# Optimal production rates in opencast coal mining: A value driven approach

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## ABSTRACT

From small exploration companies to multi-national mining houses all at some point in the project lifetime embark on evaluation studies where the most value-generating method of extracting the ore is investigated. Early phases in exploration projects will have the need for an order-of-magnitude estimation as to the scale of the potential operation, and advanced projects will have detailed mine and financial plans to guide them to execution. In both instances this thesis provides a method of optimising the mining rate to deliver the highest possible value to the mining company whilst considering the possible risks from changes in the market. This can be compared to the value the country gains from the exploitation of its natural resources to find a mutually beneficial solution.

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# LIST OF DEFINITIONS

NPV – Net present value -   Sum of the time adjusted cash flow values of the operation using the cost of capital as the discount rate

Maximum extraction -   An operational strategy to remove as much ore as possible. This strategy can be become dependant to costs and ore prices.

Mine life -   The time in which the mine remains operational, from start-up to mine closure.

Mining Rate -   The average rate at which mining takes place within a certain time period (normally per annum). The mining rate can refer to coal extracted per annum or to the waste stripped per annum.

High Grading -   A mining practice in which the higher quality coal is mined first in the mine life. This generally creates higher revenue streams early in the operation but can have limiting consequences later on.

Saleable coal -   The product coal produced according to a customer's specification, which can be after coal treatment (beneficiation) or untreated coal (raw coal).

Capital Costs -   The costs incurred to build or buy equipment or buildings that allow the mine to become operational.

Resources -   Amount of coal in the ground that has the potential of eventual economic extraction

Overburden targeting -   A mine scheduling principal that focuses on waste removal as the operational constraint and all subsequent activities as a function thereof.

Resource based mine plan -   A mine scheduling approach that treats each piece of equipment as a resource with a certain capacity. It is a very detailed method as precise equipment movements are simulated.

**APPENDICES**

# 1.    Background

As per the Code of conduct of the South African Council for Natural Scientific persons, a natural scientist must have due regard for the safety, health and general interest of the public (SACNASP, 2011).   As per the Mineral and Petroleum Resources and Development Act, the natural resources of South Africa are owned by the citizens of the country and are to be managed by the government.  It can, therefore, clearly be seen that it lies within the responsibilities of the natural scientist to ensure maximum utilisation of these non-renewable natural resources, so as to maximize value to the general public, whilst aiding companies to maximize profit in order to attract investment.  The impact of this responsibility can be far reaching and encompasses most of the essential ethical conduct required by the SACNASP.    It also leaves some room for interpretation as maximum utilisation and maximum value are relative terms.  In general these terms have driven geologists to ensure maximum resource extraction and minimizing contamination, and although these are virtuous practices, the long term value of the ore towards the public, or the company, is not always addressed.  It also does not consider the profitability of the deposit should market conditions change, and for the government to gain revenue from the exploitation of their natural resources, it must remain a profitable and attractive business for the mining companies as well.

Resource utilisation is conventionally seen as part of the Reserve Management field, which forms part of the portfolio of mining engineering; however, Reserve Management so often imposes short to medium term targets on the planning process or system, and longer term effects are often neglected.  This thus highlights a shortcoming in the current system where the ethical responsibility of the natural scientist to ensure maximum value from a resource needs to be considered along with any short to medium term requirements or constraints in the conventional planning model.

Apart from the responsible management of the resources, this evaluation approach can serve as a guideline during early phases in exploration projects where the mineral right owners have the need to establish a reasonable estimation of the scale of the potential operation.  This evaluation will also give guidance to the magnitude of the capital required to advance the project to execution.  This might reduce some of the risk expected in early exploration studies (Pincock, 2004).

For the purpose of the study exploration and sampling data from the Tweefontein complex will be used.  The Tweefontein complex is located 30km west of Emalahleni and falls within the Witbank coal field (Figure 1.1).  The Witbank Coal Fields contain five coal seams numbered No. 1 at the bottom of the sequence to No. 5 at the top as illustrated by Figure 1.2.
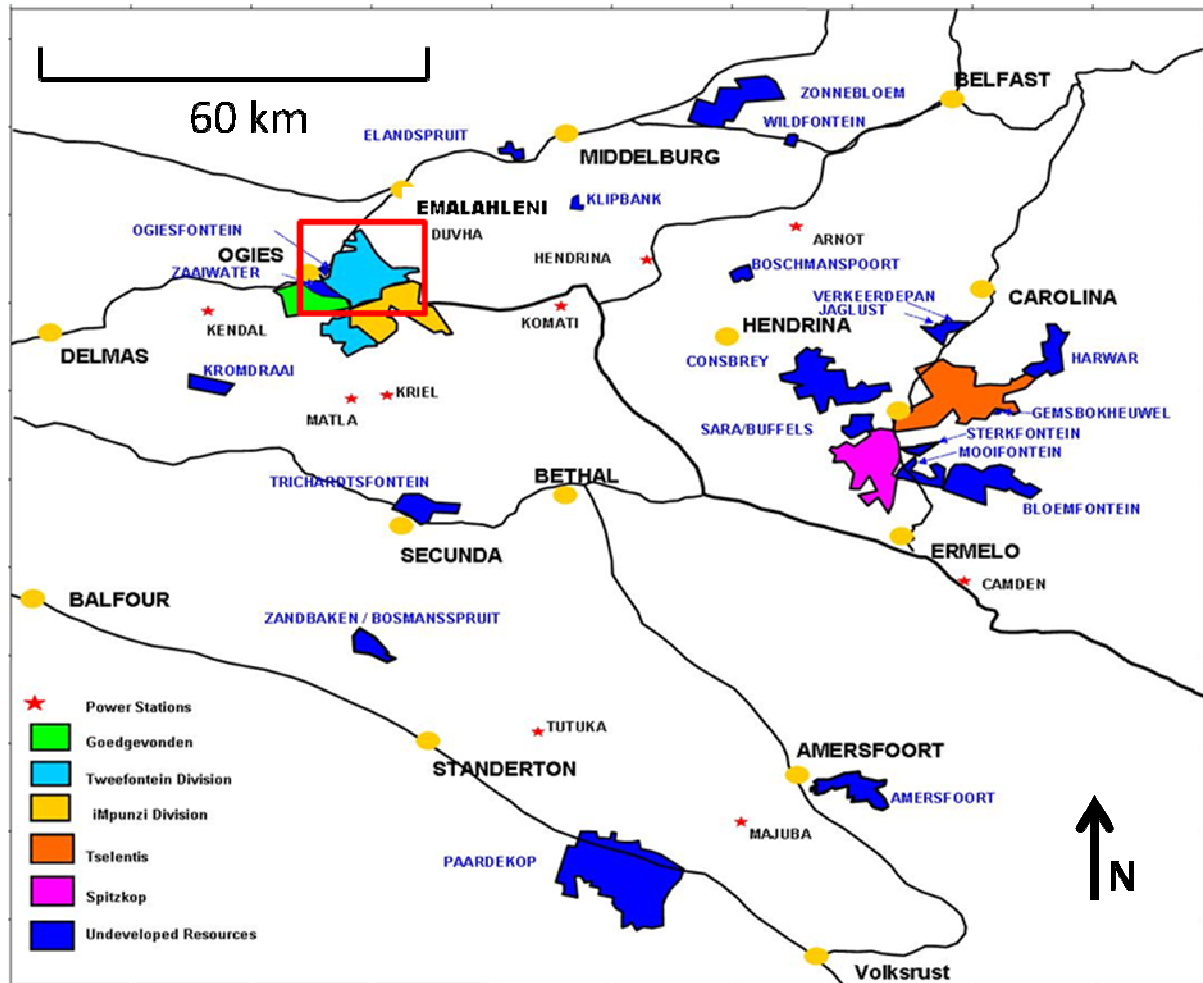
**Figure 1.1  Locality of the Tweefontein Complex in relation to Emalahleni and the other Xstrata Coal South Africa deposits (modified from Smith, 2009).**

