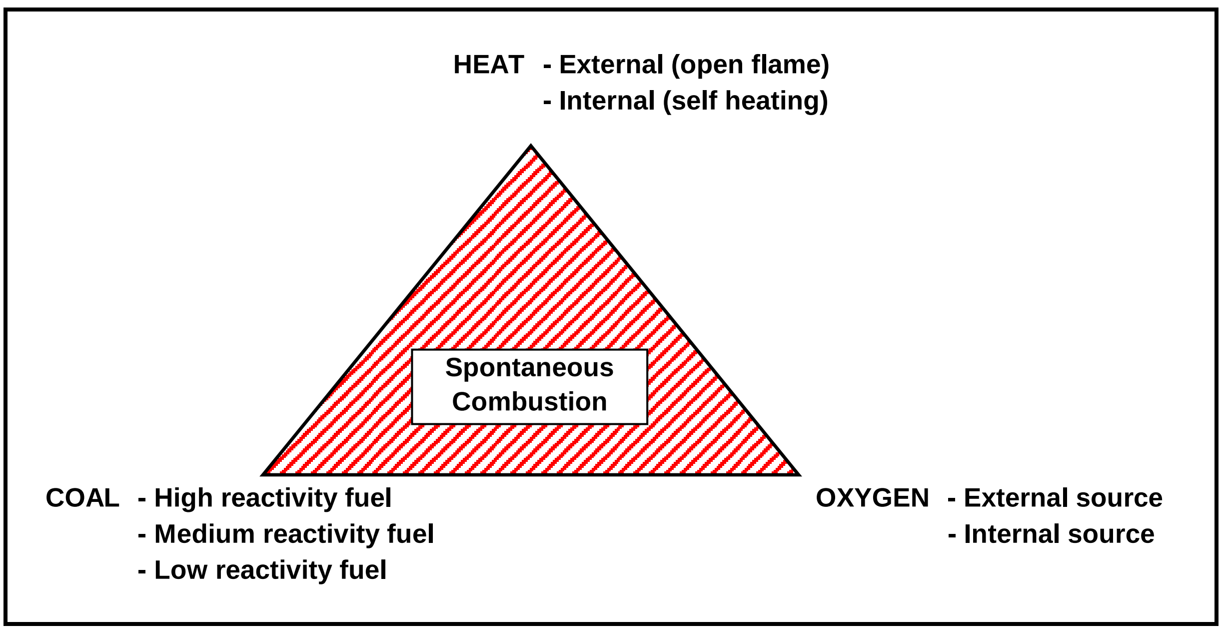


Figure 3.17 - Spontaneous Combustion Triangle

(Adopted from Falcon, 2004)

Spontaneous combustion has occurred in opencast and underground collieries, stockpiles, discard dumps, trains, trucks and ships. It is therefore not related to a specific area but rather to optimal conditions due to several contributing factors.

Over the years a number of theories evolved, namely:

a) Pyrite theory

b) Bacterial theory

c) Humidity theory

d) Oxidation theory

Pyrite theory

Already in the early 1600's, researchers were investigating the causes of spontaneous combustion. One of these was Dr Plott who published, in 1686, material on the phenomena of spontaneous combustion (Gouws, 1992). He attributed the cause to the presence of pyrite in coal and this theory was upheld since. Current thinking suggests that pyrite is only a contributory factor.

Bacterial theory

The bacterial theory originated from work done on hay, but proved not to be applicable to coal.

Humidity theory

The humidity theory is based on the principle of "heat of wetting". This causes an increase in temperature due to the addition of moisture to the coal particle as in Figure 3.18.

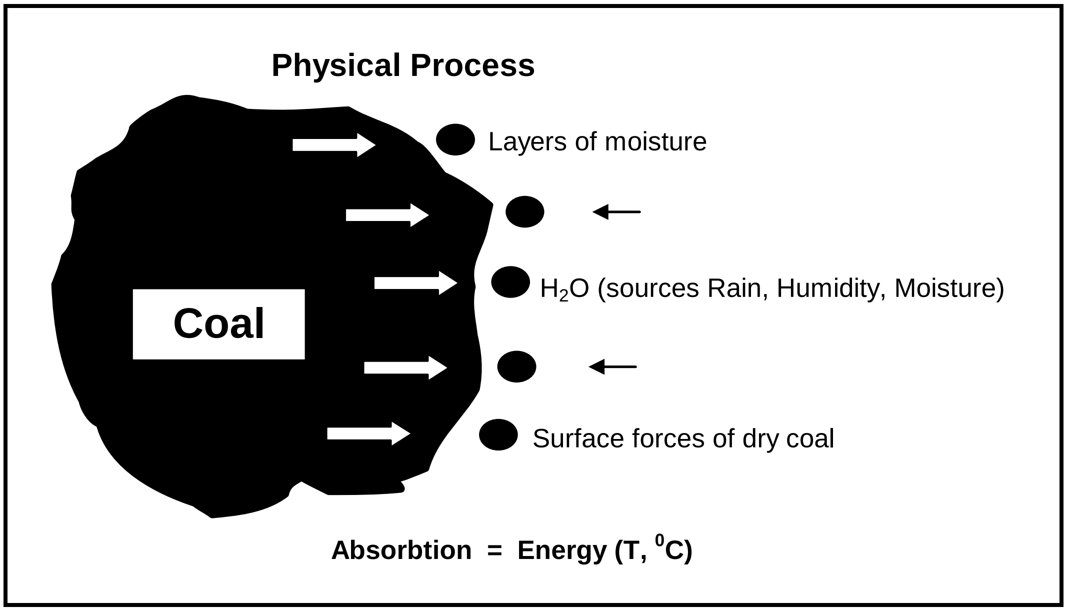


Figure 3.18 - Physical Process

(Adopted from Glasser, 2004)

This process is a physical process driven by the surface forces of dry coal and once equilibrium is achieved this process will stop.

Oxidation theory

The latest theory is the “oxidation theory” which is now the generally accepted one.

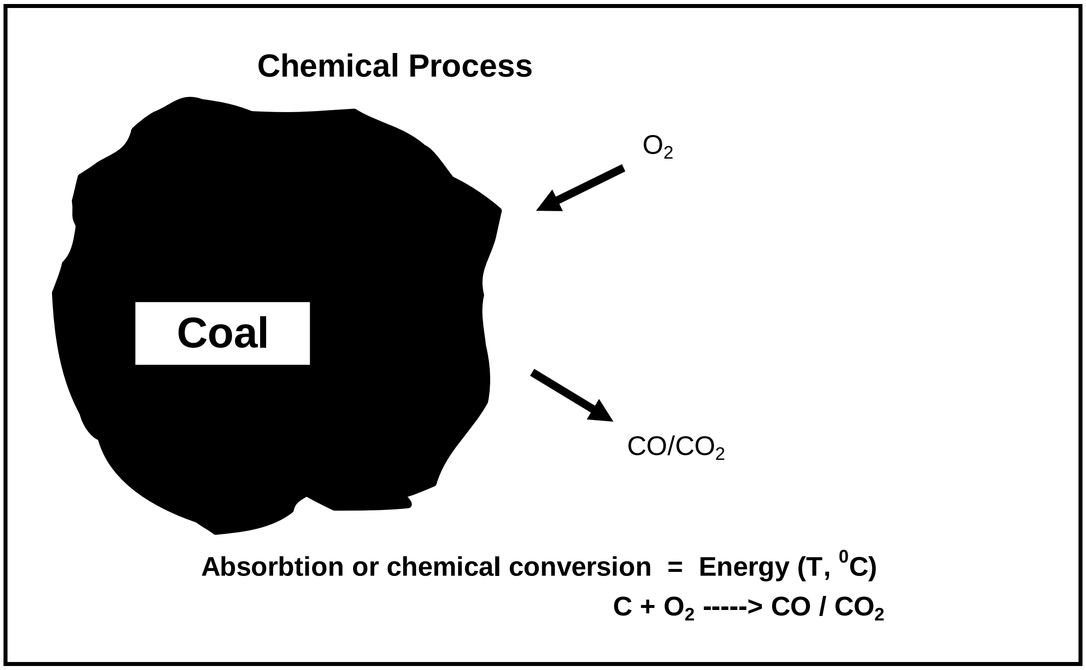


Figure 3.19 - Chemical Process

(Adopted from Glasser, 2004)

This is a chemical process driven by the absorption of oxygen by coal which can be readily oxidized at ambient temperature, seen in Figure 3.19. Oxygen is necessary to sustain this process. The rate is determined by the availability of oxygen, temperature and availability of a surface area that can be oxidized.

The heat of melting or humidity and the pyrite content are the trigger mechanisms which initiates the self heating process. The temperature increases so that exothermic oxidation generates heat faster than can be dissipated through normal convection and conduction.

The risk of spontaneous combustion of coal can be evaluated using the following equation:

<u>Total Risk</u> = Coal factor x Geological factor x Mining factor

This is based on both intrinsic factors (coal and geology) and extrinsic factors (mining). Various experimental techniques are available to determine the propensity of coal for spontaneous combustion. Two techniques were identified (Gouws and Phillips, 1993) as the most suitable, namely determining the ignition temperature and the adiabatic calorimetry tests.

Ignition temperature tests were done at the Kleinkopje Colliery to indicate the intrinsic risk of the coal to spontaneously combust (Eroglu, 2000). This test included:

a) Crossing-point temperature test (X.P.T.)

b) Differential thermal analysis (D.T.A.)

The above tests were run concurrently. The crossing-point test involves a coal sample and a sample of inert material which are both housed in identical sample holders and placed in an oil bath.

The oil is heated at a constant rate, until it reached the crossing point temperature where the coal temperature equals the inert material temperature. The differential thermal analysis (DTA) is the temperature difference between the coal and the inert material (temperature of the coal minus the temperature of the inert material). This is measured and plotted against the temperature of the inert material as shown in Figure 3.20.

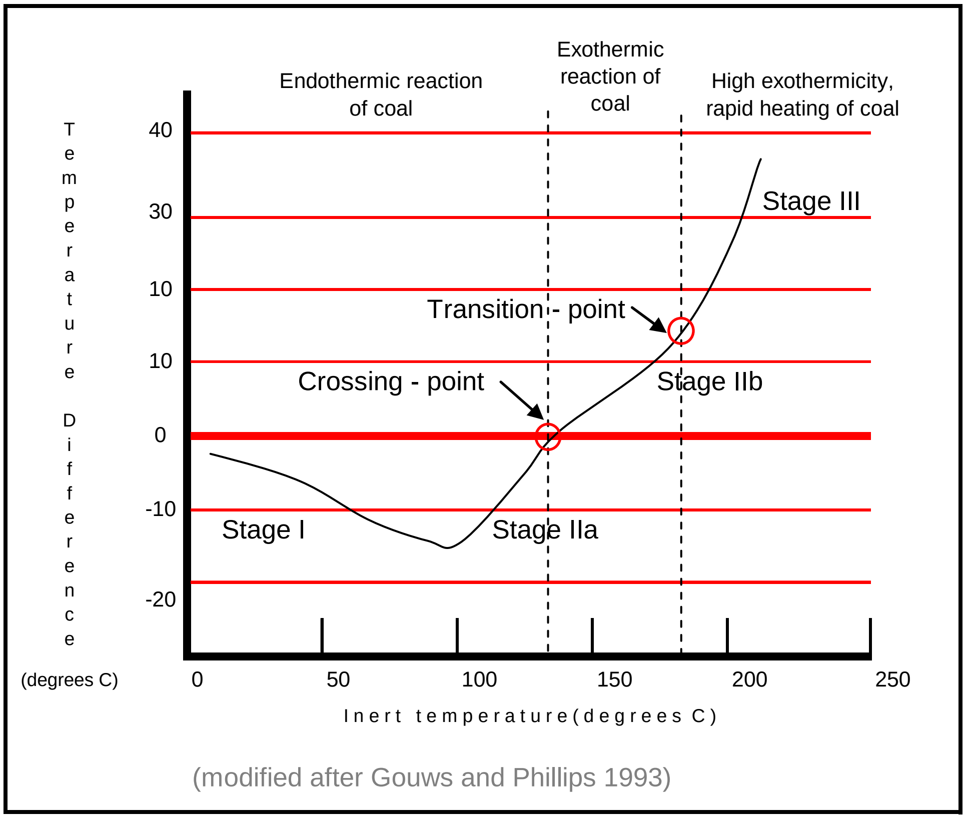


Figure 3.20 - Differential Thermal Analysis

The DTA thermogram can be used to indicate a number of characteristics of coal to self heat. They are termed simple indices:

a) A low crossing point temperature

b) A steep stage II slope

c) A low transition temperature between stages II and III

General trends are that in stage I the inert material temperature is higher than the coal temperature because of the cooling effect of the evaporation of moisture in the coal (endothermic reaction). After the moisture has evaporated the coal heats up at a faster rate than the inert material (stage II) because of both its tendency to self heat (exothermic reaction) and its attempt to reach the bath temperature. Stage III represents the high exothermicity. The point where the line crosses the zero base is called the crossing-point temperature.

A lower crossing point temperature and steeper stage II slope indicates an increase in spontaneous combustion liability. Composite indices is the combination of the simple indices. However both individual and composite indices were found not a reliable indicator of spontaneous combustion (Gouws, 1992).

The WITS–EHAC index (named after the major sponsors) was developed using both X.P.T. and D.T.A. factors. This new index successfully indicates the propensity of different coals for spontaneous combustion.

The index is defined as:

WITS-EHAC Index = (stage II slope / crossing-point temperature) * 500

A coal with a higher WITS-EHAC Index number has a greater tendency to self heat than a coal with a lower index number as illustrated in Table 3.7.

Table 3.7 - WITS EHAC Index

WITS-EHAC Index	Spontaneous Combustion Liability
0-3	Low
3-5	Medium
>5	High

(Eroglu, 2000)

The samples analysed from Kleinkopje had the following results as shown in Table 3.8 below. Also included are results from the neighboring South African Coal Estates mines.

Table 3.8 - Sample Data

Sample	WITS-EHAC Index	Spontaneous Combustion
Kleinkopje M2S	4.75	Upper Medium Risk
Kleinkopje M2T	4.33	Medium Risk
Kleinkopje Roof Shale	2.85	Low Risk
Greenside M2S	5.24	High Risk
Greenside Roof Shale	3.34	Medium Risk
Landau M2S	4.88	Medium High Risk
Landau M2S	4.57	Medium High Risk

(Eroglu, 2000)

This puts the Kleinkopje values in perspective and gives a better idea of the larger area. Trends noticeable are that all the M2S values fall within the medium high to high risk category. The M2T was slightly lower in the medium risk category with the roof shale in the medium to low risk range. Another general observation is that there are usually more disseminated pyrite and pyrite nodules present within the M2S zone. The total risk of spontaneous combustion is the product of coal, geological and mining factors. The sample results indicate that focus is required on the mining risk in order to reduce the overall risk. A starting point would be to increase the data base with more samples from different mining areas.

After the origin of spontaneous combustion and the propensity of different coal types to self heat, have been addressed in the paragraphs above, the sequence of events up to the ignition and burning of the coal and the effect on mining and reserves have to be considered. Also important is what steps can be taken to prevent spontaneous combustion or to manage it. The sequence of events from the heating of the coal to ignition can be explained by Figure 3.21, modified after Falcon,2004.

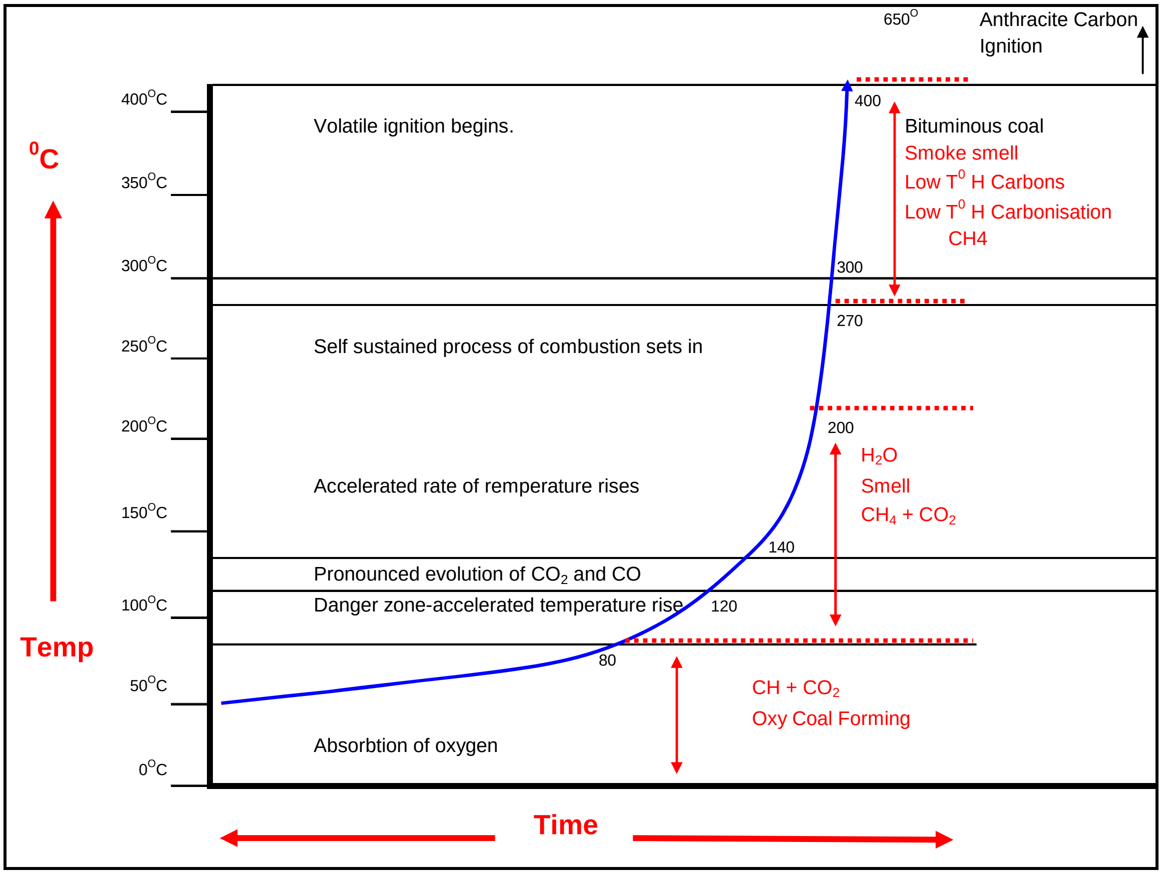


Figure 3.21 - Curve of Heating and Coal to Ignition

(Adopted from Falcon, 2004)

The process or sequence starts off with trigger conditions which may be heat of weathering or pyrite oxidation. This will then initiate the oxidation process of coal.