## 2.    Introduction

Maximum value is a vague term, as this could refer to maximum net present value (NPV), maximum extraction, maximum mine life or a combination of any of the above focus points.  This thesis will attempt to maximize the value by optimising the mining rate.  As the mining rate drives so many factors in the project process, from capital to life of the mine, it becomes the key focus point of this paper, as seen in other utilisation studies (Smith, 1997; Godoy and Dimitrakopoulos, 2004).  The advantages and disadvantages of the different focus points are:

### 2.1    Maximum NPV

This method is commonly used in isolation, where a mining schedule would produce a production report, which in turn serves as input into a financial model that calculates the NPV.  The mining layouts are then re-evaluated in an attempt to maximize the NPV.  Other short term constraints then govern the process.  NPV calculations employ the time value principals and have been criticized as they often lead to chronic high-grading of the ore deposit.  High grading will ensure high revenue in the short term, but as time progresses the country will eventually have only poorer quality coal.  In general the highest amount of

saleable product in the shortest amount of time will lead to a high NPV, however as production rate increases so do the capital costs of the operation. Other studies, as done by Godoy and Dimitrakopoulos (2004) also utilise NPV as the defining characteristic of value.
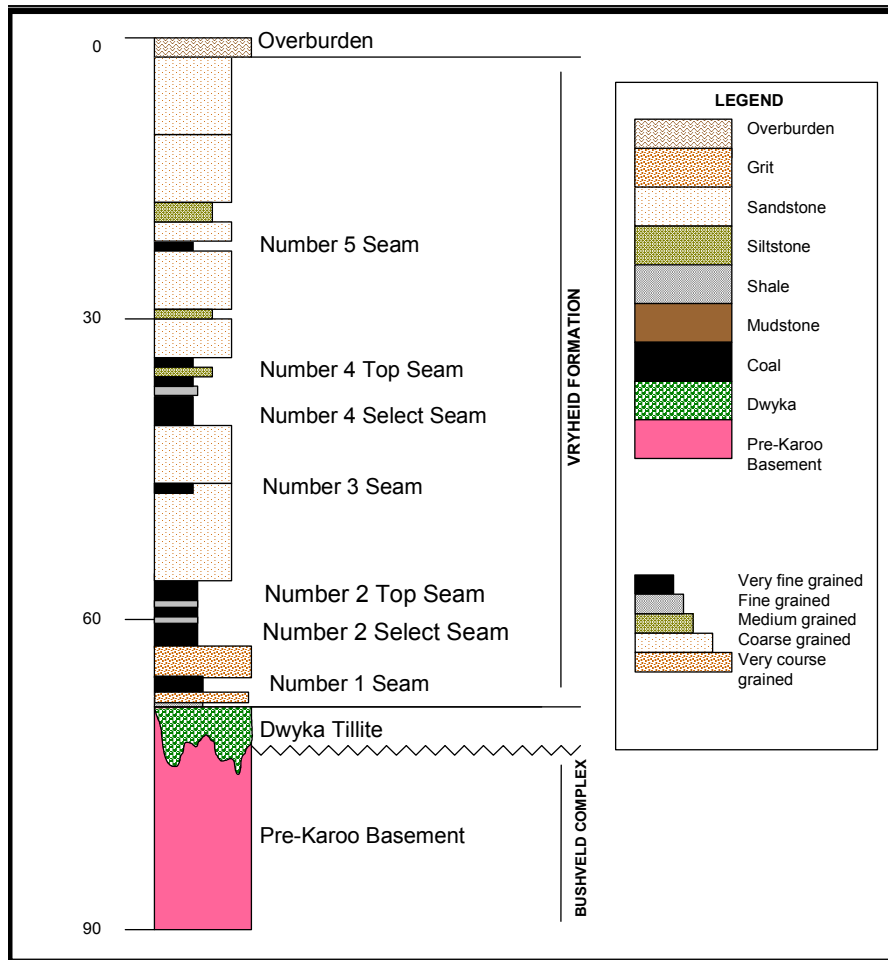


**Figure 1.2 General stratigraphy for the central Witbank Coal Field including the Tweefontein complex (modified from Smith, 2009).**

## 2.2    Maximum extraction

Extracting as much of the ore as possible is a very important component of mine planning, but many practical facets of mining normally take precedence, otherwise, for example: bord and pillar mining would never be considered. In isolation maximizing extraction also does not consider the financial impacts and time related implications of mining practices.

## 2.3    Maximum mine life

When considering mine life, the period in which the mining practice contributes towards the country and the mining company is evaluated. For a fixed size resource, maximizing the mine life can generally only be done by increasing extraction (discussed above) or by reducing the mining rate, and by reducing the mining rate less start-up capital is

required. The rate can obviously not be infinitely reduced as the value generated for the company will become insignificant, and the company might have supply obligations that need to be met.

## 2.4 Combination of focus points

A holistic approach must be taken to determine the maximum value to the company, whilst not over-utilizing the resource, considering the constraints and satisfying the most significant short term and mid term targets.

# 3. Aim of the project

It is the aim of this project is to –

- Develop a method to complete multiple mining simulations in a fast and simple way. The output schedule must be utilized in a financial model where financial tools such as NPV, IRR and sensitivity analysis can be completed.
- Determine the effect of risk on the optimal mining rate.
- Determine the optimal mining rate for various resource blocks within a complex.
- Combine the mine schedules of the blocks to verify whether the short term obligations have been met and adjust the necessary parameters.
- Determine which inherent property of the resource itself will drive the mining rate.

# 4. Methodology

## 4.1 Geological Data

The starting point for any evaluation is a reliable geological model.

### 4.1.1 Structural model

Generally for coal deposit models core drilling data are used to construct a structural (or spatial) model. The depth and thickness of each ore intersection is logged in a database. The coordinates of the collar, as well as the depth of the borehole is entered into the geological model and the seam intersections are then loaded into the model. From this information the model can calculate the coordinates for the seam floor, seam thickness and seam roof within each of the boreholes. By using any one of many generally accepted geostatistical interpolation methods (such as inverse distance weighted), an estimation of the position and quality of the entire deposit can be created.

### 4.1.2 Quality model

The core extracted during exploration is also sampled and analysed. The standard analyses that are completed included proximate analyses, calorific value and sulphur. Instead of completing these on the entire sample only, the analyses are done on various density fractions separated by sink-float-analyses. This will give an indication of the washability of the coal sample, in other words specifying the calorific value (and other quality parameters) and relative proportion of each density fraction. This information is absolutely crucial when considering that most of the South African coals need to be upgraded or beneficiated to obtain export specific energy values. Companies use various plant simulation software packages that utilize the washability information to determine product yields for specified energy values depending on their client's requirements. The output data from a plant simulation is then re-entered into the geological model along with the raw data from the analyses and interpolated using the same geostatistical methods to create continuous data across the deposit that can be used to calculate average values per area or block.

## 4.2    Layout design

### 4.2.1 Block sizes

As the model will be continuously simulated using various mining rates, the length of the mining blocks will not have much of an impact. There has to be a balance between computation and accuracy, as too large blocks will result in decreased accuracy and flexibility. With too small blocks a point exists where there is no gain in accuracy but computing requirements increase exponentially. Therefore by using a standard block width of 50m, the length was also set to 50m.

### 4.2.2 Mining sequence

As this thesis does not dispute the benefit of mining from more profitable towards lower profitable ore zones, this approach will be used as well. The simplest representation of profitability is saleable strip ratio (cubic meters waste / saleable product tons) as this considers the amount of waste to be removed relative to the amount of revenue generating coal within a specified area. Thus by mining from a low saleable strip ratio to a higher strip ratio will generally result in the highest profit margins early in the life of the mine, resulting in higher NPV values. By considering the thicknesses of the soft and hard overburden, as well as coal thicknesses, it can be seen how these contribute towards the saleable strip ratio.

The mining sequence was in all cases planned from the low strip ratio area to the high strip ratio areas.

## 4.3    Mine scheduling

### 4.3.1 Elements to be used

In reality, mining occurs in phases through the geological horizons, but in most long to medium term planning a 'cookie-cutter' approach is taken, whereby the assumption is made that mining occurs instantaneously through each mining block, thus all costs related to mining that block from the surface to the floor of the ore-body are incurred immediately. This approach is not as accurate as a resource or activity based schedule, but does give a fair representation of the mining volumes over time. To create a phased mining model will require expensive mining simulation software not available for this project. The elements of the mining operation to be scheduled are the following -

- Topsoil
- Soft overburden
- Hard overburden
- No. 4 Upper seam
- No. 4 Select seam
- Interburden
- No. 2 Upper seam
- No. 2 Select seam
- Rehabilitation

The coal beneficiation is then simulated and the following product tonnages are also recorded –

- Primary Export 6000kCal NAR
- Secondary (middlings) Eskom (21 MJ/kg)
- Primary Eskom (21 MJ/kg)

Cut-off grades applied to the coal ore are much more simplistic than with metal mining (Dowd, 1994) as practical limitations govern the process. A plant yield limit of 30% was used to reflect the capabilities of the basic cyclones used in coal beneficiation. Coal not meeting the 30% yield limit, will be washed for Eskom. If the 30% plant yield limit is not met for Eskom products, the coal will be sold to Eskom as a raw low-grade product.

Due to the significant cost impact of mining boxcut waste versus mining in a steady state, the waste volumes will be split between –

- Boxcut mining
- Steady state mining

### *4.3.2   Overburden targeting*

The data for the various mining blocks are then reported into a spreadsheet containing all the aspects to be scheduled.  The schedule is based on a target and fixed total overburden rate, with the ROM coal and product coal tonnages resulting from the overburden removal.  The is the most realistic method of scheduling as the overburden removal tends to be the biggest constraint of the mining operation as the volumes are large compared to coal removal, thus the mining equipment is chosen based on the waste volumes (Godoy and Dimitrakopoulos, 2004).

The scheduling is then done by creating a matrix with the total overburden per block along the Y-axis in the order of mining, and time along the X-axis.  A target annual waste removal is chosen and the volumes are added systematically from the initial blocks until the target for the year is reached.  Should the annual target be met half way through a mining block, the remaining overburden in the block is then carried over to the next year, where the simulation continues until the target is reached for the second year and so it continues until the total waste volumes have been accounted for.

### *4.3.3   Schedule weights*

From the waste schedule exercise the blocks that are mined for each year are recorded.  These blocks are used to determine the percentage of each block mined in a given year which are then used to determine any other element of the block included in that year, by multiplying that element with the weight and adding the values for the year.   This will produce a mining schedule for the life of the mine, containing all the elements needed to create the financial model.

## 4.4   Financial Model

### *4.4.1   Basic costing*

The basic costs of mining operations are applied to the various elements stated in the mining schedule to determine the costs of mining.  These are –

- Topsoil removal
- Pre-strip removal of soft overburden
- Hard overburden drilling
- Hard overburden blast
- Bull dozer pushover of two thirds of the blasted waste
- Truck and shovel removal of blasted waste
- Parting drill, blast and removal

- Coal drill, blast and removal

- Pit services

- Coal transport costs to wash plants

- Washing cost

- Product handling

- Discard handling

- Rehabilitation cost

- Labour costs

- General overheads

The start-up capital is driven by the required waste removal capacity as this determines the necessary equipment. The revenue gained from the mining operation is calculated from the product tonnages based on the three products specified in 4.3.1.

### 4.4.2 Cash flow

The cash flow statement is produced from the revenue and costs of the various elements. The taxable income is determined by carrying the start-up capital over from one year to the next until a positive balance is reached, upon which taxation can be calculated. Royalties are also calculated on the taxable income. By subtracting the taxation and royalties, a net profit is determined which along with the start-up capital yields a cash flow for the purpose of calculating the NPV of the mining schedule.

### 4.4.3 Adjusting the mining rate

By adjusting the mining rate the start-up capital is adjusted accordingly with the cost of the equipment. The financial model is also updated with the new volumes and tonnages, yielding a new cash flow statement and new NPV. Therefore by adjusting the mining rate and tracking the resulting NPV, a curve can be plotted of the NPV for the various mining rates. From this a maximum point is determined, which then indicates the optimal mining rate for the ore reserve.

### 4.4.4 Price and cost fluctuations

As per typical sensitivity analyses, the price and cost used in the financial reports will be fluctuated positively and negatively to test the impact thereof on the NPV. These will be adjusted independently so as to isolate the impact of a single variable without influence from the other.

The approach taken is that the current costs and prices are well understood and variances in the following year (and couple of years thereafter) from the planned values will

be small, and as time progresses the risk of large changes becomes more pertinent, in other words the costs and prices cannot be adjusted by a fixed percentage for the entire life of the operation. Thus the approach taken is to adjust the cost and price escalation. The escalation is varied from -10%, -5%, 0%, +5% to +10%, and the NPV is recorded at each point. The NPV at 0% is called NPVbase.

### 4.4.5 Risk adjustment factors

The are numerous risks to exploration and mining projects, and these need to be investigated fully during the various project cycles, such as suggested by Rodger and Petch (1999). For the purpose of this study, only the basic financial risks are to be considered. The optimal mining rate does not indicate or account for variability in the financial model, and therefore adjustment factors should be created that consider such variability and risk. Since this will have a fundamental impact on the optimal rate, these factors need to be investigated in detail. A big concern for a project is how sensitive it is to changes within the coal market. Should costs unexpectedly increase over the life of the project, the viability of the project might change. Similarly should the ore price decrease more than the estimated forecast, the return from the project might not be as favourable as previously expected. As much as these risks exist, similarly the opportunity may arise where costs decrease (less likely, but still plausible) or commodity prices rapidly increase. Should the project benefit from these scenarios under certain conditions, it needs to be taken into account. Therefore the risk adjustment accounts for two conditions: Sensitivity to future changes and opportunity from future changes. Sensitivity will be measured by the NPV. The two aspects of risk that can be accounted for are –

#### 4.4.5.1 NPV consistency

The deviation or the extent of the NPV range gives an indication of how sensitive the NPV is to the price/cost changes. A large variation will indicate a very unstable or sensitive deposit, where a narrow range will indicate stability within the value of the deposit regardless of changes to the price and cost. The adjustment factor can be calculated as per equation 1, where *NPVi* is the various NPV values gained by adjusting the price and cost escalations, and *NPVbase* is the unadjusted NPV value.

*Standard deviavation (NPVi) / NPVbase = Xsensitivity*                *(1)*

#### 4.4.5.2 Skewness of the NPV range

If a NPV range shows a large deviation, but this deviation is positively skewed, it will be an enormous advantage as it indicates that the leverage of the NPV to the price or cost changes is high positively but low negatively. This means the risk is lower for positively skewed deposits and higher with negatively skewed deposits. The adjustment factor can then be calculated based on this principal, using equation 2.

$$NPVbase / average (NPVi) = Xskew \qquad\qquad (2)$$

Higher values for *Xsensitivity* and *Xskew* will indicate higher risk and thus the original calculated NPV is then divided by the average of the two risk factors to give an adjusted NPV. As the NPV has been adjusted by estimated risk factors, the actual value of the new adjusted NPV is not of importance, but purely the comparison between the new numbers, in order to determine a maximum value at a specified mining rate.