This heating can be speeded up by wind, size segregation, climate, moisture, chimney effect, etc. Depending on the inherent characteristics of the coal which range from least reactive to most reactive, this process will take a related time period. It starts off at ~50^{0}C to 80^{0}C and eventually ignition occurs at ~400^{0}C (Falcon, 2004).

The inherent factors of the Kleinkopje coal including type, rank, grade and weathering place it in the medium to rapid self heating range. These intrinsic factors i.e. coal and geological factors are determined by laboratory analysis and testing. From this the Kleinkopje coal is classified in the medium to medium high range of self heating (Eroglu, 2000).

Type :	Humic coal
Rank :	Bituminous medium
Grade :	Mineral content
Weathering :	Varying due to depth

The external factors are related to the mining environment and this will determine the size range, mixing, handling and surface moisture of the coal.

The mining process and sequence of activities are discussed under the mining method (chapter 2). The beneficiation process also impacts on the opportunity or likelihood of spontaneous combustion.

Table 3.9 lists the steps to curb spontaneous combustion.

Table 3.9 - Steps to Curb Spontaneous Combustion

Plug all holes	- geological	cement them
	- pre split	plastic cup
	- pre split	fill/cover with soil after blast
	- pattern	plastic cup
	- pattern	cover with soil cladding after blast
	- T/C area	level and clad
		leave blasting to latest
Improve fragmentation		
Reduce unnecessary cracks		
Clean break needed		
Deck charge		
Gasbags		
Can buffer blasting and cladding reduce oxygen levels?		

Areas to focus on in order to improve spontaneous combustion management:

a) Additional WITS EHAC testing on coal susceptibility for spontaneous combustion so that this information can be built into the geological model

b) Additional focus on hot spots in conjunction the common blanket approach of using cladding, water cannons and buffer blasting everywhere

c) Drilling and blasting techniques and the degree of fragmentation of the overburden and interburden need to be investigated

d) Hard coal pillars should be prevented as this results in poor sealing and higher incidents of spontaneous combustion

e) Barrier pillars must be used as fire breaks and non permeable barriers

f) All old shafts should be properly sealed to limit airflow and source of oxygen

g) Focus needed on plugging of drill holes, pre-split holes and geological boreholes to reduce chimney effect

h) Reinvestigate the buffer zone thickness

i) Mining configuration needs attention in areas of cut width, cut length, rate of mining and scheduling of activities relating to the standing time of buffer zones

j) Thermal imaging of old areas on an annual or 6 monthly basis to see change

k) Ongoing temperature monitoring of strategically placed boreholes and blast holes

4. BENEFICIATION

4.1. INTRODUCTION

Coal may be beneficiated to produce a product at a specific quality i.e. CV, ash, volatile, moisture etc. Depending on the product quality requirement and the quality coal feed into the plant, a certain portion of the coal is discarded and the remainder kept as saleable product.

This is a trade-off between the product quality required and the yield or amount of saleable product available to market. The premium on the price of higher quality product needs to be able to compensate for the reduction in yield or amount of saleable tons produced at the higher quality specification.

The range of in-situ coal quality and characteristics of coal allows it to be beneficiated to a specified product grade, and distinguishes it from other mineral commodities. In addition to this we now add the coal price variance and demand for different product qualities and the end result is a very complex and financially sensitive commodity.

The beneficiation plant’s area of responsibility starts at the Run of Mine (ROM) tip, then follows the actual plant process itself and ends in either saleable product in the product silo or alternatively discard to the discard dumps, also preferably called a co-disposal facility. The saleable export product is transported from Kleinkopje Colliery by conveyor to the Rapid Loading Terminal (RLT) from which it is loaded on the train to Richards Bay Coal Terminal (RBCT). At RBCT it gets placed on the product stockpile for the specific coal brand, awaiting shipping to the customer. The product quality is checked at the mine, by the plant laboratory and at Anglo Coal Coal Laboratory (ACCL) in

Witbank. At the RBCT laboratory (SABS managed) they also determine an in-go and out-go coal quality, which can be compared to what the plant expected. Ultimately the customer will also analyze the product received to determine whether it complies with the agreed quality specifications, as negotiated by the producer's marketing representative.

4.2. THE BENEFICIATION PROCESS

The ROM coal at Kleinkopje Colliery is tipped at one of three tips named A, B and C. From where it then enters the beneficiation process. First it goes through two rotary Bradford breakers, to reduce the size of the feed to plant coal (nominal –75mm select / -150mm non-select) and also to remove oversize non-coal material such as sandstone boulders. The select coal is placed on the B stockpile (40 000 tons capacity) and the non-select coal on the A stockpile (45 000 tons capacity) by means of automatic stackers. The C tip is part of a new stream, and uses an Osborn breaker crushing the product to -100mm nominal size. This coal can be stacked on either A or B stockpile.

The waste material from the breaker is fed straight to a jaw crusher which reduces it to -100mm size. This is then placed on either A or B as required. The raw coal is reclaimed from each individual stockpile using its own bucket wheel reclaimer, and fed to two separate banana screens. This then screens the coal at 12mm into a -12mm fine fraction cyclone feed and +12mm course fraction, drum feed.

The feed to plant (FTP) coal then gets processed along two streams, a non-select A-stream and a select B-stream. The original plant design was for high yielding previously unmined coal to be beneficiated along these two separate streams.

The A-stream is used for non-select coal to produce a steam coal with a calorific value (CV) of 27.45MJ/kg in a single stage wash or one pass process. The B-stream washes select coal through a double stage or two pass process, producing a Low Ash Coal (LAC) of 7,0% ash and middlings Steam Coal (SC) product. Due to changes in the tip stockpile management strategy and the sourcing of the coal from the various pits and seams, the old system had to be modified accordingly. The plant now simultaneously produces SC from the A and B streams and a higher grade blend coal from the B stream only. This has improved the overall throughput and simplified the system.

The run of mine (ROM) coal particle size distribution varies depending on the source area, specifically the mix between coal from previously unmined areas and mined areas. The > 12mm to < 75mm fraction is beneficiated in a Wemco drum which is a static bath type separation. The > 0.5mm to < 12mm fraction is beneficiated through centrifugal separation, using dense medium cyclones. The smaller size fraction, ranging from minus 0.5mm to plus 250 micron, called fines is washed using spirals. This is a centrifugal and gravitational separation method, but is passive and only results in a minor quality upgrade. The minus 250 micron material, called ultra fines are upgraded using froth floatation technology.

In the plant, hourly samples are taken of the product conveyors and ash and calorific value analysis are performed. This enables the process operators to track the plant performance and product quality. The wash medium is a mixture of water and magnetite and its density is adjusted to separate at the correct cut-point density to achieve the prescribed quality and yield.

The fines product is produced at a lower calorific value and higher surface moisture than the coarser fraction. This is due to the lower efficiencies of spirals and froth floatation cells. The addition of the fines product will lower the average product quality and therefore the "coarser" fraction needs to be over-washed at a lower cut-point density to compensate. However an overall product tonnage benefit from the addition of fines must be achieved, otherwise the fines are dumped. Dumping of fines means that it is not added to the steam coal product, due to its poor quality, shown in Figure 4.1. The fines product have a calorific value (CV) ranging from 23 MJ/kg to 25MJ/kg. This is sold to the inland market on an ad hoc basis, depending on demand.

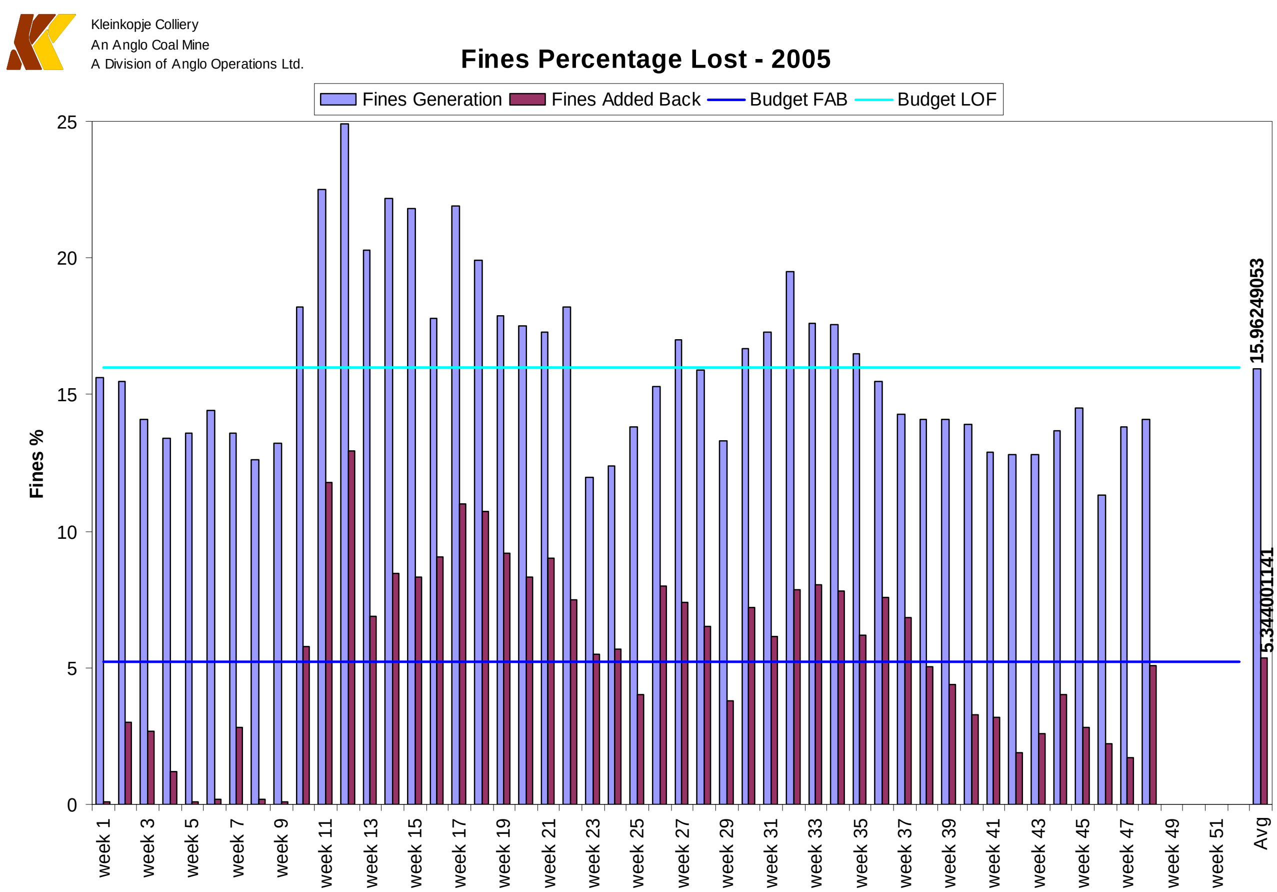


Figure 4.1 – Weekly Variation in Fines Percentage

The financial implications are as follows:

a) Fines increase the overall product tonnage which generates more revenue

b) It is still better to generate minimum fines during the mining process as only a limited amount of fines can be carried by the course fraction

At the end of the day it is a balance between maximizing the product tons and producing it within the required quality specifications (see Table 4.1).

Table 4.1 - Coal Product Qualities

Product Qualities	Product Types: Greenside Blend (GRN)	Product Types: Steam Coal (SC) Export and Inland
Calorific Value MJ/kg	28.20	27.45
Ash %	13.50	15.50
Volatile %	~25	~23

* Inland SC is identical to export SC product qualities, but excludes the spiral fines product that is added back.

4.3. BENEFICIATION PROBLEMS ASSOCIATED WITH OLD AREAS

Due to the additional contamination included in the ROM tons, the plant will need to be able to handle this. It will firstly impact on the tip area and primary breakers and crushers, but also on the capacity of the discard belt, bin and eventually on the discard disposal facility.

Mining of old workings also increase the risk of scrap metal and old conveyor belt ending up in the feed into the ROM tips. This can cause blockages and damage to equipment resulting in increased downtime and maintenance costs. The only means of managing this problem is the installation of large belt magnets which extracts or separate the scrap metal from the coal feed. Sacrificial belts are also installed along this route to minimize the cost of belt damage.

Although it is not allowable in terms of the standard procedure, hot spontaneous combusting material are occasionally delivered to the tip areas. This material will cause damage to belts and result in downtime and unplanned maintenance. The heating has negatively affected the coal quality and plant yield.

4.3.1. CONTAMINATION

The percentage contamination is estimated on a daily basis as part of the grade control procedure. It is done for every coal face separately, and then combined in the appropriate percentages and run through the Gradecon model. This information is given to the plant on a daily basis to assist them in the running of the beneficiation plant.

Previously, contamination was added and calculated as a fixed percentage per area and not related back to the variation in seam thickness. Currently it is calculated on the basis of a contamination thickness added to the mineable horizon thickness. However this needs to be weighted by the relative density of the material as illustrated in Figure 4.2.

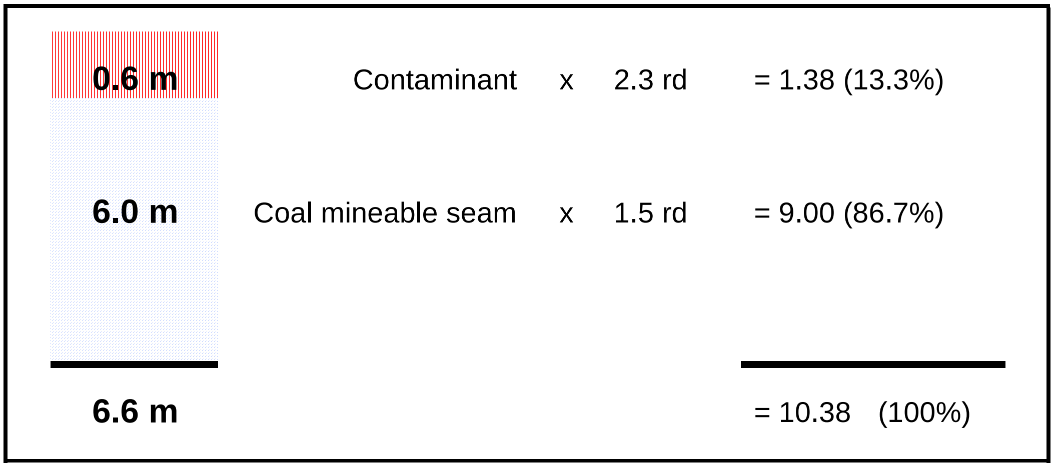


Figure 4.2 - Weighted Contamination Calculation

4.3.2. PERCENTAGE FINES

Fines in the coal industry are generally the term used for coal particles that are smaller than 0.5mm or 500micron. Most plants are not able to treat this size fraction of material in its main stream of coal throughput. Therefore it gets screened off at the beginning of the process and treated in a separate circuit.

The fine coal that gets generated is a consequence of the mining practice i.e. drilling and blasting, cleaning and loading, hauling and tipping, and going through all the transfer points on belts. This material gets removed at the start of the beneficiation circuit and is subtracted from the total feed to plant. The percentage can vary from 14% up to 17% of the ROM feed to the plant.

Every additional percent fines reduces the feed to plant (FTP) into the +0.5mm circuit by 1% and reduces the overall yield. Therefore we need to eliminate unnecessary fines generation.

Listed below are the main contributors to increased fines generation:

a) Over blasting

b) Incorrect drilling depth

c) Incorrect backfill

d) Tramming on coal

e) ROM stockpile leveling

f) Intermediate stockpiles

g) Loading on cut-offs

h) Spontaneous combustion

i) Water

Fines are normally created through the handling of the coal. The first question would be what is the susceptibility of different coal types or grades for the formation of fines. Generally the higher grade coals have more vitrinite and less inertinite, liptinite and ash particles, and they tend to degenerate or break up more readily. The poorer grades are richer in impurities with an overall higher density which requires more energy to break or fracture.

Another factor is the depth of the coal seam, with the shallower coal seams being softer and more weathered than the deeper coal seams. Generally the No.4 seam is softer than No.2 and 1 seam coals which are older coals.

4.3.3. SPONTANEOUS COMBUSTION

The effect of the spontaneous combustion on the actual coal quality and other physical properties still requires further study and clarification. There are visible signs that the coal properties have changed due to excessive heat exposure. It has almost been “coked” with visible signs of melting or fluidity structures on samples taken at the face. The change in the volatile percentage and calorific value of the coal over time need to be quantified.

The current impacts on the beneficiation process starts with the hot material entering the plant front end at the tip area. The heat affected and hot ROM feed generates significantly more dust when tipped and as it moves through the breakers and primary screening plants. The dust suppression systems were not designed for these conditions. The front end conveyors and plastic screen panels are burned and damaged by the hot material. The hot coal need to be treated as soon as possible otherwise it burns on the stockpiles and further deteriorates in quality.

Due to the negative effect of the heat on the volatiles and calorific value of the coal, it results in the plant not being able to achieve the predicted yield from the geological model. Depending on the severity of the spontaneous combustions effect on the coal, this variance between plant actual yield and geological predicted yield can range from approximately 2.0% up to approximately 6.0%. As stated before, it is almost impossible to quantify the effect of the heating and burning, on the coal quality and resulting yield, on a daily basis for reconciliation purposes.

In very severe cases carbonaceous shales have ended up on the product screens as misplaced material because of the heat that altered the shale’s material properties. The additional fine material also impacts negatively on the dense medium used in the beneficiation process.

4.4. MARKETING

The seaborne internationally traded thermal coal market can be geographically divided into the Mediterranean-Atlantic (Med-Atlantic) region and the Indian-Pacific (Indo-Pacific) region. The main importers in the Med-Atlantic region is the EU15 while on the Indo-Pacific side it is driven by Japan, Korea and Taiwan.

Anglo Coal's geographic distribution of operations and range of thermal and metallurgical coals allow it to supply various sectors. Both the global export markets and domestic markets are supplied in Australia and South Africa. A generic ranking of the Anglo Coal product range is shown in Figure 4.3 below.