## 4.5 Company scheduling

Once the optimal and risk adjusted mining rates have been determined for the various deposits, the info can be used retrospectively in existing operations or from the start in new operations to determine the manner in which the different deposits should be combined in the production profile of the company.

### 4.5.1 Retrospective application

To determine the effectiveness of the current mine plan, the actual mining rate must be plotted on a graph versus the optimal and risk adjusted rate with a 1:1 line plotted as well. In this manner it can easily be seen which operations are being under or over utilized and necessary adjustment can be brought into place. This is actually more complicated in existing operations as the capital has already been spent, and any changes in the operations will result in additional capital. This must be considered in the financial calculation before adjustments are suggested.

### 4.5.2 Application in new operations

In new operations the application is much simpler as the analysis considers the capital to be spent upon start-up where the optimal rate is dependant on the capital. It does become more difficult when short term practical considerations come into effect, such as plant feed capacity, railing capacity, export allocation and stockpile sizes. The deposits must be rescheduled and combined into a complete profile. From this the waste removal, run-of-mine tons, product tons and stockpile balances can be considered. Any changes made to the mining rate can be compared to the optimal and risk adjusted rate, in order to recognize the need for additional resources rather than over- or under-utilisation of the current resources.

## 4.6 Tax impact

As a responsible scientist, it is necessary to evaluate and understand the value the country gains from the mining operation. By changing the mining rate we impact in numerous manners on this value, from the period over which the company employs local citizens, the company tax, royalties and duties or VAT on any imported goods or equipment.

Oyinlola (2003) states the different objectives of the government and the companies, but ultimately agrees that "it is therefore important to reconcile these interests in order for both objectives to be obtained so as to sustain investment".

From the taxation profile over time a present value calculation can also be done to determine the value from the mining operations at various mining rates.

# 5.    Results

## 5.1    Geological data

The exploration data for the Tweefontein colliery was made available for this project and a geological model was created using Minex 6 modelling software.  The deposit contains five separate mining operations each to be evaluated individually before a holistic combined mining program is determined.  The basic geological features are indicated in Appendix C. For each of the operations the following geological parameters were investigated:

### 5.1.1    Exploration details

1,438 boreholes were drilled on the Tweefontein Division Project area in the period 1964 to 2010 of which 894 of the boreholes have quality information.

All drilling is done using diamond core drilling techniques.  Boreholes were cored from below the zone of soft weathering to Pre-Karoo basement.  A standard, TNW-sized core was utilized in all drilling campaigns.  Core recoveries were high ranging between 98% and 100%.

Most boreholes were drilled down to basement through the full sequence of seams present in the area.  A complication that arises from drilling in areas overlain by previous underground workings is the difficulty of drilling through loose debris in the old workings. Boreholes are planned out above the expected pillar centre location to minimize the risk, but when a hole-in does occur and drillers are unable to continue with the hole, the lower seam samples cannot be obtained.   The table below gives a breakdown of the number of intersections per coal seam.

**Table 5-1:  Number of Intersections per Coal Seam**

| Seam | Number of Intersections |
|---|---|
| No. 5 Seam | 583 |
| No. 4 Seam | 979 |
| No. 2 Seam | 854 |
| No. 1 Seam | 662 |

### 5.1.2  Seam geology

### 5.1.2.1 No.1 seam

The best developed, No. 1 Seam was intersected in the north-western corner of the Tweefontein Complex area.  The thickness varies between 0.0 m and 2.0 m with an average of 1.1 m.  The table below gives a summary of the qualities and thicknesses.

**Table 5-2:  No. 1 Seam data**

| No 1 Seam | Depth M | Thickness M | RD | Moisture % | Ash % | Volatiles % | FC % | Sulphur % | CV MJ/kg |
|---|---|---|---|---|---|---|---|---|---|
| Min | 3.0 | 0.0 | 1.26 | 0.83 | 11.0 | 7.6 | 24.7 | 0.18 | 9.6 |
| Max | 147.3 | 2.0 | 2.09 | 4.51 | 62.7 | 37.9 | 66.5 | 4.49 | 29.9 |
| Average | 66.3 | 1.1 | 1.56 | 2.26 | 25.3 | 23.5 | 48.9 | 1.13 | 23.8 |

The separation between the No. 1 and No. 2 Seams varies between 1.0 m and 29 m and consists predominantly of sandstones and grits.

### 5.1.2.2 No.2 seam

The No. 2 Seam is the most extensively developed seam underlying the Tweefontein Complex area.  The No. 2 Seam varies in thickness between 0.1 m and 13.5 m with an average of 4.8 m.  The No.2 Seam sub-outcrops around the Pre-Karoo highs where it is a mere 10 m below surface.  At its deepest point it is 139 m below surface with an average depth to the floor of 60.2 m.

The No. 2 Seam group consists of a lower zone (No. 2 Select Seam) which consists of a basal, vitrain-rich, bright band overlain by lustrous to dull lustrous coal with thin bright bands.  The No. 2 Top Seam consists mainly of dull coal with carbonaceous shale or mudstone zones.
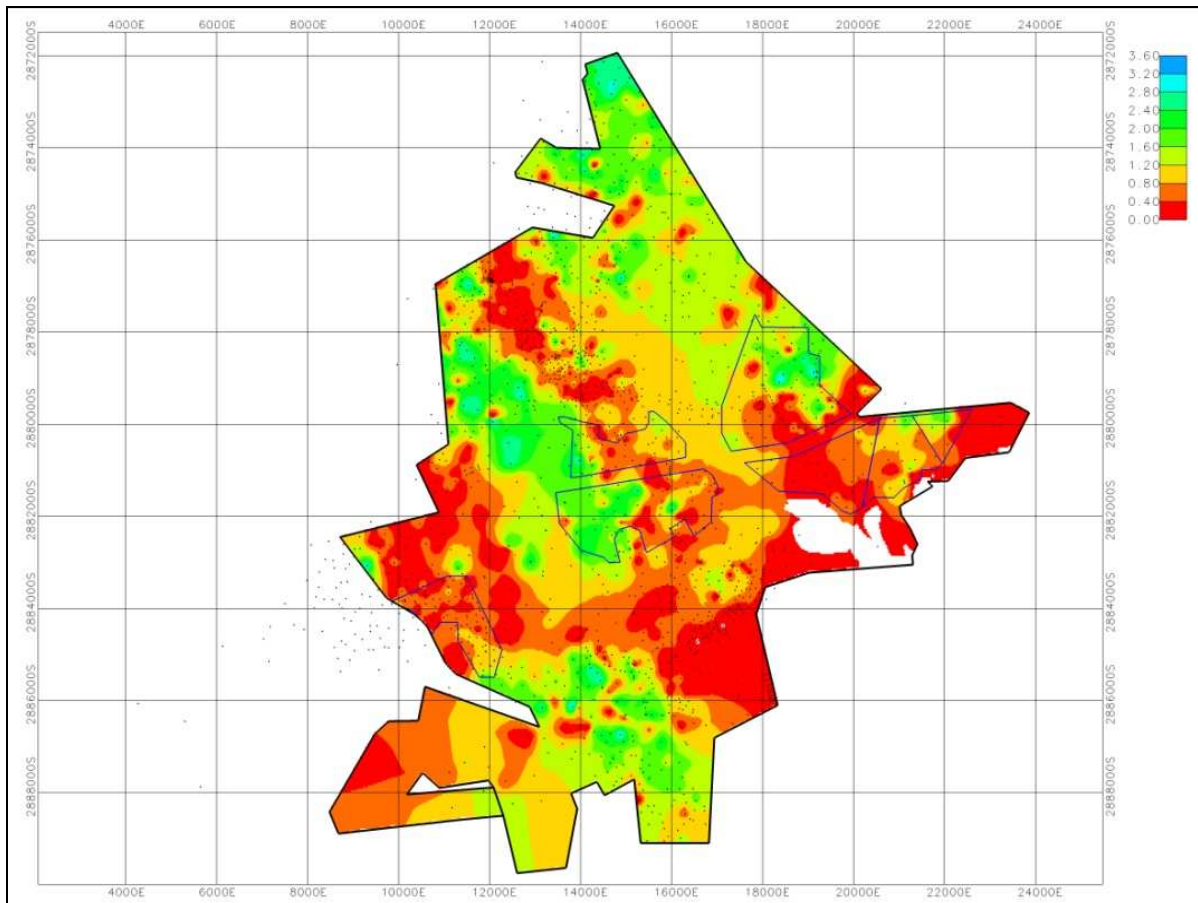
**Figure 5.1  The No. 1 seam thickness contours across the Tweefontein permit area, with the open pit boundaries indicating the specific evaluation areas.**

The No. 2 Select Seam horizon is highly variable within the 2-seam group and the thickness varies between 0.1 m and 8 m with an average of 2.9m.   The quality is better than that of the No. 2 Top Seam with an average raw CV of 22.3 MJ/kg.

**Table 5-3:  No. 2S Seam data**

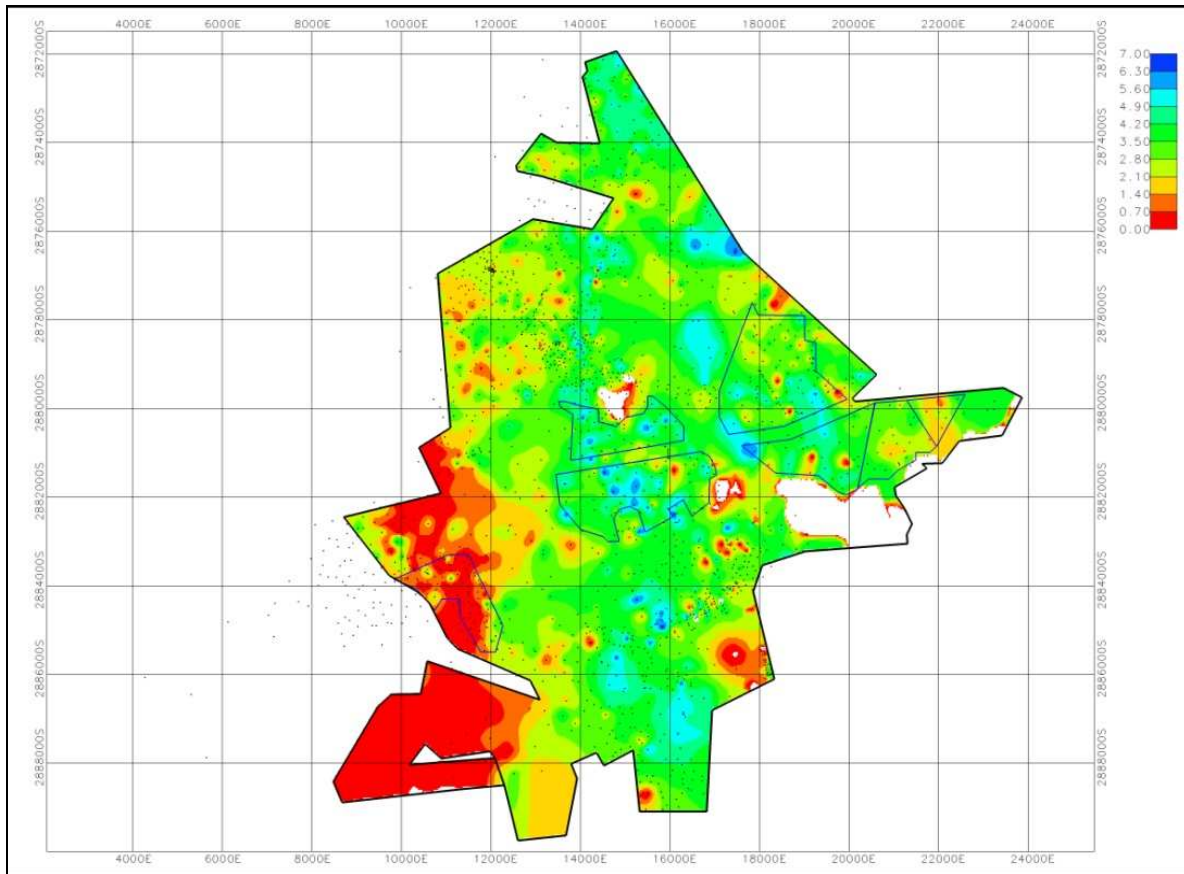| No 2S Seam | Depth M | Thickness M | RD | Moisture % | Ash % | Volatiles % | FC % | Sulphur % | CV MJ/kg |
|---|---|---|---|---|---|---|---|---|---|
| Min | 2.0 | 0.0 | 1.3 | 0.82 | 12.3 | 2.48 | 14.6 | 0.03 | 3.13 |
| Max | 139.0 | 8.0 | 2.2 | 7.42 | 77.0 | 32.21 | 66.0 | 4.40 | 28.40 |
| Average | 60.2 | 2.9 | 1.6 | 2.66 | 27.1 | 21.94 | 48.5 | 1.13 | 22.32 |

**Figure 5.2 The No. 2 select seam thickness contours across the Tweefontein permit area, with the open pit boundaries indicating the specific evaluation areas.**

The No. 2 Upper Seam consists mainly of dull coal with carbonaceous shale / mudstone zones and has an average CV of only 18.4 MJ/kg.

**Table 5-4: No. 2U Seam data**

| No 2T Seam | Depth M | Thickness M | RD | Moisture % | Ash % | Volatiles % | FC % | Sulphur % | CV MJ/kg |
|---|---|---|---|---|---|---|---|---|---|
| Min | 11.1 | 0.0 | 1.4 | 0.89 | 13.2 | 8.3 | 7.1 | 0.05 | 3.2 |
| Max | 129.0 | 9.3 | 2.2 | 5.30 | 77.1 | 38.3 | 84.1 | 5.17 | 28.2 |
| Average | 48.9 | 0.7 | 1.7 | 2.62 | 35.8 | 19.9 | 39.5 | 0.88 | 18.4 |