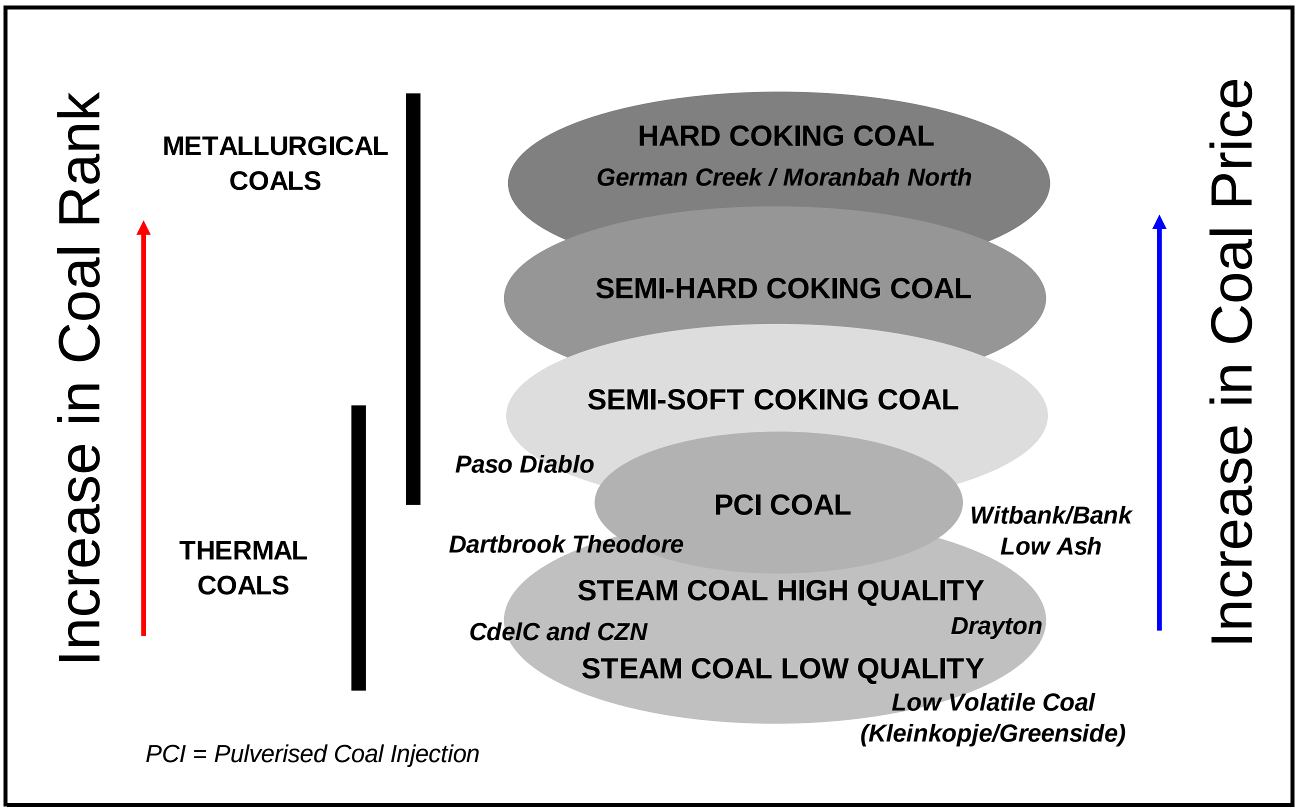


Figure 4.3 - Anglo Coal Product Range

(Adapted from Kleinkopje, 2001)

From this we see that low volatile thermal coal is situated at the lower end of the classification spectrum. The thermal coal export market has been grown since its inception in the late 1970's, with the low volatile thermal coal now well established.

The volatile thermal coal is also known as AAC brand or steam coal and forms the bulk of the production from Kleinkopje Colliery, approximately 64% of the total annual saleable tons produced. The coal from the Kleinkopje, North-West block will be mined and beneficiated to produce a low volatile thermal coal. The coal market is very sensitive to coal price in US$, shipping cost and in our case the Rand/US$ exchange rate. Fluctuations in the coal price are demonstrated in Figure 4.4 below, from 1985 to 2005 and a forecasted price up to 2010.

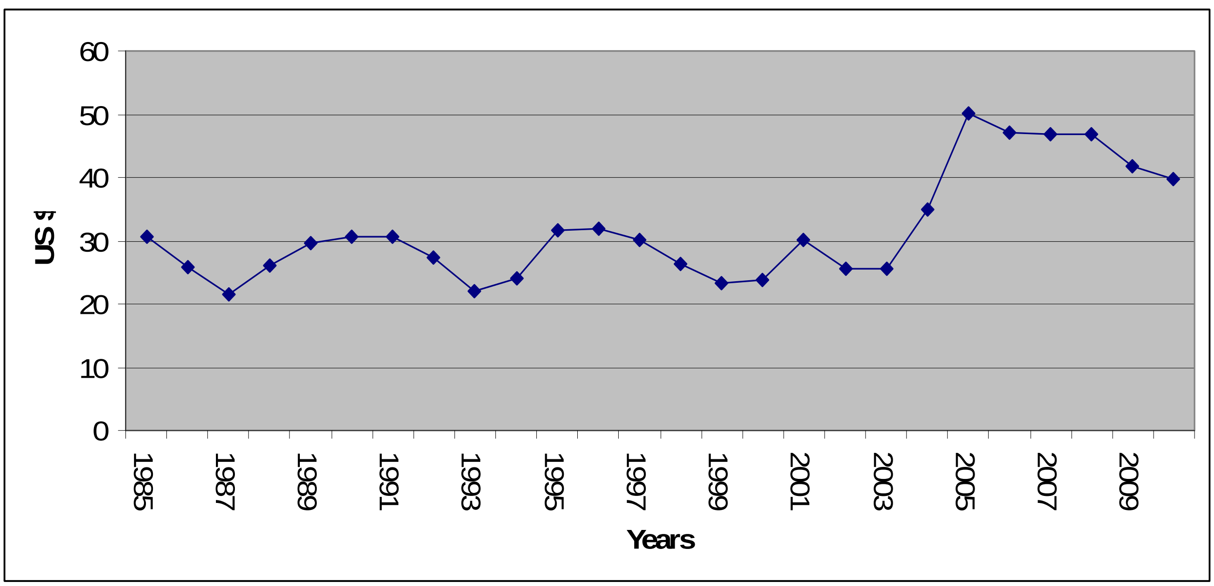


Figure 4.4 - Coal Price Evolution

(Adapted data from internal presentations)

Historically the price was cyclical over approximately seven years, peak to peak. At that stage long-term coal contracts were the norm with only a small fraction sold on the spot market. This has changed to a very short cycle from peak to peak of approximately three to four years. The spot market is much larger with a preference for shorter term contracts.

In 2004 an exceptional price increase was experienced, driven by high demand for especially metallurgical coals and also an increase in the thermal coal demand and prices. The coal production increased in North America and the European Union by approximately 3.6% and in the Asian Pacific and Russia by 13.4% and 18.3% respectively (South Africa's Mineral Industry 2004/2005). China, Australia, Indonesia and India were the main drivers for the Asian Pacific's increased output by supplying an additional combined 325 Mt. Unfortunately South Africa only increased production by 5 Mt or 2.0%. The steel industry has been the main driving force behind this.

Although the international coal price increased by 64.2% from 2003 to 2004 (South Africa's Mineral Industry 2004/2005), the South African Rand to US $ exchange dropped significantly during the same period, which reduced the positive impact. This illustrated in Figure 4.5.

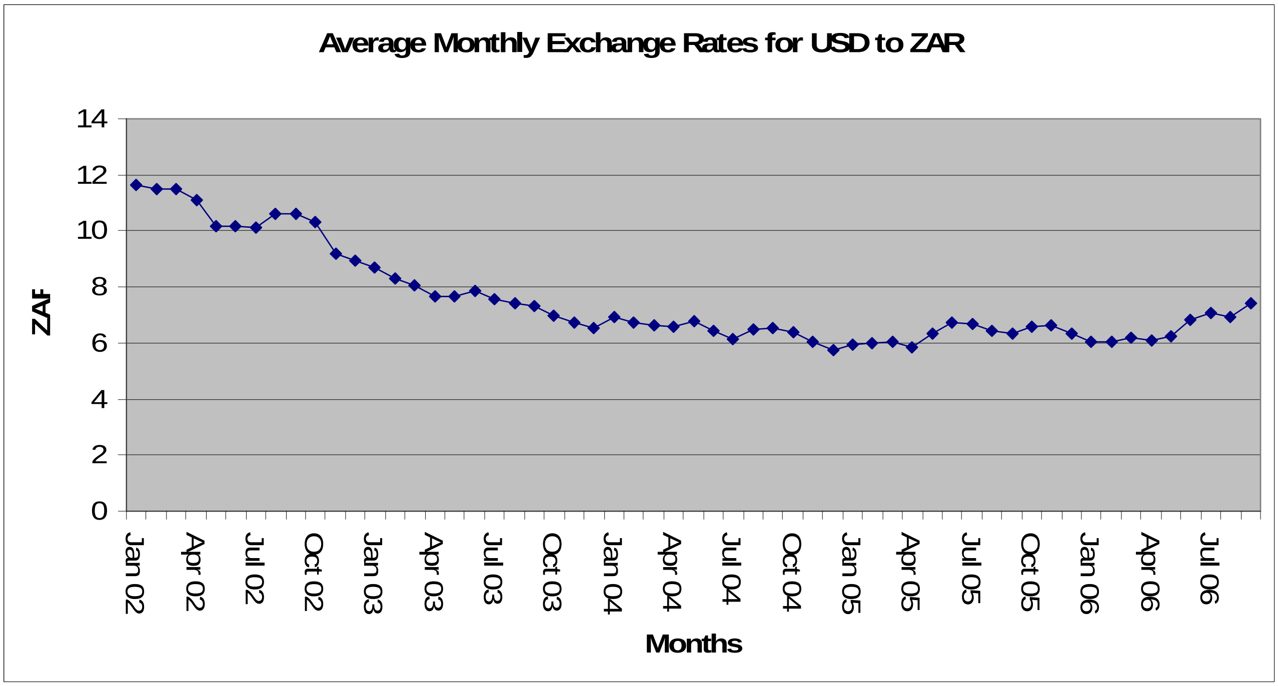


Figure 4.5 – Changes in exchange rate

(Sourced from www.gocurrency.com)

Exports from Anglo Coal's South African operations (ACSA) are routed via the Richards Bay Coal Terminal (RBCT). The total export capacity of this port is currently 72 Mt per annum but the pending expansion will increase this to 92 Mt per annum. The current ACSA entitlement at RBCT is 19.8 Mt per annum which will remain the same after the expansion with most of the new capacity going towards Black Economic Empowerment (BEE) companies. These companies obtained an RBCT export allocation of 3 Mt per annum for the period 2005/2006.

The South African coal benefits from the lower freight cost to Europe and the majority of our customers are power utilities and cement producers in Europe. The majority of our exports (82%) are to countries in the European Union. The main destinations are Spain, Great Britain, Italy, Germany and France. The supply contracts to Asian customers have been decreasing over time and have limited prospects of growth or renewal. This is due to price competitiveness playing a more significant role as purchasing criteria. Historically, regional diversification of supply played a role, allowing the South African producers into the Asian market. The only significant consumer of South African coal in the Asian region, is India, because it is geographically closer.

The global coal production and export is illustrated in Figure 4.6. This indicates that the Asian Pacific countries are the largest producers by far, with 59.4% of the global production. The global supply is dominated by Australia, Indonesia, China and South Africa.

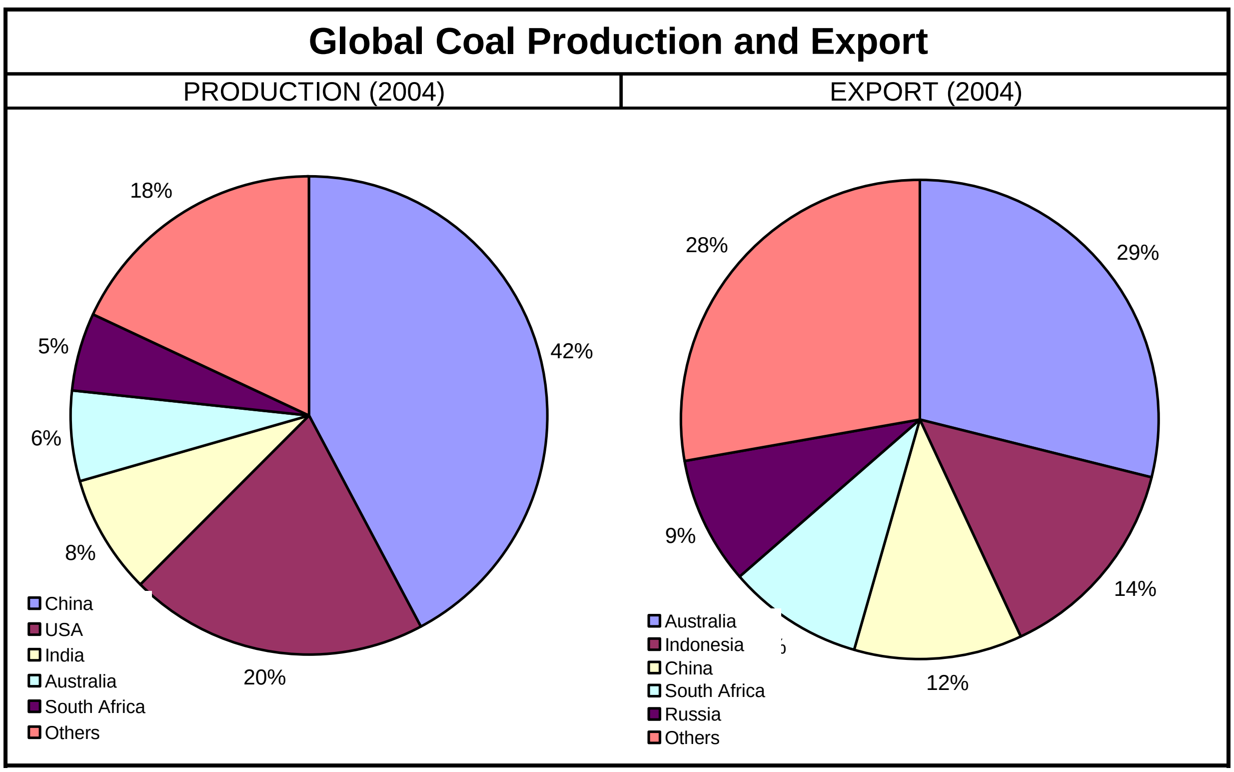


Figure 4.6 – Global Coal Production and Export

(South Africa's Mineral Industry 2004/2005)

A larger player in the global coal supply, such as China, is in a position to flood the market, due to the sheer volume it produces annually, 42.3% of the worlds production. Due to China's high growth rate, the domestic demand is equally large and most of the annual production is absorbed internally.

Anglo Coal was the largest coal producer in South Africa during 2005 with 56.5 Mt, followed by BHP Billiton with 54.5 Mt, Sasol 40.0 Mt, Eyesizwe 25.0 Mt, Xstrata 20.0 Mt and Kumba 18.0 Mt. In the global arena Anglo Coal is currently the sixth largest coal producer after Arch Coal, Shenhua, BHP Billiton, Rio Tinto and Peabody at number one.

5. RISK ANALYSIS

5.1. INTRODUCTION

This chapter will combine all the major risk factors that have already been discussed throughout this treatise, and some additional ones, and rank them according to their risk rating. The method used will start with the hazards identification using a hazard identification sheet, followed by the risk rating of these hazards using a risk matrix. The process will be focused on the North West block and will address the block specific risks, risks related to the mining method and operation and also the larger market and global risk environment.

The latest Anglo American plc risk matrix will be used to determine the risk rating for all the identified hazards. This risk matrix was developed as a common management tool to be used across all operations and replaces the multitude of previous systems. The objective was to create a more efficient integrated risk management system covering all key business functions and processes including safety, health, environment and the community (SHEC) and human resources (HR) and all other disciplines and functions.

Risk management is also the responsibility of line management and needs to be integrated into all the business processes to allow for risk based decision making in the planning and execution. This needs to be applied to all mines, departments and projects such as the North West block. It will allow for performance management against key performance indicators (KPI's) and provide assurance that risks are managed within the continuously changing business environment.

In the current global business environment a company do not want to be risk adverse but rather risk intelligent. This will enable the benefiting from opportunities and mitigation of risk. In the case of the North West block the identified hazards and appropriately ranked risks need to be integrated into the business plan, objectives and targets. This plan must also consider the changes in the risk environment and have a system for review and action. This will require a structure for ownership and responsibility as illustrated in Figure 5.1. It is illustrated as a pyramid with the employees at the base, followed by the risk managers which are the operations managers and then the risk owner, which is the general manager. The same structure applies to the divisional, corporate and global levels with the ultimate risk owner being the chief executive officer (CEO) and board of directors.

Each role player has specific responsibilities:

Employees: They need to participate in the daily identification of risks and have a general safety responsibility to themselves and their co-workers. They are required to comply with risk management practices and directives.

Risk Managers: They are responsible for the daily frontline risk management by facilitating and participating in hazard identification, risk assessment and the development of action plans. They are also tasked with the ongoing monitoring and measuring of the effectiveness of the risk management.

Risk Coordinator: They have no direct line responsibility and ownership but are required to identify and report specific divisional and plc risks. Their function is the oversight and promotion of learning consistency within the company.

Risk Owner: They are ultimately accountable for quality and performance of the risk management within the company, division, operation or project. They have to create, define and champion the risk management policy. It is their responsibility to delegate and assign the responsibility of the individual headline risk areas to specific owners. See Figure 5.1 below.