The No. 2 Seam is overlain by on average a 14m thick sequence consisting of a prominent 8 m to 10 m thick carbonaceous mudstone / siltstone, which grades upwards into a highly micaceous, bioturbated, sandstone. A thick interlaminated siltstone / sandstone, and a cross-bedded sandstone follow.
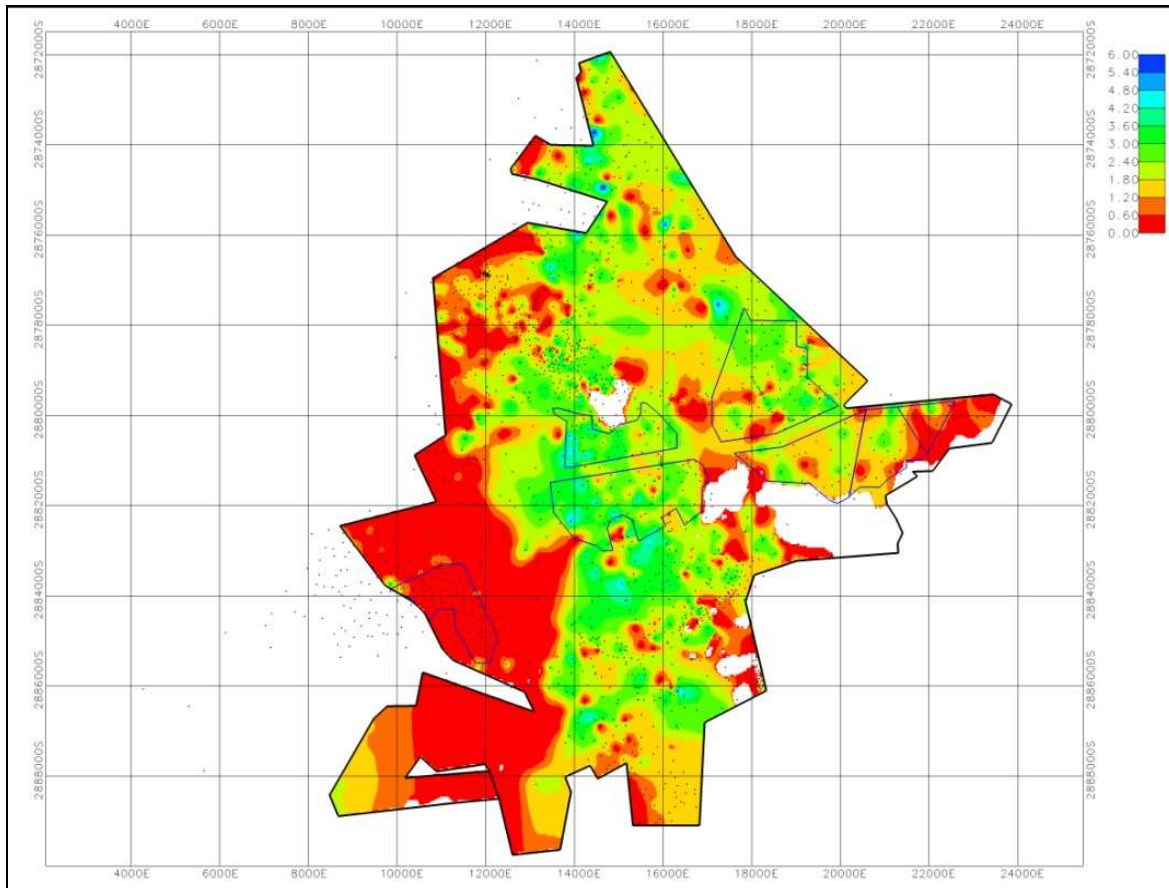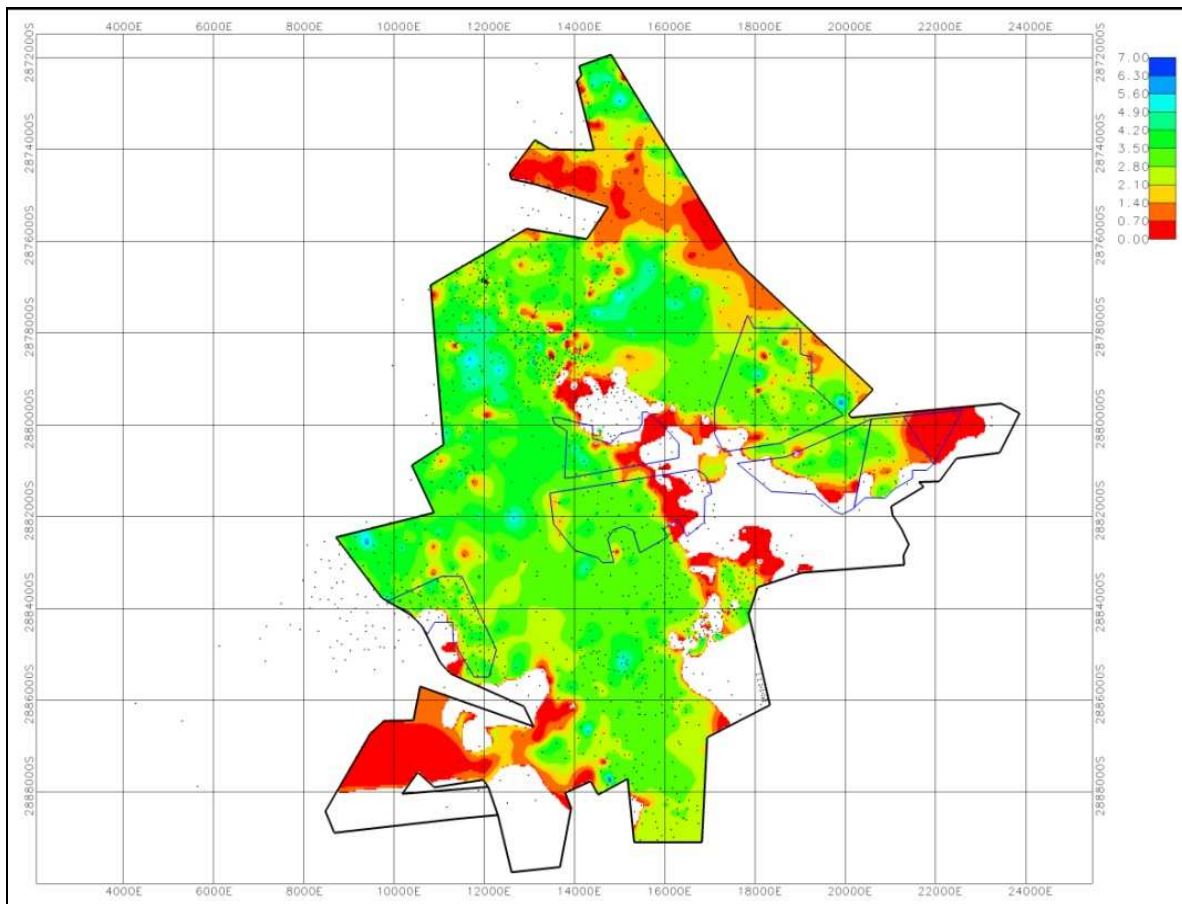
**Figure 5.3  The No. 2 upper seam thickness contours across the Tweefontein permit area, with the open pit boundaries indicating the specific evaluation areas.**

### 5.1.2.3 No.3 seam

The No. 3 seam is generally the only seam of no economic importance due to its thickness (0.2 to 0.5m) (Jeffery, 2005) and is not included in the investigation.

### 5.1.2.4 No.4 seam

The No. 4 Seam being closer to surface is more influenced by weathering and is subsequently not as extensively developed as the No. 2 Seam.  The No.4 Seam sub-outcrops around the Pre-Karoo.  At its deepest point it is 107 m below surface with an average depth to the floor of 47.6 m.  The No. 4 Seam varies in thickness between 0.5 m and 14.1 m with an average of 5.9 m.

The No. 4 Seam group consists of a lower zone (No. 4 Select Seam) which consists of a basal vitrain-rich bright zone.  The No. 4 Select Seam is overlain by a zone of dull and dull lustrous coal with thin bright bands and carbonaceous shale / mudstone partings called the No. 4 Top Seam.

The No. 4 Select Seam horizon is even more variable than the No.2 Select Seam horizon and the thickness varies between 0.1 m and 10.1 m with an average of 3.55 m.  The quality is better than that of the No. 4 Upper Seam with an average raw CV of 23.8 MJ/kg.

**Table 5-5:  No. 4S Seam**

| No 4S Seam | Depth M | Thickness M | RD | Moisture % | Ash % | Volatiles % | FC % | Sulphur % | CV MJ/kg |
|---|---|---|---|---|---|---|---|---|---|
| Min | 0.06 | 0.01 | 1.3 | 1.02 | 9.1 | 8.4 | 16.4 | 0.22 | 8.6 |
| Max | 107.6 | 10.17 | 2.0 | 7.00 | 62.4 | 54.7 | 63.7 | 3.69 | 28.8 |
| Average | 47.6 | 3.55 | 1.5 | 2.91 | 23.4 | 23.2 | 49.9 | 1.5 | 23.8 |



**Figure 5.4  The No. 4 select seam thickness contours across the Tweefontein permit area, with the open pit boundaries indicating the specific evaluation areas.**

The overlying No. 4 Upper varies between 0.1 m and 8.8 m, is generally of low quality with a CV of 19 MJ/kg and is split by several partings.