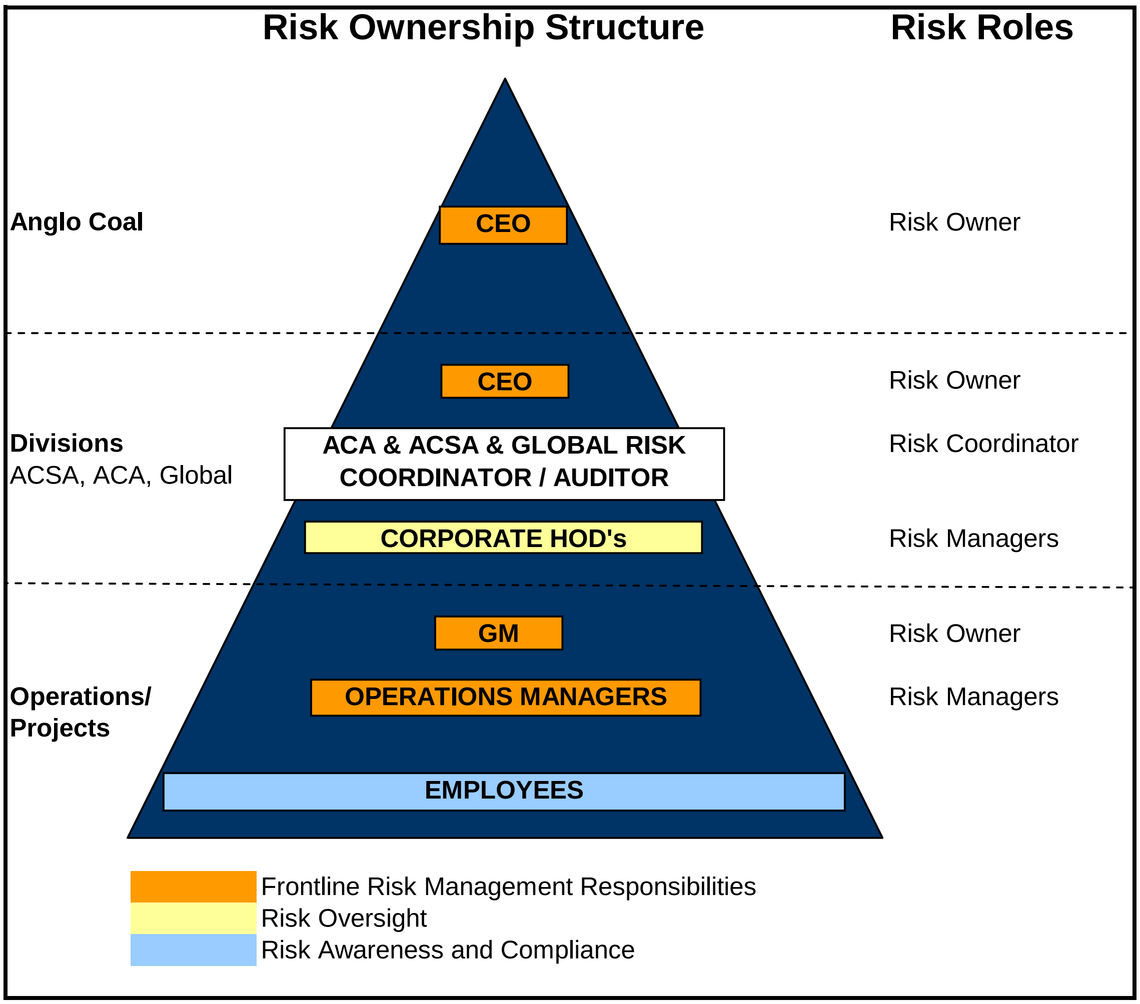


Figure 5.1 – Risk Ownership Structure

(Adapted from internal document)

5.2. RISK MATRIX AND RISK RATING

The headline risk areas identified within Anglo Coal are reviewed and revised on a continuous basis. Currently there are 18 headline risk areas (HRA's) identified. All these HRA's were investigated to determine whether they are applicable to the North West block and the main risks were then highlighted as shown in Table 5.1.

Table 5.1 - Headline Risk Areas applicable to the North West block

Headline Risk Areas		
Operation / Project: Kleinkopje Colliery, North West block		
Headline Risk Area	**Applicable**	**Not applicable**
Employee Safety	X	
Commodity Price	X	
Technology	X	
Infrastructure	X	
Operational Performance	X	
Counterparty	X	
Event Risk	X	
Legal / Regulatory / Finance	X	
Reserves and Resources	X	
Employees	X	
Political	X	
Mergers and Acquisitions	X	
Social	X	
Capital Projects	X	
Treasury	X	
Environment	X	
Foreign Exchange	X	
Employee Health	X	

These HRA's that have been identified as applicable need to be further assessed to determine a risk rating for each. It is important to note that some of these risks are on a global level which is not manageable while others are on an operational level which can be managed. An example of a non manageable HRA is **Foreign Exchange** whilst **Operational Performance** is a manageable HRA. The tool used for determining the risk rating, is the Anglo American plc Risk Matrix as illustrated in Figure 5.2. It was decided to use this matrix rather than the self developed version because it will allow for future comparative studies and benchmarking with other project areas.

Anglo American Plc Risk Matrix	Hazard Effect / Consequence (Where an event has more than than one "Loss Type", choose the "Consequence" with the highest rating)				
Loss Type (Adittional "Loss Types" may exist, identify and rate them)	1 Insignificant	2 Minor	3 Moderate	4 Major	5 Catastrophic
S/H Harm to People (Safety / Health)	First aid case/Exposure to minor health risk	Medical treatment case / Exposure to major health risk	Lost time injury / Reversible impact on health.	Single fatality or loss of quality of life/Irreversible impact on health.	Multiple fatalities / Impact on health ultimately fatal
EI Environmental Impact	Minimal environmental harm-L1 incident	Material environmental harm-L2 incident remediable short term	Serious environmental harm-L2 incident remediable within LOM	Major environmental harm-L2 incident remediable post LOM	Extreme environmental harm-L3 incident irreversable
BI/MD Business Interuption/ Material Damage Other Consequential Losses	No disruption to operation / US$20k to US$100k	Brief disruption to operation / US$100k to US$1.0M	Partial shutdown of operation / US$1.0M to US$10.0M	Partial loss of operation / US$10.0M to US$75.0M	Substantial or total loss of operation / more than US$75.0M
L & R Legal and Regulatory	Low level legal issue	Minor legal issue, non compliance & breaches of the law.	Serious breach, investi-gation/report to authority, prosecution &/or penalty	Major breach of law considerable prosecution and penalties	Considerable penalties & prosecution, multiple law suits & jail terms
R/S/C Impact on Reputation/ Social / Community	Slight impact - public Awareness may exist but no public concern	Limited impact - local public concern	Considerable impact - regional public concern	National impact - national public concern	Interational impact - international public attention

Likelihood	Examples (Consider near hits also)	Risk Rating				
5 (Almost Certain)	The unwanted event has occurred frequently,occurs in order of 1 or more times per year, likely within the year.	11 (M)	16 (H)	20 (H)	23 (Ex)	25 (Ex)
4 (Likely)	The unwanted event has occurred infrequently, occurs less than 1 per yr , likely to reoccur within 5 years.	7 (M)	12 (M)	17 (H)	21 (Ex)	24 (Ex)
3 (Possible)	The unwanted event has happened in the business at some time, or could happen within 10 years.	4 (L)	8 (M)	13 (H)	18 (H)	22 (Ex)
2 (Unlikely)	The unwanted event has happened in the business at some time, or could happen within 20 years.	2 (L)	5 (L)	9 (M)	14 (H)	19 (H)
1 (Rare)	The unwanted event has never been known to occur in the business, or it is highly unlikely within the next 20 years.	1 (L)	3 (L)	6 (M)	10 (M)	15 (H)

Risk Rating	Risk Level	Guidelines for Risk Matrix
21 to 25	(Ex) - Exstreme	Eliminate, avoid, implement specific action plans/procedures to manage & monitor
13 to 20	(H) - High	Proactively manage
5 to 12	(M) - Medium	Actively manage
1 to 5	(L) - Low	Monitor & manage as appropriate

Figure 5.2 – Anglo American plc Risk Matrix

(Adapted from internal document)

This risk matrix works on a five by five matrix, with the horizontal hazard effect or consequence axis rated from insignificant (1) to catastrophic (5) and the vertical likelihood axis rated from rare (1) to almost certain (5). Each rating has very tangible and measurable guideline description to ensure the correct rating is applied on both axis. The hazard effect or consequence is also classified into different loss types such as safety and health, business interruption / material damage & other consequential losses and legal & regulatory. The combined risk rating is given as a number, ranging from 1 at the lowest end of the scale up to 25 at the opposite end. These ratings are subdivided into four levels of risk, namely low, medium, high and extreme. Because of the examples given for each rating the results will be comparable for different risk exercises done by independent persons at various areas or sites.

A hazard identification and risk rating sheet was developed as part of this treatise, to address each HRA in more detail. The sheet lists various hazards under each HRA, however in some instances the same hazard appears in more than one HRA. It also assesses whether the hazard affects the specific site or project and whether it is manageable or not. This sheet was designed so that it can be used at different sites, or at the same site at different times, as a check list to highlight the main risks and determine a risk rating for each. This will help to focus the effort in the correct area. The list can also be adapted, changed and further developed to suite specific operation types. In this project this hazard identification and risk rating sheet was applied to the North West block as illustrated in Figure 5.3. It is important to note that the ratings are for the current situation, and view this block as part of the larger Kleinkopje Colliery operation and not as a new mine by itself. This view is also not in the context of the larger Anglo Coal and Anglo American plc.

HAZARD IDENTIFICATION AND RISK RATING SHEET — Page 1 of 3

Site description: Kleinkopje Colliery, North West block

Name: Bert Schalekamp
Date: September 2006

Headline Risk Area	Hazard	Effected by		Manageable		Loss Type					Hazard Effect	Likelihood	Risk Rating	
		Yes	No	Yes	No	S/H	EI	BI/MD	L&R	R/S/C	1 to 5	1 to 5	Total	Level
Employee Safety														
	Not using personal protective equipment	X		X		X					3	4	17	H
	Work in hazardous mining environment	X		X		X					3	3	13	H
	Transport, vehicle accidents	X		X		X					4	3	18	H
	Handling material incorrectly	X		X		X					3	3	13	H
	Not adhearing to standards and procedures	X		X		X					4	4	21	E
	Not trained to perform work safely	X		X		X					3	3	13	H
	Hazardous substances exposure	X		X		X					2	2	5	L
	Hearing conservation program failure	X		X		X					4	2	14	H
	Safety statistics worsening	X		X		X					3	3	13	H
	OHSAS 18000 non compliance	X		X		X					2	3	8	M
Commodity Price														
	Coal price drops significantly	X			X			X			4	3	18	H
	Reduced product demand	X			X			X			4	2	14	H
	Product supply increase	X			X			X			3	3	13	H
Technology														
	Mining equipment design & development	X		X				X			2	5	16	H
	Beneficiation equipment design & development	X		X				X			2	5	16	H
	Computer equipment, software	X		X				X			1	5	11	M
	Communication technology	X		X				X			1	5	11	M
	Radar technology		X		X			X			1	3	4	L
	Space technology		X		X			X			1	2	2	L
	Thermal technology		X		X			X			1	3	4	L
Infrastructure														
	Buildings	X		X				X			2	3	8	M
	Power lines	X		X				X	X	X	2	4	12	M
	Roads	X		X				X	X	X	1	3	4	L
	Pipe lines	X		X				X			1	3	4	L
	Residential areas	X			X				X	X	2	5	16	H
	Airfields	X		X				X			1	3	4	L
	Railway lines		X		X			X	X	X	1	2	2	L
	Conveyor belts		X	X				X			1	2	2	L
	Servitudes	X		X					X		1	2	2	L
	Nature area		X		X		X				1	2	2	L
	Shafts	X		X				X			2	1	2	L
	Waste dumps		X	X			X				1	2	2	L
Operational Performance														
	Mining extraction	X		X				X			3	4	17	H
	Dilution	X		X				X			3	5	20	H
	Contamination	X		X				X			3	5	20	H
	Fines generation	X		X				X			2	5	16	H
	Spontaneous combustion	X		X				X			4	5	23	E
	Coal losses	X		X				X			3	5	20	H
	Not achieving mine design requirements	X		X				X			4	3	18	H
	Not achieving planned production rate	X		X				X			4	4	21	E
	Not achieving planned production cost	X		X				X			4	3	18	H
	Water accumulations in old workings	X		X				X			4	4	21	E
	Mine planning	X		X				X			2	3	8	M
	Mining conditions	X		X				X			2	3	8	M
	Mining equipment performance	X		X				X			3	3	13	H
	Engineering availability	X		X				X			2	3	8	M
	Engineering utilization	X		X				X			2	3	8	M
	Plant production rate	X		X				X			2	3	8	M
	Plant availability	X		X				X			2	3	8	M
	Plant utilization	X		X				X			2	3	8	M
	Plant production cost	X		X				X			2	2	5	L
	Plant efficiency	X		X				X			2	3	8	M
	Train loading	X		X				X			2	2	5	L
	Coalink performance	X			X			X			4	4	21	E
	RBCT	X			X			X			3	2	9	M
	Shipping	X			X			X			3	2	9	M

Figure 5.3 – Hazard Identification and Risk Rating Sheet 1

(Adapted from internal document)

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UNIVERSITY OF PRETORIA
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HAZARD IDENTIFICATION AND RISK RATING SHEET

Page 2 of 3

Site description:	Kleinkopje Colliery, North West block										Name:	Bert Schalekamp		
											Date:	September 2006		
Headline Risk Area	**Hazard**	**Effected by**		**Manageable**		**Loss Type**					**Hazard Effect**	**Likelihood**	**Risk Rating**	
		Yes	**No**	**Yes**	**No**	**S/H**	**EI**	**BI/MD**	**L&R**	**R/S/C**	**1 to 5**	**1 to 5**	**Total**	**Level**
Counterparty														
	Market competition	X			X			X			2	3	8	M
	Eskom contract competition		X											
	Personnel and skills competition	X			X			X			2	4	12	M
Event Risk														
	Natural disaster	X			X			X			4	1	10	M
	Manmade disaster	X			X			X			3	1	6	M
	Catastrophic event	X			X			X			5	1	15	H
Legal / Regulatory / Finance														
	Current legislation	X		X				X			2	4	12	M
	New legislation	X			X			X			2	4	12	M
	Legal compliance	X		X				X			2	4	12	M
	Legal non compliance	X		X				X			2	2	5	L
	Current regulatory requirements	X		X				X			2	4	12	M
	New regulatory requirements	X			X			X			2	4	12	M
	Regulatory compliance	X		X				X			2	4	12	M
	Regulatory non compliance	X		X				X			2	2	5	L
	Relationship with DME	X		X				X			2	3	8	M
	Operational expenditure	X		X				X			2	3	8	M
	FOB Rand per saleable ton	X		X				X			3	3	13	H
	R/$ exchange rate	X			X			X			4	3	18	H
	Coal price drops significantly	X			X			X			4	3	18	H
	Deviation from budget	X		X				X			3	3	13	H
	Low profit margin	X		X				X			3	3	13	H
	Tax increase	X			X			X			3	2	9	M
	Low EBIT	X		X				X			3	3	13	H
	Low EBITDA	X		X				X			3	3	13	H
Reserves and Resources														
	Borehole density	X		X				X			3	2	9	M
	Sample analyses	X		X				X			3	2	9	M
	Laboratory accuracy	X		X				X			3	2	9	M
	Model confidence	X		X				X			3	2	9	M
	Dolerite dykes & sills	X		X				X			2	3	8	M
	Weathering	X		X				X			2	2	5	L
	Seam undulations	X		X				X			2	3	8	M
	Washouts	X		X				X			1	2	2	L
	Thickness variations	X		X				X			2	3	8	M
	Groundwater, river, dam, vlei	X		X				X			1	2	2	L
	Coal losses	X		X				X			3	3	13	H
	Old workings, inaccurate old plans	X		X				X			2	3	8	M
	Top coaled areas	X		X				X			2	4	12	M
	Water accumulations in old workings	X		X				X			4	4	21	E
	Spontaneous combustion	X		X				X			4	5	23	E
	Structural complications, slips, faults	X		X				X			1	2	2	L
	Dilution	X		X				X			2	4	12	M
	Contamination	X		X				X			2	4	12	M
	Sampling accuracy	X		X				X			2	2	5	L
	Derating	X		X				X			3	4	17	H
	Reconciliation	X		X				X			3	3	13	H
	Grade control	X		X				X			2	3	8	M
	Overall recovery	X		X				X			3	3	13	H
	Sinkholes	X			X	X	X			X	4	3	18	H
	Subsidence	X			X	X	X			X	4	3	18	H
	Slimes in old workings	X			X			X			3	3	13	H
	Pillar stability related to age	X			X	X	X			X	4	3	18	H
	Poor roof conditions		X											

Figure 5.4 – Hazard Identification and Risk Rating Sheet 2

(Adapted from internal document)

HAZARD IDENTIFICATION AND RISK RATING SHEET

Site description: Kleinkopje Colliery, North West block

Name: Bert Schalekamp
Date: September 2006

Headline Risk Area	Hazard	Effected by		Manageable		Loss Type					Hazard Effect	Likelihood	Risk Rating	
		Yes	No	Yes	No	S/H	EI	BI/MD	L&R	R/S/C	1 to 5	1 to 5	Total	Level
Employees														
	Skills shortages	X		X				X			3	4	17	H
	Training needs	X		X				X			3	3	13	H
	Retention of skills problem	X		X				X			3	3	13	H
	Ageing workforce	X			X			X			2	3	8	M
	Remuneration not competitive	X		X				X			2	3	8	M
	Work and accommodation far from main centres	X			X			X			2	3	8	M
	Labour action	X		X				X			2	2	5	L
	Discrimination, racism	X		X				X			2	2	5	L
	Sexual harassment	X		X				X			2	3	8	M
	Women in mining increasing	X		X				X			1	2	2	L
	Cultural diversity	X		X						X	1	2	2	L
	Safety, security, crime	X		X		X		X			2	3	8	M
Political														
	Political instability	X			X			X			1	2	2	L
	Political conflict		X		X			X			1	2	2	L
	Investor confidence	X			X			X			3	2	9	M
	International boycotts	X			X			X			4	2	14	M
	International trade restrictions	X			X			X			3	2	9	M
Mergers and Acquisitions														
	Hostile takeover		X		X			X		X	1	3	4	L
	Joint ventures		X	X				X		X	1	2	2	L
	Merger		X	X				X		X	1	2	2	L
	Acquisitions		X	X				X		X	1	2	2	L
	Asset sell off	X		X				X			1	2	2	L
Social														
	Interested and affected parties	X		X			X		X	X	2	3	8	M
	Neighbour relations	X		X					X	X	2	3	8	M
	Informal settlements encroachment	X		X		X			X	X	1	3	4	L
	Trespassers	X		X		X					3	3	13	H
	Residential area	X			X		X		X	X	2	4	12	M
Capital Projects														
	Capital expenditure return	X		X				X			3	2	9	M
	Life of project reduced	X		X				X			2	3	8	M
	NPV not achieved	X		X				X			3	3	13	H
	IRR % not achieved	X		X				X			3	3	13	H
	Payback period increase	X		X				X			3	3	13	H
Treasury														
	Interest rates		X											
	Liquidity		X											
Environment														
	Environmental non compliance	X		X			X		X	X	2	3	8	M
	Environmental complaints increase	X			X		X		X	X	2	3	8	M
	Relationship with DWAF deteriorates	X		X					X		2	3	8	M
	Relationship with Dept. of Nature Conservation	X		X					X		2	3	8	M
	Relationship with DME deteriorates	X		X					X		2	3	8	M
	Mist and Smog	X			X	X	X	X			2	5	16	H
	Gasses and fumes	X		X			X			X	2	4	12	M
	Dust fallout	X		X			X		X	X	2	5	16	H
	VOC's	X			X		X		X		2	4	12	M
	Noise exceeds limits	X		X			X		X	X	2	3	8	M
	Blasting noise exceeds limits	X		X			X		X	X	2	3	8	M
	Blasting vibrations exceeds limits	X		X			X		X	X	2	3	8	M
	Visual pollution	X		X						X	1	3	4	L
	Wind direction negatively impacts	X			X		X				1	5	11	M
	Rainfall hamper operation	X			X			X			1	5	11	M
	Aesthetics	X		X						X	1	3	4	L
Foreign Exchange														
	FOB US$ per saleable ton cost increase	X		X				X			3	3	13	H
	FOB Rand per saleable ton cost increase	X		X				X			3	3	13	H
	R/$ exchange rate deteriorates	X			X			X			4	3	18	H
	Coal price drop significantly	X			X			X			4	2	14	H
Employee Health														
	HIV & Aids impact	X			X	X		X			4	4	21	E
	Anti retroviral treatment not affective	X			X	X		X			3	2	9	M
	Medical aid deteriorates	X			X	X		X			2	2	5	L
	VCT program effectiveness	X		X		X		X		X	3	2	9	M

Comments:

Figure 5.5 – Hazard Identification and Risk Rating Sheet 3

(Adapted from internal document)

This risk rating was the result of an assessment by a single person as opposed to that of a project team, which will be able to give a more balanced and less biased view. The results of this risk ranking exercise on the North West block are discussed below in more detail, focusing on some of the extreme and high risk ratings that were identified. A total of 6 extreme level risks were identified and 11 high level risks (above 18 rating) and are listed below.

Extreme level risks

- Not adhering to standards and procedures, (21 E)
- Spontaneous combustion, (23 E)
- Not achieving planned production rates, (21 E)
- Water accumulations in old workings, (21 E)
- Coalink performance, (21 E)
- HIV & Aids impact, (21 E)

High level risks

- Transport and vehicle accidents, (18 H)
- Coal price drops significantly, (18 H)
- Dilution, (20 H)
- Contamination, (20 H)
- Coal losses, (20 H)
- Not achieving mine design requirements, (18 H)
- Not achieving planned production cost, (18 H)
- R/$ Exchange rate, (18 H)
- Sinkholes, (18 H)
- Subsidence, (18 H)
- Pillar stability related to age, (18 H)

5.3. DISCUSSION OF RISK RATING FINDINGS

5.3.1. COAL PRICE DROPS SIGNIFICANTLY

The coal price is currently high and this has benefited the coal producers. However the R/US$ exchange rate has off-set this somewhat. This is not a manageable risk and is related to many factors, mostly international currency trends and global supply and demand for coal. This is a difficult risk to manage as the major producer role players, such as China and Australia, and consumers such as Europe and Asia may change the trend over a very short period, due to the volumes they control.

The effect or consequence of a declining coal price on this type of operation is major (4) because of the lower profit margin. It can result in a loss of revenue of US$ 10.0 million to US$ 75.0 million. It could happen and it has happened in the past 10 years as illustrated in Figure 4.4. Therefore the likelihood of a significant drop in the coal price is possible (3). Kleinkopje Colliery and the North West block will be a low quartile producer due to the difficult mining conditions, derated reserves and subsequent lower yield. The overall risk rating is 18 and the level is high. This risk can be ameliorated by entering into longer term of-take agreements with consumers.

5.3.2. RAND (ZAR) DOLLAR (US$) EXCHANGE RATE

The volatility of the ZAR versus the US$ exchange rate is one of the bigger risks because of the high unpredictability and because it is not controllable. In January 2002 the exchange rate was R11,63 to the US$, after which it dropped down to R8,69 by January 2003 and R6,94 in January 2004. It reached a low of R5,72 by December 2005 after which it started rising slowly and remaining around the low R6,00 level.

Since May / June 2006 it has started climbing again. The industry is forecasting at approximately R7,00 into the future.

The effect or consequence of a strengthening Rand on this type of operation will be major (4), because of the almost total dependency on the coal export market. The likelihood of this happening is possible (3). This has happened before and could happen within the next 10 years. The overall risk rating is 18 and the level is high.

5.3.3. SPONTANEOUS COMBUSTION IMPACTS

Spontaneous combustion poses a risk to the production process, coal quality, coal losses, beneficiation process and overall saleable production. However it is a risk that can be managed. Several options are available, such as:

- Buffer blasting
- Cladding
- Water cannons
- Mining at a faster advance rate
- General awareness and focus

These additional activities result in an increase in mining cost which is not always measurable, but by not reducing the risk it would certainly compromise the viability of the operation. Without controls, the effect or consequence would be catastrophic (5) and the likelihood would both be almost certain (5), resulting in an extreme level risk. With the current controls in place the effect will still be major (4) and the likelihood would be almost certain (5), resulting in a risk rating of 23 and an extreme risk level. It is important to note that spontaneous combustion has already started and is ongoing.

There is scope to reduce the likelihood in the future down to a possible (3) risk rating, through continuous improvement initiatives. It is very unlikely to reduce it further than this because it is an inherent property of this coal.

5.3.4. WATER ACCUMULATIONS IN OLD ORKINGS

Rain water run-off and water recharge from the spoils have had serious impacts on coal production and reserve losses in the past and present. This is manageable with a well designed and implemented pit water reticulation system. The main concern is the storage and disposal of poor quality water, since it is not suitable for direct discharge into the river system. The current situation is critical due to the overall poor water quality in the natural rivers and dams. A controlled release permit from the Department of Water Affairs (DWAF) is very unlikely, and will only be considered once the river and dam qualities have improved. Without controls in place the effect or consequence will be catastrophic (5) and the likelihood almost certain (5) resulting in a risk rating of 25 and extreme risk level leading to eventual mine closure. The current effect is still major (4) and likely (4). This still leaves the overall risk rating at 21 and the risk level extreme. The main reason is the lack of storage capacity for the large volumes of poor quality water. The situation has been unchanged for the past five years and this has necessitated the construction of a water treatment plant that will become operational in 2007, it was designed to treat the SACE water to meet potable quality standards.

5.3.5. DILUTION

Dilution is planned contamination and this is based on reconciliation data and experience gained over time. The effect or consequence of inaccurate dilution numbers is moderate (3) because it results in incorrect forecasting and planning culminating in a financial loss. The likelihood of this happening in the North West block is almost certain (5) because of the different and unknown conditions. The overall risk rating is 18 and the level is high. A solution to this risk will only be possible after the first year of mining and reconciliation.

The other two high level operational performance risks identified, namely Contamination, (20 H) and Coal losses, (20 H) will also need to be managed in a similar manner.

5.3.6. NOT ACHIEVING MINE DESIGN REQUIREMENTS

This mine is somewhat unique in its design due to the four separate pits and the mining of multiple seams and old workings. The pit or cut lengths are short by normal standards and the mining of, up to three seams per pit makes this a very complex and highly constrained operation. The depth increases progressively with every subsequent cut, which requires a certain amount of continuous improvement in order to sustain the same level of production. The maximum depth constraint is limited to 70m and this will require a large portion of material to be moved by pre-strip truck and shovel operations to assist the draglines. The future reserve blocks include the case study area which is even more challenging than the current mining area. There is a definite risk to achieving the mine design expectations. The risk is mainly because this mining method with all the mine specific complexities, as mentioned above and throughout this treatise, has not been proven yet.

The effect or consequence of not meeting the mine design expectations will be major (4), because that will make the viability of the operation questionable. The likelihood is possible (3), because it has not been done and proved yet. This results in an overall risk rating of 18 and a high risk level. However a lot of technical expertise and practical experience has been accumulated at Kleinkopje Colliery, mining similar conditions and using this mine design.

5.3.7. SINKHOLES

Sinkholes in old workings are a real hazard especially in the older areas such as the North West block. It is difficult to control access of people because they tend to steal or destroy the fences and trespass into these areas. The effect or consequence is major (4) from a safety and health loss type because it could result in a fatality. The likelihood is possible (3) which gives an overall risk rating of 18 and the risk level is high. This has happened before and could happen again in the next 10 years. The solution would be to mine these areas as we are currently doing and rehabilitate the area for grazing purposes.

Rock engineering always plays a role in any mining operation in a greater or lesser degree. In this operation the increase in depth with time will increase the importance of adherence to proper rock engineering standards. The main aspects are short term highwall and low wall stability due to the continuous cut and fill sequence of opencast mining. The effect and likelihood is low, because of the management systems in place to ameliorate the risk, such as employee training, a code of practice, mining standards and awareness. Longer term concerns will be the stability of old workings dating to pre-Salomon time (1967), which were mined to very shallow depths and low safety factors. This was highlighted by both the Subsidence, (18 H) and Pillar stability related to age, (18 H) risks.

5.3.8. COALINK PERFORMANCE

This risk is related to business interruptions and will impact on marketing, customer service and contract fulfillment. If Spoornet fails to improve on the current service levels, this risk definitely has the potential to escalate to a national disaster for the South African coal export industry. The impact is already limiting and has reduced the annual railings and export sales forecast figures. This could have a negative impact on the financial viability of export coal mines.

The effect and consequence will be major (4), due to the loss of revenue because of reduced sales, loss of credibility because of inability to meet export commitments to customers and additional cost incurred to stockpile product on the mine and possible demurrage at RBCT. Currently the likelihood of this happening is likely (4). The result is a risk rating of 21 which is an extreme level of risk. The latest forecast and speculation by economists is that the RBCT will miss its 2006 target of 76 million tons by approximately 8 million tons (Brown, 2006). This is a reduction of approximately 10.5% in export tons, which will have a significant impact on the export coal mines.

5.3.9. NOT ACHIEVING PLANNED PRODUCTION RATE

The risk is related to the new or unproved nature of the North West block. The effect or consequence will be major (4) due to the financial loss of underproduction and capital burden. The likelihood based on the last five years performance history is likely (4), because of the fact that Kleinkopje Colliery has been unable to achieve forecasted production rates. The total risk rating is 21 which is an extreme risk level. However the current outlook is positive and recent production successes have restored some confidence.

5.3.10. HIV AND AIDS IMPACT

The effect or consequence of HIV and Aids on the health of employees is rated as major (4) because it can result in death and irreversible impact on health. The likelihood is rated as likely (4) because has occurred infrequently. The overall risk rating is 21 which is an extreme level of risk. There are several plans and initiatives being used to manage and monitor this risk. This is not manageable by the company because it is a national crisis and needs intervention by government.

5.3.11. OTHER RISKS

Mining standards and the detrimental impact of the failure of this has been discussed throughout this treatise. Issues such as percentage contamination, grade control, fines generation and percentage mining extraction fall under this heading. I will therefore not focus on the detail, but would rather re-emphasize the negative impact this may have on the overall financial viability of the operation.

Environmental legislation is continuously evolving and becoming stricter every year. This is necessary to protect the environment and ensure responsible and accountable mining operations. With ISO 14001 compliance, environmental issues have become an integral part of mines. Compliance to these standards is non-negotiable and need to be managed effectively not withstanding cost. The impact will be high, because of the mines close proximity to Witbank, the mining method which makes it is very visible, and the mines location next to residential areas is a sensitive issue. Other contributing factors are, the large volumes of poor quality water and the smoke and dust originating from the spontaneous combustion. There is a continuous focus on and adherence to legislation, through the current management systems.

Due to the close proximity of Kleinkopje Colliery, North West block to the city of Witbank residential area, it is important to monitor possible encroachment of informal settlements next to the mine or even on to mine property. An estates manager, and the SACE security department monitor this on a regular basis. The reserve base is a mines biggest asset and can not be compromised. The North West block mining limit has been placed at a minimum distance of 500m from the residential area. This was done to comply with mining regulation.

6. FINANCIAL MODEL

6.1. INTRODUCTION

A financial model as a decision making, strategic tool and a method for continuous optimal management, in the context of a mining operation should take all factors into account that play a role in this type of operation. It should realistically simulate all inputs and outputs and take cognisance of the individual and combined impacts on the viability and profitability of this venture.

The aim of this treatise is to evaluate the financial viability of previously mined areas within the Witbank Coalfield, more specifically Kleinkopje Colliery using the North West pit as a case study, compared to current mining operations.

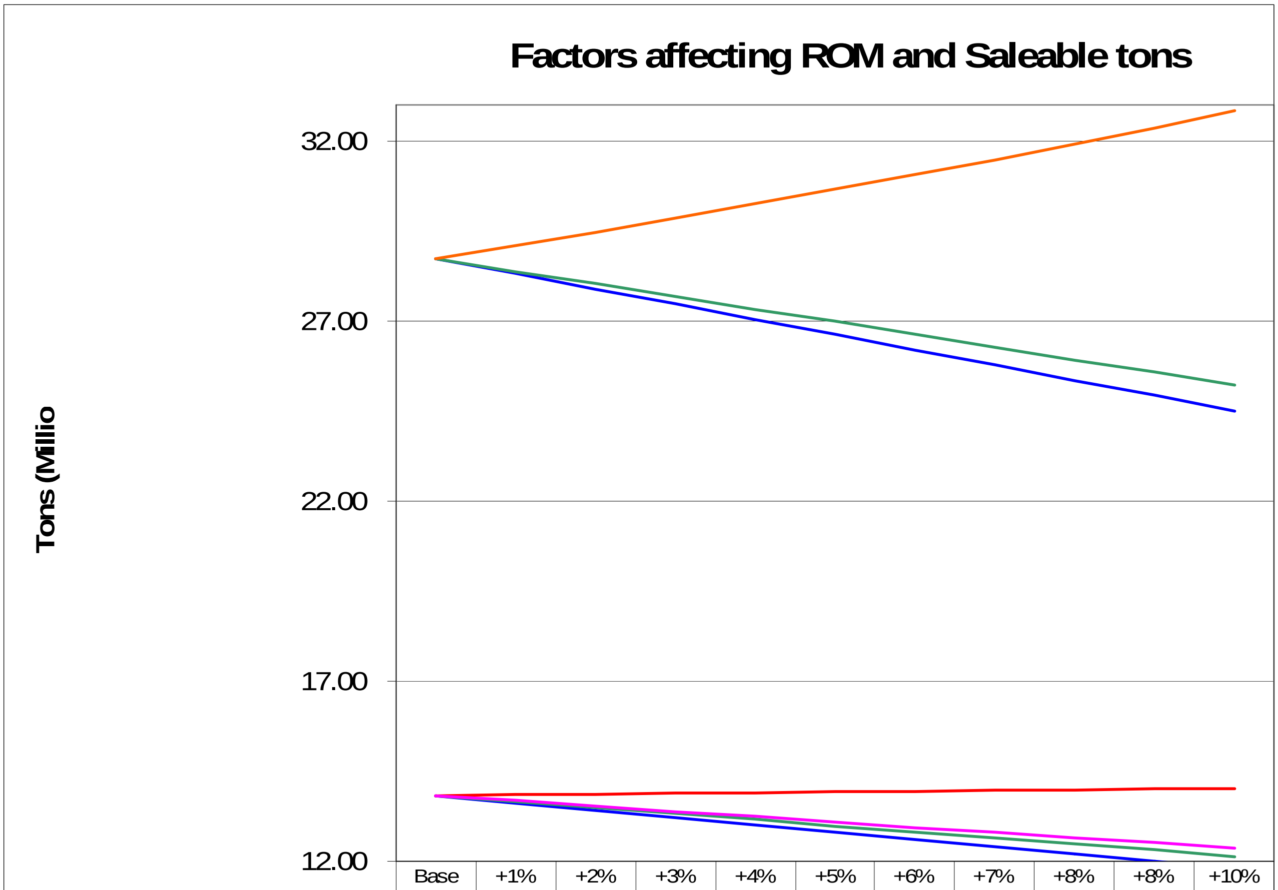
Bias and subjective evaluation or feasibility studies can result in certain projects not measuring up to the plan and others slipping through the scrutiny process. What is required is a system or process that is objective and comprehensive in evaluating previously mined resources, and can be applied to various projects to make the results comparable.

It was important to familiarize the reader firstly with the actual mining process and all the factors that may negatively affect this case study area. A matrix was constructed to rank the various risks in terms of their potential impact. This matrix was then used to evaluate the North West block as a case study, but the objective is to be able to apply the same matrix to different areas and to create an unbiased comparison.

6.2. THE MATRIX

The idea of the matrix is to give us a tool to evaluate the impact of mostly mining related factors on the viability of a resource and reserve block. An annual five year forecast process is done for the total mine reserve which is based on the latest geological model, mine plan and beneficiation plant factors. However, this process lacks the flexibility to do small and large scale “what if” scenarios. It is more a “baseline forecasting exercise”. It is suggested that application of the risk matrix will provide a method of iterative exercises to determine sensitivities of the various factors. These results may then be used in subsequent forecasts on a bigger scale. See appendix A.

In the following paragraphs the impact of derating, mining extraction, dilution and contamination, fines generation, and spontaneous combustion, on the Run Of Mine tons and Saleable tons will be discussed referring to the calculated tons provided in Figure 6.1. As the impact, related to the above mentioned risks increase, the ROM tons available would decrease. Should the production rate and other factors stay the same this would result in a shorter Life of Mine (LOM). This example only illustrates the negative effect of the various impacts’, there is also a positive side, which will require an improvement on the base case number.