**Table 5.6:  No 4U Seam**

| No 4T Seam | Depth M | Thickness M | RD | Moisture % | Ash % | Volatiles % | FC % | Sulphur % | CV MJ/kg |
|---|---|---|---|---|---|---|---|---|---|
| Min | 6.4 | 0.1 | 1.3 | 0.98 | 17.0 | 8.9 | 21.1 | 0.26 | 7.0 |
| Max | 101.5 | 8.8 | 2.0 | 7.30 | 60.0 | 38.0 | 62.5 | 4.41 | 26.6 |
| Average | 39.1 | 0.98 | 1.7 | 2.52 | 35.2 | 21.6 | 41.2 | 1.32 | 19.3 |

The interburden between the No. 4 Seam and the No. 5 Seam averages 15.9 m and consists of a dark, carbonaceous mudstone with inter laminated fine grained siltstone.  This

is followed by a cross laminated sandstone, grading into a laminated sandstone, topped by a thin mudstone.
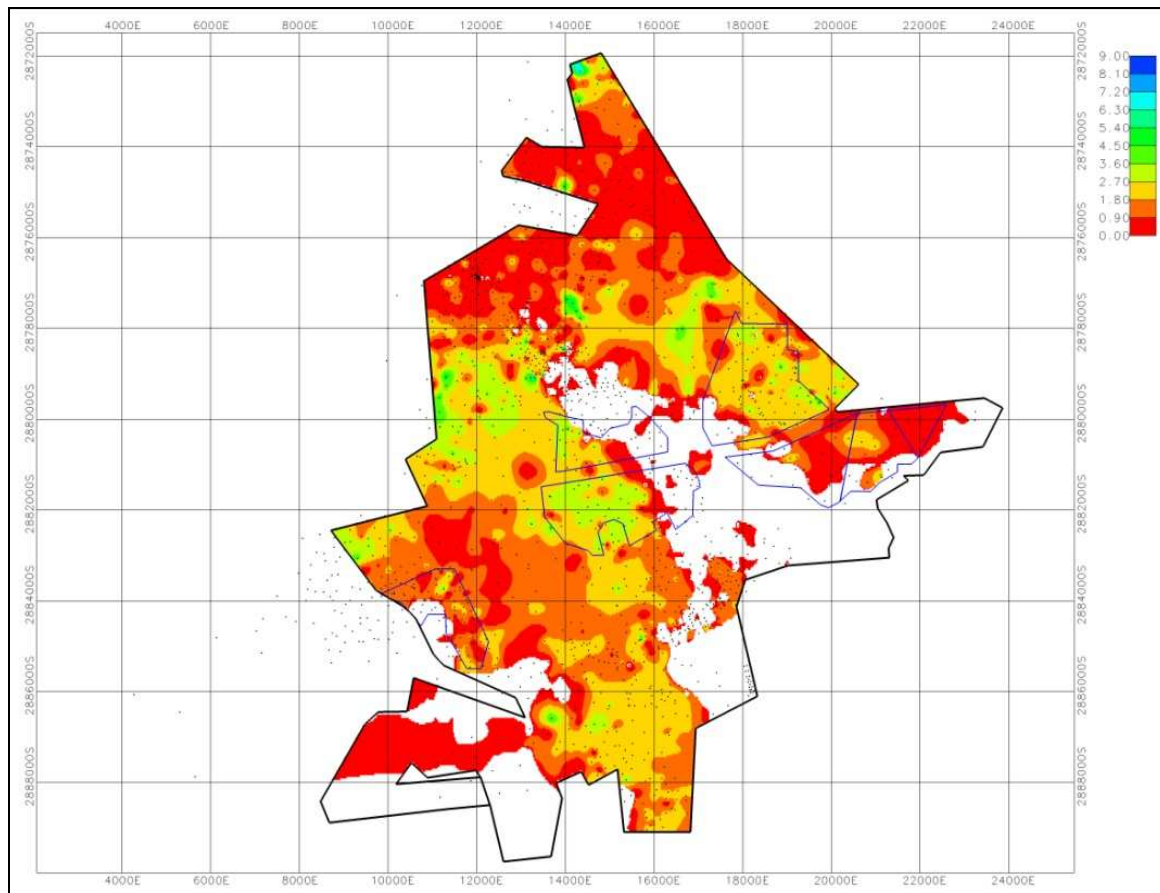


**Figure 5.5  The No. 4 upper seam thickness contours across the Tweefontein permit area, with the open pit boundaries indicating the specific evaluation areas.**

### 5.1.2.5 No.5 seam

The No. 5 Seam is preserved as erosional remnants on higher elevations.  It varies in thickness between 0.3 m and 5.8 m and is a bright, well-banded, vitrain-rich coal.  Raw CVs as high as 27 MJ/kg can be encountered, but the average of the seam is much lower at 20.5 MJ/kg.  In the north-east of the Tweefontein Complex area a 0.1 m to 0.4 m thick shaly coal/carbonaceous mudstone band occurs at the top of the seam.    A cm-scale mudstone band may also occur some 0.10 m to 0.25 m above the soft mudstone floor.   This thin mudstone band is locally known as the "false floor".

Table 5-7:  No. 5 Seam

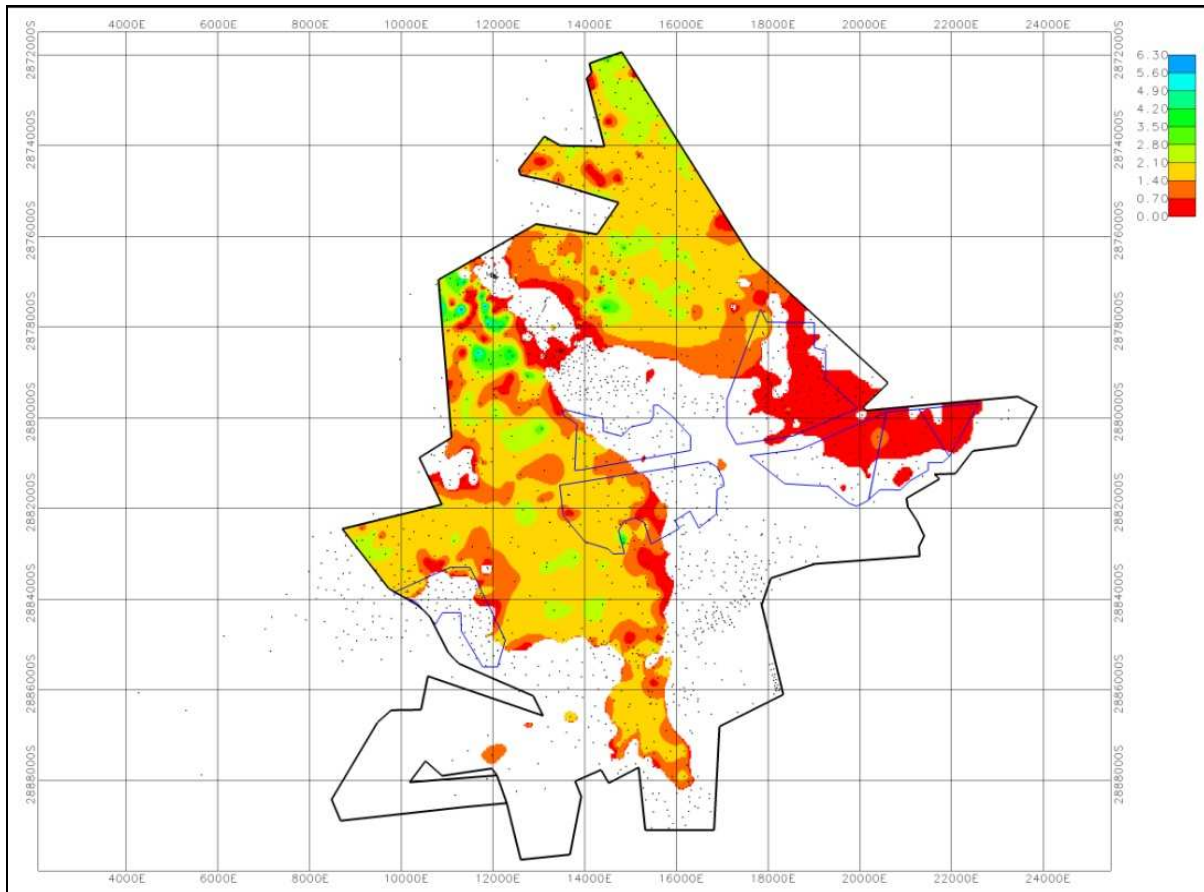| No 5 Seam | Depth M | Thickness M | RD | Moisture % | Ash % | Volatiles % | FC % | Sulphur % | CV MJ/kg |
|-----------|---------|-------------|-----|------------|-------|-------------|------|-----------|----------|
| Min | 0.4 | 0.3 | 1.4 | 0.53 | 15.6 | 8.31 | 6.2 | 0.25 | 3.8 |
| Max | 84.4 | 5.8 | 2.2 | 9.67 | 74.2 | 31.2 | 56.1 | 4.99 | 26.7 |
| Average | 36.1 | 1.75 | 1.7 | 2.74 | 32.9 | 23.0 | 39.7 | 1.50 | 20.5 |