	Base	+1%	+2%	+3%	+4%	+5%	+6%	+7%	+8%	+8%	+10%
ROM tons - Derating increase	28.73	28.31	27.88	27.46	27.04	26.62	26.19	25.77	25.35	24.93	24.50
ROM tons - Extraction decrease	28.73	28.38	28.03	27.68	27.33	26.98	26.63	26.28	25.93	25.58	25.23
ROM tons - Contamination increase	28.73	29.09	29.47	29.85	30.24	30.64	31.06	31.48	31.92	32.37	32.83
Saleable tons - Derating increase	13.82	13.62	13.42	13.21	13.01	12.81	12.60	12.40	12.20	11.99	11.79
Saleable tons - Extraction decrease	13.82	13.66	13.49	13.32	13.15	12.98	12.81	12.64	12.48	12.31	12.14
Saleable tons - Contamination increase	13.82	13.84	13.86	13.88	13.90	13.92	13.94	13.96	13.98	14.01	14.03
Saleable tons - Fines increase	13.82	13.68	13.53	13.38	13.23	13.09	12.94	12.80	12.66	12.52	12.38

%Increase

Figure 6.1 - Factors Affecting ROM and Saleable Tons

6.2.1. DERATING

<u>ROM-Tons</u>

The ROM tons decrease with an increase in the derating percentage. The impact on the ROM tons is demonstrated by increases in the percentage derating starting from a base of 32% and increasing from 1% to 10% in Figure 6.1. This assumes that the mining extraction and contamination values are kept constant at the base case number of 82% and 20% respectively. The reason for the decrease in ROM tons is because there is less coal left to be recovered and the void is larger than initially anticipated.

SALEABLE-Tons

The impact of derating on the saleable tons is demonstrated in Figure 6.1 by increasing the percentage derating starting from a base of 32% and then increasing by 1% to 10%. Because there is less coal remaining to be recovered it will follow that once this reduced coal ROM tonnage is beneficiated the saleable tons will also be less than the initial base case estimates. The saleable tons are directly related to the ROM tons assuming the yield is constant.

6.2.2. MINING EXTRACTION

ROM-Tons

This is very similar to the derating and also shows a steep drop in the ROM tons recovered from a specific block with reduced percentage extraction in that block. It means that less coal tons are extracted from the planned base value. This will result in a shorter LOM for the block. This is mainly affected by mine design and mining standards. In this case the percentage derating and contamination were kept constant at the base case value of 32% and 20% respectively.

SALEABLE-Tons

A similar trend is obtained to that of the derating. The less coal extracted, the less saleable can be produced, at the same yield percentage.

6.2.3. DILUTION AND CONTAMINATION

ROM-Tons

The percentage contamination, was increased at 1% increments from 20% to 30%, which meant that for the same amount of coal moved, there is more rock or contaminant, which increases the overall ROM tons. The derating and mining extraction were kept constant at the base case values of 32% and 82% respectively. Alternatively, the ROM tons may be kept constant, that will mean that with an increase in contamination the amount of coal that can be extracted will be reduced.

<u>SALEABLE-Tons</u>

As the percentage contamination is increased, the saleable tons will decrease, if the ROM tons are kept constant.

6.2.4. FINES GENERATION

<u>ROM-Tons</u>

An increase of the percentage fines will not affect the ROM tons. However, the change in size distribution of the ROM tons will have an impact on the coal hauling expressed in the truck factor. The finer the ROM feed the higher the truck factor, or tons per load.

<u>SALEABLE-Tons</u>

The percentage fines is increased from 17% at 0.5% increments up to 22%. This reduces the saleable tons, due to the loss of fines to the spiral and slimes circuits. The recovery of fine coal product -0,5mm is less effective than the coarser fractions +0,5mm.

6.2.5. SPONTANEOUS COMBUSTION

<u>ROM-Tons</u>

An increase in the area affected by spontaneous combustion in the pit will have a significant negative impact on ROM production. The tonnages and coal quality will decrease.

SALEABLE-Tons

Less saleable tons will be produced at a lower quality as larger areas in the pit are affected by spontaneous combustion. The negative affect will be visible in the calorific values and percentage volatiles.

6.3. FINANCIAL IMPLICATIONS

"It comes at a cost"

This evaluation is based on the continuation of current mining operations with the North West block later (2014) starting up as the replacement for Block 5W which will have been depleted by then. The main factors driving this evaluation is the cost of producing one ton of coal, taking into account the lower ROM and saleable tons due to difficult circumstances as described above.

The assumptions used in the financial model are based on the Matrix in Appendix A, and is consolidated in Table 6.1 below.

All factors impacting on the saleable tons have been split into three scenarios, with an increasing negative impact on saleable tons produced. The ROM, saleable tons and yield are taken from the current five year forecast document for the North West block, and are adjusted to expected results after these three scenarios have been applied. These scenarios are done using the matrix to determine the impacts on saleable tons and yields.

Table 6.1 - The impact of identified risk factors on ROM tons and Saleable tons, calculated for three scenarios.

	Current Mining Practice	Adjustments anticipated for case area		
		Low	Medium	High
Annual ROM tons opencast	2,800,000	2,800,000	2,800,000	2,800,000
Derating %				
- Historical derating @ 68%				
- Anticipated derating @ 63%		-3%	-5%	-7%
Mining Extraction %				
- Historically extraction @ 82%				
- Anticipated extraction @ 79%		-1%	-3%	-5%
ROM tons for case study area	2,800,000	2,688,000	2,576,000	2,464,000
Historical yield	50.33%	50.33%	50.33%	50.33%
Annual Saleable tons	1,409,240	1,352,870	1,296,501	1,240,131
Contamination %				
- Historical contamination @ 20%				
- Anticipated contamination @ 30%		-5%	-10%	-15%
Fines %				
- Historical fines @ 17%				
- Anticipated fines @ 20%		-1%	-3%	-5%
Saleable tons for case study area	1,409,240	1,271,698	1,127,956	992,105

6.3.1. FACTORS AFFECTING DERATING

The current derating and the derating percentage used in the base case scenario is 32%, which is very close to the 30% quoted for Block 2A. However, the area top coaled in Block 2A is approximately 35%, while it is in the order of 65% in the North West block. The order of magnitude of the difference is very large. Additional concerns are the age of the workings, survey plan accuracies and lack of mining height measurements to do proper volumetric derating (Refer to 3.6.1).

As explained, the main factors affected by errors in the derating are:

a) ROM tons and LOM estimates

b) Yield prediction variances due to less coal and more contamination

c) Inaccurate mining horizon combinations leading to yield discrepancies

This aspect of reserve estimation requires urgent attention because of the magnitude of the impact it can have.

The percentage extraction is very sensitive to an increase in mining height. If the remnant pillars are small relative to the excavation, this will worsen the situation (see Table 6.2 below).

Table 6.2 - Relation between Percentage Volumetric Extraction and Pillar Size

Mining Height		**% VOLUMETRIC EXTRACTION**					
		20m	**18m**	**16m**	**14m**	**12m**	Pillar centres decrease
Mining height of old workings	3m	26,3	27,8	29,3	30,7	31,3	
	4m	35,0	27,0	39,1	40,8	41,7	% Primary
	5m	43,8	46,3	48,8	51,0	52,1	extraction increase
	6m	52,5	55,6	58,6	61,3	62,5	

The case study area is well known for its small pillars and high prevalence of top coaling. Current mining areas have primary extraction percentages of 25 to 35 percent which is viable, however in the top coaled areas this increases to more than 40%.

6.3.2. FACTORS AFFECTING MINING EXTRACTION

Mining extraction is the percentage of the coal theoretically planned to be extracted by mining based on the physical limits of the individual cuts. This is reconciled annually in order to determine a realistic number to plan into the future. Currently 82% extraction is assumed for the North West block, but this is unproven, as mining has not yet commenced in the area.

The main factors that may negatively influence this assumption are:

a) Poor drilling and blasting

b) Ineffective O/B removal by the dragline

c) Water in pit (Refer to 3.6.2.2)

d) Poor coaling operations

e) Spontaneous combustion (Refer to 3.6.5)

f) Top coaling

For the neighbouring block, 2A North, a value of 82% mining extraction is used, but this area has significantly less top coaled areas. The probability is therefore high that a lower extraction percentage should be expected in the North West block.

6.3.3. FACTORS AFFECTING DILUTION AND CONTAMINATION

Contamination is an unavoidable feature of this type of mining due to the collapse of the overburden and interburden into existing bords. It requires constant focus and plays a critical part in the financial viability. An increase in contamination is directly related to a decrease in yield, and vice versa.

The negative factors associated with increased levels of contamination are:

a) Reduced yield

b) Increase in tonnage to haul and treat in the beneficiation plant for the same product tons

c) Reduction in plant efficiencies

d) Increased discard tonnages

e) Larger dump facilities required

f) End of life rehabilitation cost increase

Currently the mine uses a contamination figure of 20% for old working areas such as Block 2A. This is too optimistic for the North West area since there is much more top coaled areas and smaller pillars. Due to these factors there will definitely be higher levels of contamination. Based on previous experience in other top coaled areas and using the high primary percentage extraction as an indicator. It is suggested that this should be in the order of 30%. A future continuous improvement project would be to develop a method of separating or eliminating a portion of the contamination from the ROM tons before loading and hauling to the plant.

6.3.4. FACTORS AFFECTING FINES GENERATION

The percentage fines increase, in areas of higher contamination and higher levels of spontaneous combustion. The North West area will have both due to the higher primary extraction percentage. An additional concern is the increased haul distance from the plant which may necessitate an intermediate stockpile causing re-handling. Alternatively an intermediate tip and conveyor system to the Block 2A area can be used. This will require capital and the additional handling and transferring of coal will increase the percentage fines.

All these factors combined can easily increase the overall fines percentage from 17% to approximately 20%, and higher.

6.3.5. FACTORS AFFECTING SPONTANEOUS COMBUSTION

Spontaneous combustion will remain an issue for the rest of the Kleinkopje Mine’s life. However the impact can be managed by implementing good mining practice, discipline and mine wide awareness. Factors to consider will be the effectiveness of the drilling and blasting of the buffer, batter angles of the highwall and speedy cladding.

6.4. CAPITAL EXPENDITURE

The capital required for this project is set out in Table 6.3 below. This was determined as part of the original Kleinkopje expansion project.

Table 6.3 – Capital Expenditure

CAPITAL ESTIMATE					R000's
Establish initial boxcut	**Rate**	**Quantity**	**Measure**		12,332
- Boxcut establishment	0	26,250	m3	778	
- Topsoil stripping	0	170,000	m3	1,828	
- Ramp construction	3,242	3		9,727	
Overburden stripping equipement					19,373
- Overburden drill, BE 49R	19,319	1		19,319	
- Overburden drill, transport	54	1		54	
Coal hauling to tip and roads					13,743
- Coal hauler, Euclid CH 130	7,172	1		7,172	
- Grader, Komatsu GD 825A	2,910	1		2,910	
- Watertanker, Komatsu HD 465	3,615	1		3,615	
- Coal hauler, transport	-	1		-	
- Grader, transport	24	1		24	
- Watertanker, transport	24	1		24	
Pre-stripping equipement					127,310
- Rear dump truck, Euclid R170	31,683	3		95,048	
- Pre-strip dozer, Komatsu D375	3,288	1		3,288	
- Shovel, O&K RH 200	28,950	1		28,950	
- Rear dump truck, transport				-	
- Pre-strip dozer, transport	24	1		24	
- Shovel, transport				-	
Provincial road underpass					8,815
- Haul road under provincial road	8,815	1		8,815	
Prepare dragline walkroute					5,192
- Grading 30m wide track	0	3,000		242	
- Construction of culverts, levelling	54	1		54	
- Track contruction over waterlines	54	1		54	
- Reinstate fences along walkroute	11	1		11	
- Provincial road crossing	65	1		65	
- Rehabilitation of walkroute	38	1		38	
- Electrical 400 kV high crossing	4,730	1		4,730	
In-pit roads					5,725
- Construct haul road 26m wide	1	3,500	m	4,327	
- Haul road pollution control	538	1		538	
- Topsoil stripping	0	52,500	m3	565	
- Highwall, Lowwall access	296	1		296	
Drainage in mining area					263
- In-pit cut-off trenches and drains	0	7,000	m	241	
- Topsoil stripping	0	2,100	m3	23	
Pollution control in mining area					2,311
- Pipeline 250mm, steel	0	5,500	m	1,774	
- Pollution control dam	538	1		538	
Pit fencing					255
- Fencing	0	19,000	m	255	
Power supply to the opencast					2,202
- 22kV Circuit breaker at tip	323	1		323	
- 22 kV Overhead powerline from tip to pit	0	4,000	m	774	
- Spurr lines, 3 off	0	4,500	m	871	
- Spurr isolators	17	3		52	
- In line isolators	65	1		65	
- 22kV cabling	65	1		65	
- Instrumentation	54	1		54	
Cable cost					344
- Shovel electrics	172	1		172	
- Drill electrics	172	1		172	
Outside consultants					2,043
- QS services	1,506	1		1,506	
- Road consultant	538	1		538	
Technical investigations					4,461
- EMPR	3,225	1		3,225	
- Geology	1,075	1		1,075	
- Feasibility study, project services	161	1		161	
Purchase of land and servitudes					1,223
- Exhumation of graves	108	1		108	
- Purchase land	6	173	ha	1,116	
Additional spares for new equipement					11,554
- Mining machines	11,262	1		11,262	
- Electrics	292	1		292	
Mine related implementation cost					5,161
- Working costs	5,000	1		5,000	
- Implementation	161	1		161	
					222,309
Engineering Fees (4%)					8,892
Reimbursables (8%)					17,785
Contingencies (15%)					33,346
Total (base)					282,332
Escalation					16,369
Total (escalated)					298,701

The phasing of the above capital expenditure is set out in Table 6.4 below. The phasing of the capital was done taking into consideration the natural progression of this type of project. A new mine or additional pit as in this case, will start up with the technical investigations, ground works, infrastructure establishment, procurement of production equipment and then the production build-up.

Table 6.4 – Phasing of Capital Expenditure

Quarters	R'000	Annual
Q1 2014	R 40,988	
Q2 2014	R 46,859	
Q3 2014	R 51,201	
Q4 2014	R 52,373	R 153,136
Q1 2015	R 38,871	
Q2 2015	R 5,223	
Q3 2015	R 5,779	
Q4 2015	R 15,243	R 52,092
Q1 2016	R 18,329	
Q2 2016	R 18,399	R 29,383
Total	**R 293,264**	**R 234,611**

The North West pit can be established over a 30 month period at a total capital outlay of approximately R290m. In the Kleinkopje life of mine plan it is envisaged that this pit will be the replacement for the 5 West pit which would run out of reserves towards 2014.

6.5. FINANCIAL MODEL

Due to the dynamic nature of the mining environment with specific reference to the change in commodity price, worldwide growth in demand, equipment technology availability and price it is not realistic to do a financial evaluation for 2014 and beyond. This Financial model was therefore run using the current price (at July 2005, for Kleinkopje thermal steam coal, FOB $, Richards Bay Coal Terminal) of $43 per ton and an exchange rate of R6.00 to the dollar. The discounted cashflow model is presented in Table 6.5.

Table 6.5 – Financial Model, North West Block

Profit & Loss a/c in R000's	Year 0	2004	2005	2006	2007	2008	2009	2010	2011	2012	2013	2014	2015	TOTAL
ROM tons underground		-	-	-	-	-	-	-	-	-	-	-	-	-
ROM tons opencast		2,800	2,800	2,800	2,800	2,800	2,800	2,800	2,800	2,800	2,800	2,800	2,800	33,597
Sales Tons														
Medium Grade		-	-	-	-	-	-	-	-	-	-	-	-	-
Low Grade		1,409	1,409	1,409	1,409	1,409	1,409	1,409	1,409	1,409	1,409	1,409	1,409	16,906
Total sales tons		1,409	1,409	1,409	1,409	1,409	1,409	1,409	1,409	1,409	1,409	1,409	1,409	16,906
FOB $ Turnover		60,581	61,990	63,399	64,808	66,217	67,626	68,843	70,082	71,344	72,628	73,935	75,266	816,719
FOB rand turnover		363,488	390,538	412,094	434,213	456,896	480,143	497,583	515,657	534,388	553,799	573,916	594,762	5,807,479
Commission		(4,593)	(3,212)	(3,269)	(3,212)	(3,409)	(3,580)	(3,759)	(3,947)	(4,144)	(4,351)	(4,569)	(4,797)	(46,843)
Other selling expenses		(74,529)	(77,798)	(81,292)	(83,800)	(89,773)	(94,262)	(98,975)	(103,924)	(109,120)	(114,576)	(120,305)	(126,320)	(1,174,672)
FOR Turnover		284,366	309,529	327,534	347,202	363,714	382,301	394,849	407,787	421,124	434,872	449,042	463,645	4,585,964
Working costs - underground		-	-	-	-	-	-	-	-	-	-	-	-	-
Working costs - opencast		(209,981)	(224,890)	(237,934)	(250,306)	(262,821)	(275,962)	(289,761)	(304,249)	(319,461)	(335,434)	(352,206)	(369,816)	(3,432,821)
EBITDA		74,385	84,639	89,600	96,896	100,892	106,338	105,089	103,538	101,663	99,438	96,836	93,829	1,153,144
Depreciation		(4,558)	(19,739)	(23,207)	(23,207)	(23,207)	(23,207)	(23,207)	(23,207)	(23,207)	(23,207)	(23,207)	(23,207)	(256,363)
EBIT		69,827	64,900	66,394	73,689	77,686	83,132	81,882	80,332	78,456	76,231	73,629	70,622	896,781
Current taxation		-	-	-	(12,823)	(30,268)	(31,902)	(31,527)						(106,519)
STC		(7,759)	(7,211)	(7,377)	(6,763)	(5,269)	(5,692)	(5,595)	(5,474)	(5,329)	(5,156)	(4,953)	(4,719)	(71,297)
Net earnings		62,069	57,689	59,017	54,103	42,149	45,538	44,760	74,857	73,128	71,076	68,676	65,903	718,965
FOB $ price		*43.00*	*44.00*	*45.00*	*46.00*	*47.00*	*48.00*	*48.86*	*49.74*	*50.64*	*51.55*	*52.48*	*53.42*	*48.31*
Rand/$ exchange rate		*6.0000*	*6.3000*	*6.5000*	*6.7000*	*6.9000*	*7.1000*	*7.2278*	*7.3579*	*7.4903*	*7.6252*	*7.7624*	*7.9021*	*7.1107*
Turnover per sales ton		*258.00*	*277.20*	*292.50*	*308.20*	*324.30*	*340.80*	*353.18*	*366.01*	*379.30*	*393.08*	*407.36*	*422.16*	*343.51*
Selling expenses per ton		*(56.16)*	*(57.50)*	*(60.02)*	*(61.76)*	*(66.14)*	*(69.45)*	*(72.92)*	*(76.57)*	*(80.39)*	*(84.41)*	*(88.63)*	*(93.07)*	*(72.25)*
Cash costs per sales ton		*(149.04)*	*(159.62)*	*(168.88)*	*(177.66)*	*(186.55)*	*(195.88)*	*(205.67)*	*(215.95)*	*(226.75)*	*(238.09)*	*(249.99)*	*(262.49)*	*(203.05)*
Non cash costs per sales ton		*(3.23)*	*(14.01)*	*(16.47)*	*(16.47)*	*(16.47)*	*(16.47)*	*(16.47)*	*(16.47)*	*(16.47)*	*(16.47)*	*(16.47)*	*(16.47)*	*(15.16)*
Profit per ton		*49.56*	*46.07*	*47.13*	*52.30*	*55.14*	*59.01*	*58.12*	*57.02*	*55.69*	*54.11*	*52.26*	*50.13*	*53.04*
CASHFLOW IN R000's														
Profit before tax and depreciation		74,385	84,639	89,600	96,896	100,892	106,338	105,089	103,538	101,663	99,438	96,836	93,829	1,153,144
STC paid		(7,759)	(7,211)	(7,377)	(6,763)	(5,269)	(5,692)	(5,595)	(5,474)	(5,329)	(5,156)	(4,953)	(4,719)	(71,297)
Capital expenditure escalated	-	(191,420)	(69,739)	(41,618)	-	-	-	-	-	-	-	-	-	(302,776)
Taxation paid		-	-	-	(12,823)	(30,268)	(31,902)	(31,527)	(31,062)	(30,499)	(29,831)	(29,051)	(28,149)	(255,110)
Cash generated/(utilised)	-	(124,794)	7,689	40,606	77,310	65,356	68,745	67,967	67,002	65,836	64,451	62,832	60,961	523,961
RSA inflation index (CPI)	*1.0000*	*1.0000*	*1.0710*	*1.1331*	*1.1920*	*1.2516*	*1.3142*	*1.3799*	*1.4489*	*1.5214*	*1.5974*	*1.6773*	*1.7612*	
Present value of cash flows	-	(124,794)	7,179	35,835	64,855	52,216	52,308	49,254	46,243	43,274	40,346	37,460	34,614	338,790
Cumulative cash flows	-	(124,794)	(117,615)	(81,779)	(16,924)	35,292	87,600	136,854	183,097	226,370	266,716	304,176	338,790	
NPV Discounted values at:														
10.0%	**130,567**													
12.0%	**106,359**													
15.0%	**76,559**													
IRR %	**29%**													
Payback	**4.3 years**													

At the current coal price and exchange rate, a NPV (Net Present Value) of R130,6m is determined with an IRR of 29% and a payback period of 4.3 years. The financial results are set out in the graph below. At current levels, with a payback period of 4.3 years the project does seem viable. The expected life of mine (LOM) of the project is 12 years. The financial results are shown in Figure 6.2.

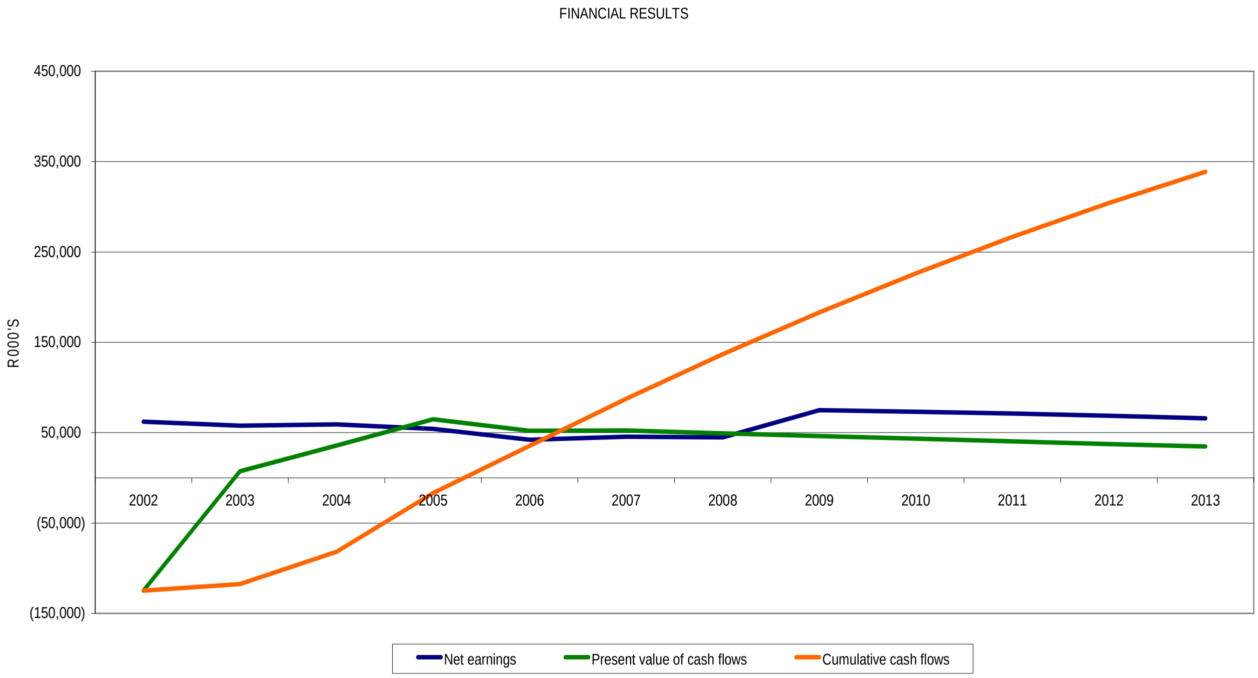


Figure 6.2 – Financial Results graph

6.6. SENSITIVITIES

With the volatility of the ZAR/US$ exchange rate illustrated in Figure 4.5 and coal price in US$ illustrated in Figure 4.4, significant changes can occur in a relatively short time frame to the Rand value per ton of coal. The following sensitivities were performed to establish the viability of this project under those conditions. The results obtained are summarized in Table 6.6 and illustrated in Figure 6.3.

Table 6.6 – Sensitivities to coal price, change in saleable production tons and capital expenditure

Results from Sensitivities				
Change Price	% Change			
		NPV @ 10%	IRR	Payback years
	1.2	442,123	117%	2.0
	1.1	287,700	60%	3.0
	base	130,567	29%	4.3
	0.9	(31,631)	6%	8.1
	0.8	(226,046)		
Change in Production	% Change			
		NPV @ 10%	IRR	Payback years
	1.2	377,872	88%	2.3
	1.1	255,111	52%	3.2
	base	130,567	29%	4.3
	0.9	3,038	10%	6.8
	0.8	(138,416)	-10%	
Change in Cost of Capex	% Change			
		NPV @ 10%	IRR	Payback years
	1.2	90,741	20%	5.2
	1.1	110,837	24%	4.7
	base	130,567	29%	4.3
	0.9	150,297	35%	3.9
	0.8	169,874	44%	3.5

These sensitivities are illustrated in the graphs below:

Figure 6.3 – Movement in IRR%

The IRR% in the base case is 29% which would under normal circumstances be acceptable. The original Kleinkopje expansion project had a similar projected IRR%. An increase or decrease in price, production and capital expenditure indicated that the coal price had the greatest impact. The increase in capital cost had a surprisingly small negative effect. The effect on the NPV is illustrated in Figure 6.4 below.

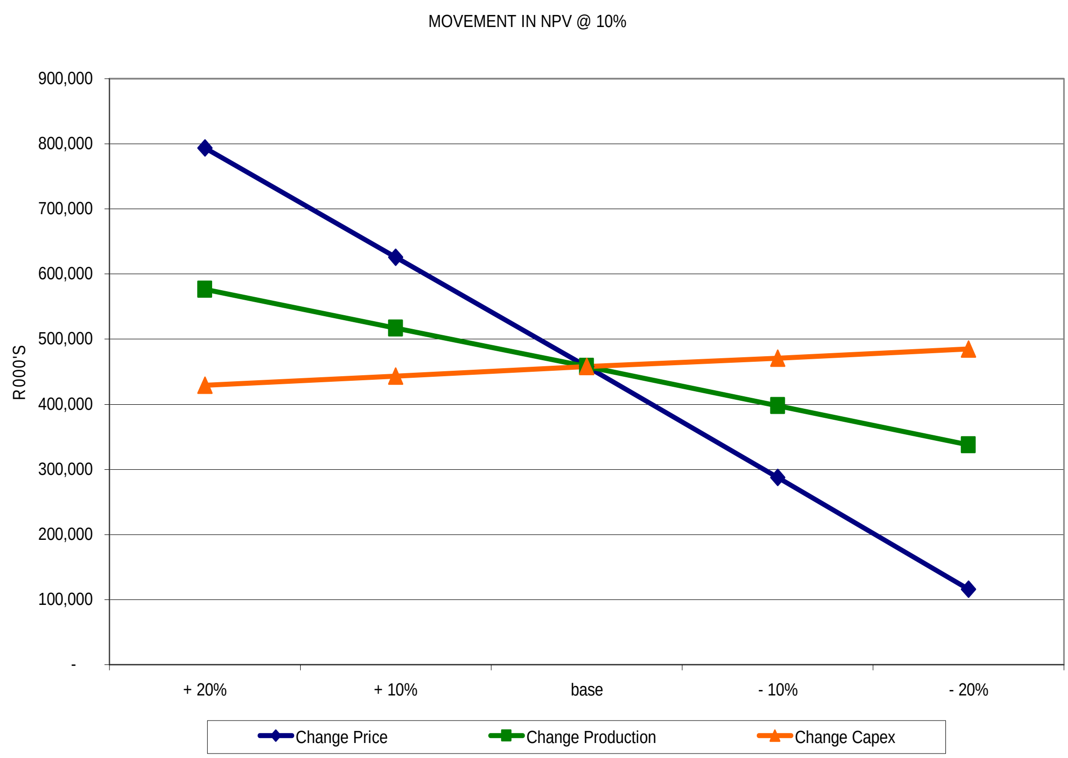


Figure 6.4 – Movement in NPV

In the case of NPV, there is a very visible difference in the sensitivity with regards to price, production and capital expenditure. The least sensitive response again is the capital expenditure. The change in production has a more pronounced impact on the NPV whilst the change in coal price has a very significant effect. This was the same as with the IRR exercise. The sensitivity of the payback period is illustrated in Figure 6.5below.

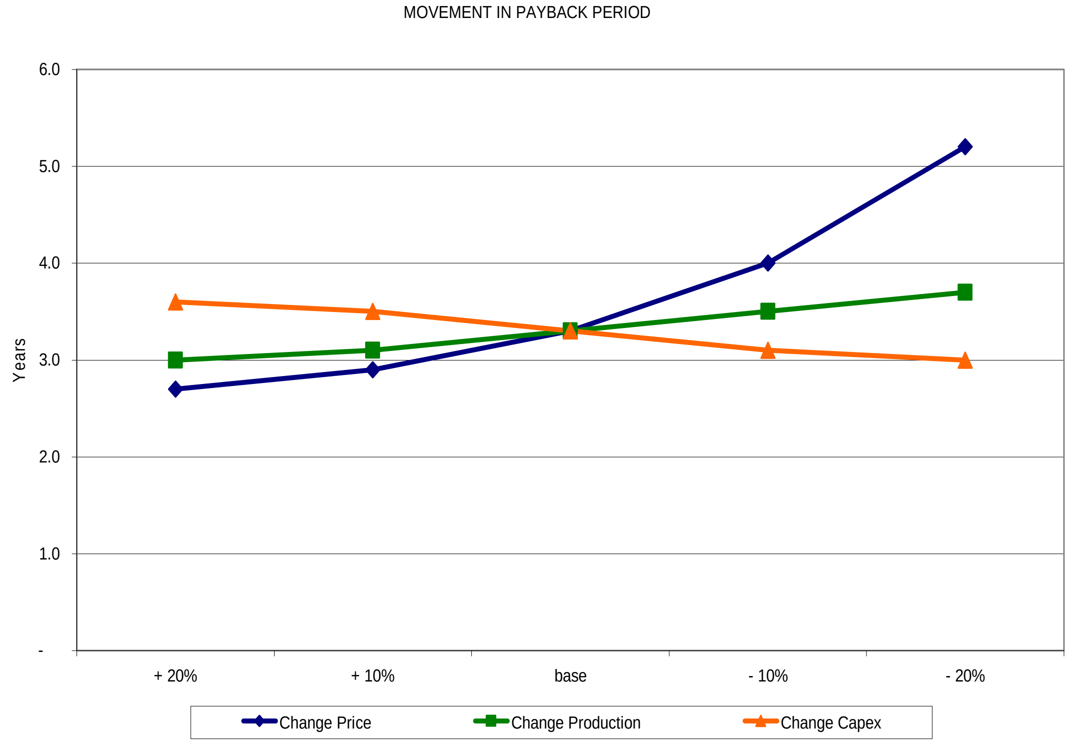


Figure 6.5 – Movement in Payback Period

The base case forecasts a payback within 4.3 years. The payback period is most sensitive to a negative change in price and least affected by a negative change in cost of capital expenditure. A drop of 20% in both price and production will make this project unviable as the capital can not be paid back over the estimated life of the project.

7. SUMMARY CONCLUSIONS AND RECOMMENDATIONS

7.1. CONCLUSION

The case study area is still to be mined, therefore no actual data is available, only best estimates can be made. Unfortunately it appears to be significantly different from any of the current mining areas which makes the estimation less accurate, and increases the risk for errors.

The main factors affecting the ROM tons and saleable tons have been highlighted as the derating percentage, mining extraction percentage, contamination percentage and fines generated percentage. The estimation of these numbers need to be improved and more accurately calculated first. They are already included in the current resource and reserve determination process. The financial model hence project viability relies on the correct inputs.

Other contributing factors, which are not included in the current estimation process, need to be included in future. A very large number relate to mining practice and standards and should be addressed by using the historically achieved figures.

The monthly and annual reconciliation process should highlight the areas of concern which must then be addressed. The risk analysis is also very important and should not be seen as a once-off exercise, but rather as a dynamic process highlighting risks before they become impacts.

If nothing else, then hopefully this treatise highlighted the inter dependencies off all the different factors and problems related to mining old workings. Ultimately the overall impact it has on the financial viability of a reserve block or mine.

7.2. RECOMMENDATIONS

The coal price and exchange rate are factors that determine the financial viability of a specific reserve at a specific time, and this should be addressed in the life of mine plan. In terms of responsible ore resource management, the more difficult and lower yielding reserves should be mined during periods of high coal prices and favourable exchange rates. This will require an optimal mix or balance between the amount of production from “easier” and more “difficult” areas, such as the North West block. The mine should also have the flexibility to change as the conditions change.

With the current coal price cycles being relatively short (refer Figure 4.5) it should be easier to manage this optimal reserve utilization. As stated in the risk analysis the coal price, exchange rate and the product demand are impacts that cannot be controlled or managed. However, the mineplan can be adjusted to maximise the positive benefit or minimise the negative impact.

As far as the manageable aspects go, there is no excuse for poor performance. In order to be able to manage the impacts it is necessary to have accurate information. Derating, for example, needs to be fixed so we have a high level of confidence in our reserve estimates.

Other impacts such as percentage mining extraction, contamination and fines generation is strongly related to mining methods and standards. Assuming the ultimate mining design or method, then standards remain an area of possible improvement. The financial viability of a reserve depends on the inherent characteristics of the reserve, the mine design and the mining standards when executing the plan.

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9. APPENDIX A – MATRIX OF FACTORS AFFECTING SALEABLE TONS

Table 1 Derating

RESERVE ESTIMATION

ROM TONS	**Comments**	**Basis**	**Input**										
Horizon		Name	M2FS										
Area		ha	482.67	482.67	482.67	482.67	482.67	482.67	482.67	482.67	482.67	482.67	482.67
Seam thickness		m	4.22	4.22	4.22	4.22	4.22	4.22	4.22	4.22	4.22	4.22	4.22
RD			1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53
GTIS	Fixed	Mt	45.582	45.582	45.582	45.582	45.582	45.582	45.582	45.582	45.582	45.582	45.582
Derating		%	32.00	33.00	34.00	35.00	36.00	37.00	38.00	39.00	40.00	41.00	42.00
Mineable area		ha											
TTIS	Coal only	Mt	30.995	30.540	30.084	29.628	29.172	28.716	28.261	27.805	27.349	26.893	26.437
Geological loss		%	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00
MTIS		Mt	28.206	27.791	27.376	26.961	26.547	26.132	25.717	25.302	24.888	24.473	24.058
MTIS/GTIS		%	61.88	60.97	60.06	59.15	58.24	57.33	56.42	55.51	54.60	53.69	52.78
Mining loss		%	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
Mining extraction		%	82.00	82.00	82.00	82.00	82.00	82.00	82.00	82.00	82.00	82.00	82.00
Recovery ROM ADB/MTIS		%	77.90	77.90	77.90	77.90	77.90	77.90	77.90	77.90	77.90	77.90	77.90
ROM tons ADB	Coal uncontaminated	Mt	21.972	21.649	21.326	21.003	20.680	20.357	20.034	19.710	19.387	19.064	18.741
Contamination		%	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00
ROM tons ADC		Mt	27.465	27.062	26.658	26.254	25.850	25.446	25.042	24.638	24.234	23.830	23.426
Inherent H2O		%	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20
Total H2O		%	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50
ROM tons as del.		**Mt**	**28.729**	**28.306**	**27.884**	**27.461**	**27.039**	**26.616**	**26.194**	**25.771**	**25.349**	**24.926**	**24.504**
ROM as del./MTIS		%	101.85	101.85	101.85	101.85	101.85	101.85	101.85	101.85	101.85	101.85	101.85
Goal Seek	Contamination adjusted		20.00	21.18	22.35	23.53	24.70	25.88	27.06	28.23	29.41	30.59	31.76
	ROM ADC adjusted		27.465	27.465	27.465	27.465	27.465	27.465	27.465	27.465	27.465	27.465	27.465
	Loss of ROM coal		0.000	0.422	0.845	1.267	1.690	2.112	2.535	2.957	3.380	3.802	4.225
	%		0.000	0.015	0.030	0.046	0.063	0.079	0.097	0.115	0.133	0.153	0.172
	ROM adjusted		**28.729**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**
SALEABLE TONS													
ROM tons as del.		Mt	28.729	28.306	27.884	27.461	27.039	26.616	26.194	25.771	25.349	24.926	24.504
Fines percentage lost		%	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00
Feed to plant excluding fines		Mt	23.845	23.494	23.143	22.793	22.442	22.091	21.741	21.390	21.039	20.689	20.338
Feed to spiral plant		Mt	4.884	4.812	4.740	4.668	4.597	4.525	4.453	4.381	4.309	4.237	4.166
Single stage wash product:													
ROM tons as del.		Mt	28.729	28.306	27.884	27.461	27.039	26.616	26.194	25.771	25.349	24.926	24.504
Feed to plant		Mt	23.845	23.494	23.143	22.793	22.442	22.091	21.741	21.390	21.039	20.689	20.338
Theoretical Yield	Excluding Contamination	%	77.52	77.52	77.52	77.52	77.52	77.52	77.52	77.52	77.52	77.52	77.52
Contamination		%	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00
Theoretical Yield	Including Contamination	%	62.02	62.02	62.02	62.02	62.02	62.02	62.02	62.02	62.02	62.02	62.02
Plant efficiency	(2)	%	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00
Borehole correlation factor (3)		%	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Plant factors	(2) * (3)	%	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00
Plant yield	(1) * (4)	%	58.92	58.92	58.92	58.92	58.92	58.92	58.92	58.92	58.92	58.92	58.92
Practical plant yield ADC	((5) minus (Fines % lost)	%	48.90	48.90	48.90	48.90	48.90	48.90	48.90	48.90	48.90	48.90	48.90
Fines product % added back	Fines product from spirals	%	5.20	5.20	5.20	5.20	5.20	5.20	5.20	5.20	5.20	5.20	5.20
Fines product tons added back (7	Fines product from spirals	Mt	1.344	1.325	1.305	1.285	1.265	1.246	1.226	1.206	1.186	1.167	1.147
Saleable tons	Feed to plant * (6) + (7)	Mt	13.004	12.813	12.622	12.431	12.239	12.048	11.857	11.666	11.475	11.283	11.092
Inherent H2O		%	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20
Contract H2O	Contract specified	%	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00
Saleable tons @ Contract H2O		**Mt**	**13.824**	**13.621**	**13.418**	**13.214**	**13.011**	**12.808**	**12.605**	**12.401**	**12.198**	**11.995**	**11.791**
Practical Yield	@ Contract H2O	%	48.12	48.12	48.12	48.12	48.12	48.12	48.12	48.12	48.12	48.12	48.12

Table 2 Mining extraction

RESERVE ESTIMATION

ROM TONS	**Comments**	**Basis**	**Input**										
Horizon		Name	M2FS										
Area		ha	482.67	482.67	482.67	482.67	482.67	482.67	482.67	482.67	482.67	482.67	482.67
Seam thickness		m	4.22	4.22	4.22	4.22	4.22	4.22	4.22	4.22	4.22	4.22	4.22
RD			1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53
GTIS		Mt	45.582	45.582	45.582	45.582	45.582	45.582	45.582	45.582	45.582	45.582	45.582
Derating		%	32.00	32.00	32.00	32.00	32.00	32.00	32.00	32.00	32.00	32.00	32.00
Mineable area		ha											
TTIS	Coal only	Mt	30.995	30.995	30.995	30.995	30.995	30.995	30.995	30.995	30.995	30.995	30.995
Geological loss		%	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00
MTIS		Mt	28.206	28.206	28.206	28.206	28.206	28.206	28.206	28.206	28.206	28.206	28.206
MTIS/GTIS		%	61.88	61.88	61.88	61.88	61.88	61.88	61.88	61.88	61.88	61.88	61.88
Mining loss		%	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
Mining extraction		%	82.00	81.00	80.00	79.00	78.00	77.00	76.00	75.00	74.00	73.00	72.00
Recovery ROM ADB/MTIS		%	77.90	76.95	76.00	75.05	74.10	73.15	72.20	71.25	70.30	69.35	68.40
ROM tons ADB	Coal uncontaminated	Mt	21.972	21.704	21.436	21.168	20.901	20.633	20.365	20.097	19.829	19.561	19.293
Contamination		%	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00
ROM tons ADC		Mt	27.465	27.131	26.796	26.461	26.126	25.791	25.456	25.121	24.786	24.451	24.116
Inherent H2O		%	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20
Total H2O		%	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50
ROM tons as del.		**Mt**	**28.729**	**28.378**	**28.028**	**27.678**	**27.327**	**26.977**	**26.626**	**26.276**	**25.926**	**25.575**	**25.225**
ROM as del./MTIS		%	101.85	100.61	99.37	98.13	96.88	95.64	94.40	93.16	91.92	90.67	89.43
Goal Seek	Contamination adjusted		20.00	20.97	21.95	22.93	23.90	24.88	25.85	26.83	27.80	28.78	29.76
	ROM ADC adjusted		27.465	27.465	27.465	27.465	27.465	27.465	27.465	27.465	27.465	27.465	27.465
	Loss of ROM coal		0.000	0.350	0.701	1.051	1.401	1.752	2.102	2.452	2.803	3.153	3.503
	%		0.000	0.012	0.025	0.038	0.051	0.065	0.079	0.093	0.108	0.123	0.139
	ROM adjusted		**28.729**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**	**28.729**
SALEABLE TONS													
ROM tons as del.		Mt	28.729	28.378	28.028	27.678	27.327	26.977	26.626	26.276	25.926	25.575	25.225
Fines percentage lost		%	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00
Feed to plant excluding fines		Mt	23.845	23.554	23.263	22.972	22.682	22.391	22.100	21.809	21.518	21.228	20.937
Feed to spiral plant		Mt	4.884	4.824	4.765	4.705	4.646	4.586	4.527	4.467	4.407	4.348	4.288
Single stage wash product:													
ROM tons as del.		Mt	28.729	28.378	28.028	27.678	27.327	26.977	26.626	26.276	25.926	25.575	25.225
Feed to plant		Mt	23.845	23.554	23.263	22.972	22.682	22.391	22.100	21.809	21.518	21.228	20.937
Theoretical Yield	Excluding Contamination	%	77.52	77.52	77.52	77.52	77.52	77.52	77.52	77.52	77.52	77.52	77.52
Contamination		%	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00
Theoretical Yield	Including Contamination	%	62.02	62.02	62.02	62.02	62.02	62.02	62.02	62.02	62.02	62.02	62.02
Plant efficiency	(2)	%	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00
Borehole correlation factor	(3)	%	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Plant factors	(2) * (3)	%	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00
Plant yield	(1) * (4)	%	58.92	58.92	58.92	58.92	58.92	58.92	58.92	58.92	58.92	58.92	58.92
Practical plant yield ADC	((5) minus (Fines % lost)	%	48.90	48.90	48.90	48.90	48.90	48.90	48.90	48.90	48.90	48.90	48.90
Fines product % added back	Fines product from spirals	%	5.20	5.20	5.20	5.20	5.20	5.20	5.20	5.20	5.20	5.20	5.20
Fines product tons added back (7	Fines product from spirals	Mt	1.344	1.328	1.312	1.295	1.279	1.263	1.246	1.230	1.213	1.197	1.181
Saleable tons	Feed to plant * (6) + (7)	Mt	13.004	12.846	12.687	12.529	12.370	12.212	12.053	11.894	11.736	11.577	11.419
Inherent H2O		%	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20
Contract H2O	Contract specified	%	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00
Saleable tons @ Contract H2O		**Mt**	**13.824**	**13.656**	**13.487**	**13.319**	**13.150**	**12.981**	**12.813**	**12.644**	**12.476**	**12.307**	**12.138**
Practical Yield	@ Contract H2O	%	48.12	48.12	48.12	48.12	48.12	48.12	48.12	48.12	48.12	48.12	48.12

Table 3 Contamination

RESERVE ESTIMATION

ROM TONS	**Comments**	**Basis**	**Input**										
Horizon		Name	M2FS										
Area		ha	482.67	482.67	482.67	482.67	482.67	482.67	482.67	482.67	482.67	482.67	482.67
Seam thickness		m	4.22	4.22	4.22	4.22	4.22	4.22	4.22	4.22	4.22	4.22	4.22
RD			1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53
GTIS		Mt	45.582	45.582	45.582	45.582	45.582	45.582	45.582	45.582	45.582	45.582	45.582
Derating		%	32.00	32.00	32.00	32.00	32.00	32.00	32.00	32.00	32.00	32.00	32.00
Mineable area		ha											
TTIS	Coal only	Mt	30.995	30.995	30.995	30.995	30.995	30.995	30.995	30.995	30.995	30.995	30.995
Geological loss		%	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00
MTIS		Mt	28.206	28.206	28.206	28.206	28.206	28.206	28.206	28.206	28.206	28.206	28.206
MTIS/GTIS		%	61.88	61.88	61.88	61.88	61.88	61.88	61.88	61.88	61.88	61.88	61.88
Mining loss		%	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
Mining extraction		%	82.00	82.00	82.00	82.00	82.00	82.00	82.00	82.00	82.00	82.00	82.00
Recovery ROM ADB/MTIS		%	77.90	77.90	77.90	77.90	77.90	77.90	77.90	77.90	77.90	77.90	77.90
ROM tons ADB	Coal uncontaminated	Mt	21.972	21.972	21.972	21.972	21.972	21.972	21.972	21.972	21.972	21.972	21.972
Contamination		%	20.00	21.00	22.00	23.00	24.00	25.00	26.00	27.00	28.00	29.00	30.00
ROM tons ADC	Coal contaminated	Mt	27.465	27.813	28.170	28.536	28.911	29.296	29.692	30.099	30.517	30.947	31.389
Inherent H2O		%	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20
Total H2O		%	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50
ROM tons as del.		**Mt**	**28.729**	**29.092**	**29.465**	**29.848**	**30.241**	**30.644**	**31.058**	**31.483**	**31.921**	**32.370**	**32.833**
ROM as del./MTIS		%	101.85	103.14	104.46	105.82	107.21	108.64	110.11	111.62	113.17	114.76	116.40
Goal Seek	Mining extraction adjusted		82.00	80.97	79.95	78.92	77.90	76.87	75.85	74.82	73.80	72.77	71.75
	Recovery ROM ADB/MTIS		77.90	76.92	75.95	74.98	74.00	73.03	72.06	71.08	70.11	69.14	68.16
	ROM ADB		21.972	21.697	21.423	21.148	20.873	20.599	20.324	20.049	19.775	19.500	19.226
	ROM ADC adjusted		27.465	27.465	27.465	27.465	27.465	27.465	27.465	27.465	27.465	27.465	27.465
	Loss of ROM coal		0.000	-0.364	-0.737	-1.119	-1.512	-1.915	-2.329	-2.755	-3.192	-3.642	-4.104
	%		0.000	-0.013	-0.025	-0.038	-0.050	-0.063	-0.075	-0.088	-0.100	-0.113	-0.125
	ROM adjusted		**28.729**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**	**28.728**
SALEABLE TONS													
ROM tons as del.		Mt	28.729	29.092	29.465	29.848	30.241	30.644	31.058	31.483	31.921	32.370	32.833
Fines percentage lost		%	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00
Feed to plant excluding fines		Mt	23.845	24.147	24.456	24.774	25.100	25.434	25.778	26.131	26.494	26.867	27.251
Feed to spiral plant		Mt	4.884	4.946	5.009	5.074	5.141	5.209	5.280	5.352	5.427	5.503	5.582
Single stage wash product:													
ROM tons as del.		Mt	28.729	29.092	29.465	29.848	30.241	30.644	31.058	31.483	31.921	32.370	32.833
Feed to plant		Mt	23.845	24.147	24.456	24.774	25.100	25.434	25.778	26.131	26.494	26.867	27.251
Theoretical Yield	Excluding Contamination	%	77.52	77.52	77.52	77.52	77.52	77.52	77.52	77.52	77.52	77.52	77.52
Contamination		%	20.00	21.00	22.00	23.00	24.00	25.00	26.00	27.00	28.00	29.00	30.00
Theoretical Yield	Including Contamination	%	62.02	61.24	60.47	59.69	58.92	58.14	57.36	56.59	55.81	55.04	54.26
Plant efficiency	(2)	%	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00
Borehole correlation factor	(3)	%	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Plant factors	(2) * (3)	%	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00
Plant yield	(1) * (4)	%	58.92	58.18	57.44	56.71	55.97	55.23	54.50	53.76	53.02	52.29	51.55
Practical plant yield ADC	((5) minus (Fines % lost)	%	48.90	48.29	47.68	47.07	46.45	45.84	45.23	44.62	44.01	43.40	42.79
Fines product % added back	Fines product from spirals	%	5.20	5.20	5.20	5.20	5.20	5.20	5.20	5.20	5.20	5.20	5.20
Fines product tons added back (7	Fines product from spirals	Mt	1.344	1.362	1.379	1.397	1.415	1.434	1.454	1.473	1.494	1.515	1.537
Saleable tons	Feed to plant * (6) + (7)	Mt	13.004	13.021	13.039	13.057	13.075	13.094	13.113	13.133	13.154	13.175	13.197
Inherent H2O		%	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20
Contract H2O	Contract specified	%	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00
Saleable tons @ Contract H2O		**Mt**	**13.824**	**13.842**	**13.861**	**13.880**	**13.900**	**13.920**	**13.940**	**13.961**	**13.983**	**14.005**	**14.028**
Practical Yield	@ Contract H2O	%	48.12	47.58	47.04	46.50	45.96	45.42	44.88	44.35	43.81	43.27	42.73

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Table 4 Fines percentage lost

RESERVE ESTIMATION

ROM TONS	Comments	Basis	Input										
Horizon		Name	M2FS										
Area		ha	482.67	482.67	482.67	482.67	482.67	482.67	482.67	482.67	482.67	482.67	482.67
Seam thickness		m	4.22	4.22	4.22	4.22	4.22	4.22	4.22	4.22	4.22	4.22	4.22
RD			1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53
GTIS		Mt	45.582	45.582	45.582	45.582	45.582	45.582	45.582	45.582	45.582	45.582	45.582
Derating		%	32.00	32.00	32.00	32.00	32.00	32.00	32.00	32.00	32.00	32.00	32.00
Mineable area		ha											
TTIS	Coal only	Mt	30.995	30.995	30.995	30.995	30.995	30.995	30.995	30.995	30.995	30.995	30.995
Geological loss		%	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00
MTIS		Mt	28.206	28.206	28.206	28.206	28.206	28.206	28.206	28.206	28.206	28.206	28.206
MTIS/GTIS		%	61.88	61.88	61.88	61.88	61.88	61.88	61.88	61.88	61.88	61.88	61.88
Mining loss		%	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
Mining extraction		%	82.00	82.00	82.00	82.00	82.00	82.00	82.00	82.00	82.00	82.00	82.00
Recovery ROM ADB/MTIS		%	77.90	77.90	77.90	77.90	77.90	77.90	77.90	77.90	77.90	77.90	77.90
ROM tons ADB	Coal uncontaminated	Mt	21.972	21.972	21.972	21.972	21.972	21.972	21.972	21.972	21.972	21.972	21.972
Contamination		%	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00
ROM tons ADC		Mt	27.465	27.465	27.465	27.465	27.465	27.465	27.465	27.465	27.465	27.465	27.465
Inherent H2O		%	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20
Total H2O		%	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50
ROM tons as del.		**Mt**	**28.729**	**28.729**	**28.729**	**28.729**	**28.729**	**28.729**	**28.729**	**28.729**	**28.729**	**28.729**	**28.729**
ROM as del./MTIS		%	101.85	101.85	101.85	101.85	101.85	101.85	101.85	101.85	101.85	101.85	101.85
SALEABLE TONS													
ROM tons as del.		Mt	28.729	28.729	28.729	28.729	28.729	28.729	28.729	28.729	28.729	28.729	28.729
Fines percentage lost		%	17.00	17.50	18.00	18.50	19.00	19.50	20.00	20.50	21.00	21.50	22.00
Feed to plant excluding fines		Mt	23.845	23.701	23.557	23.414	23.270	23.126	22.983	22.839	22.696	22.552	22.408
Feed to spiral plant		Mt	4.884	5.027	5.171	5.315	5.458	5.602	5.746	5.889	6.033	6.177	6.320
Single stage wash product:													
ROM tons as del.		Mt	28.729	28.729	28.729	28.729	28.729	28.729	28.729	28.729	28.729	28.729	28.729
Feed to plant		Mt	23.845	23.701	23.557	23.414	23.270	23.126	22.983	22.839	22.696	22.552	22.408
Theoretical Yield	Excluding Contamination	%	77.52	77.52	77.52	77.52	77.52	77.52	77.52	77.52	77.52	77.52	77.52
Contamination		%	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00
Theoretical Yield	Including Contamination	%	62.02	62.02	62.02	62.02	62.02	62.02	62.02	62.02	62.02	62.02	62.02
Plant efficiency	(2)	%	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00
Borehole correlation factor (3)		%	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Plant factors	(2) * (3)	%	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00
Plant yield	(1) * (4)	%	58.92	58.92	58.92	58.92	58.92	58.92	58.92	58.92	58.92	58.92	58.92
Practical plant yield ADC	((5) minus (Fines % lost)	%	48.90	48.61	48.31	48.02	47.72	47.43	47.13	46.84	46.54	46.25	45.95
Practical plant yield ADC	Gradecon yield	%	48.90	48.90	48.90	48.90	48.90	48.90	48.90	48.90	48.90	48.90	48.90
Fines product % added back	Fines product from spirals	%	5.20	5.20	5.20	5.20	5.20	5.20	5.20	5.20	5.20	5.20	5.20
Fines product tons added back (7	Fines product from spirals	Mt	1.344	1.344	1.344	1.344	1.344	1.344	1.344	1.344	1.344	1.344	1.344
Saleable tons	Feed to plant * (6) + (7)	Mt	13.004	12.864	12.725	12.587	12.449	12.313	12.177	12.042	11.908	11.774	11.642
Inherent H2O		%	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20
Contract H2O	Contract specified	%	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00
Saleable tons @ Contract H2O		**Mt**	**13.824**	**13.675**	**13.527**	**13.380**	**13.234**	**13.089**	**12.944**	**12.801**	**12.658**	**12.517**	**12.376**
Practical Yield	@ Contract H2O	%	48.12	47.60	47.09	46.58	46.07	45.56	45.06	44.56	44.06	43.57	43.08