**Figure 5.6  The No. 5 seam thickness contours across the Tweefontein permit area, with the open pit boundaries indicating the specific evaluation areas.**

### 5.1.3  *Structure*

All seams terminate against a major Pre-Karoo high that exist in the centre of the Tweefontein Complex area.   The seams trend upwards approaching the pre-Karoo high causing steep dips around these areas.   This basement high plunges to the northwest resulting in the complete succession of seam horizons in the northern portions of the Tweefontein Complex.   Undulations in the Pre-Karoo floor influence the distribution of the coal seams, especially the No. 1 Seam, which thickens and is better preserved in deeper pockets. Due to differential compaction over Pre-Karoo floor ridges, deep fractures develop which allow the ingress of water.   The base of weathering (weathering depth) is far deeper over these areas which cause the localized removal of the seam by weathering. Small scale faulting may also be present in the vicinity of any Pre-Karoo highs due to the draping of the seams over these highs and the influence of differential compaction.   Cross sections illustrate the structure across the complex in Appendix A and B.
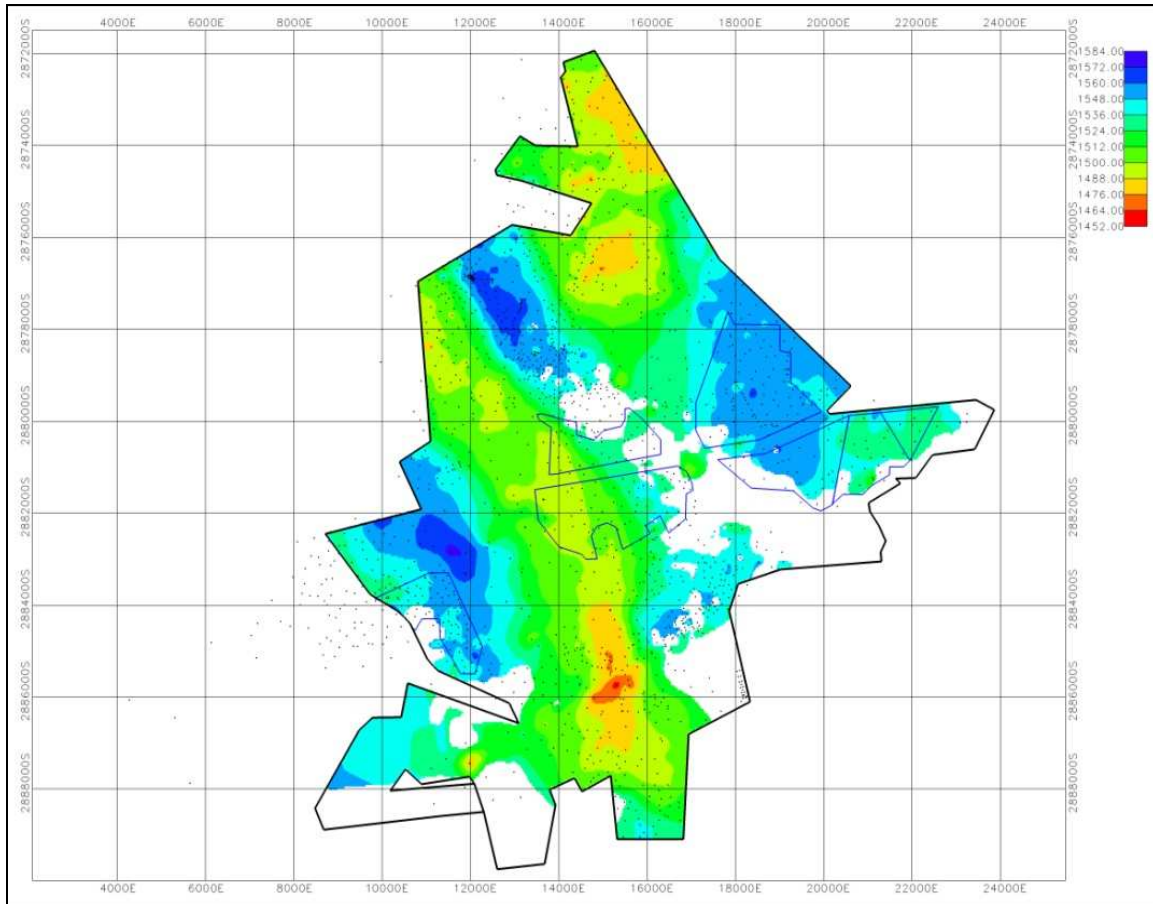
**Figure 5.7  The No. 4 select seam elevation contours across the Tweefontein permit area, with the open pit boundaries indicating the specific evaluation areas.  From this the seam outcrop against the palaeo-high is clearly visible.**

### 5.1.3.1 Intrusions

Post Karoo Dolerite Dykes are present throughout the deposit.  One prominent feature is the Ogies Dyke, a ~250m wide intrusive system striking in a west to east direction across the Witbank Coalfields with an associated burnt coal halo.  The Ogies dyke, which has an extensive strike length, splits Tweefontein roughly in half.  Along the southern boundary striking southwest to northeast is a 250 m wide graben displacing the seams by 20m.

### 5.1.3.2 Coal washability

Results from float-sink analyses were uploaded into a plant simulation application and the data were simulated for a 6000kCal primary export product with a 21.5 MJ/kg secondary Eskom product.  The results from the simulation software were re-entered into the Minex geological model and estimated across the study area.  The export yields would also be used in the saleable strip ratio calculation dictating the preliminary mine design.

## 5.2 Layout design

There are numerous factors that need to be considered when deciding on the basic pit boundaries within a resource area. As these studies have already been completed as part of the life-of-mine process of Tweefontein complex, the basic pit boundaries will be used. Factors varying from environmental impact to surface infrastructure were considered to define the following five pit outlines –
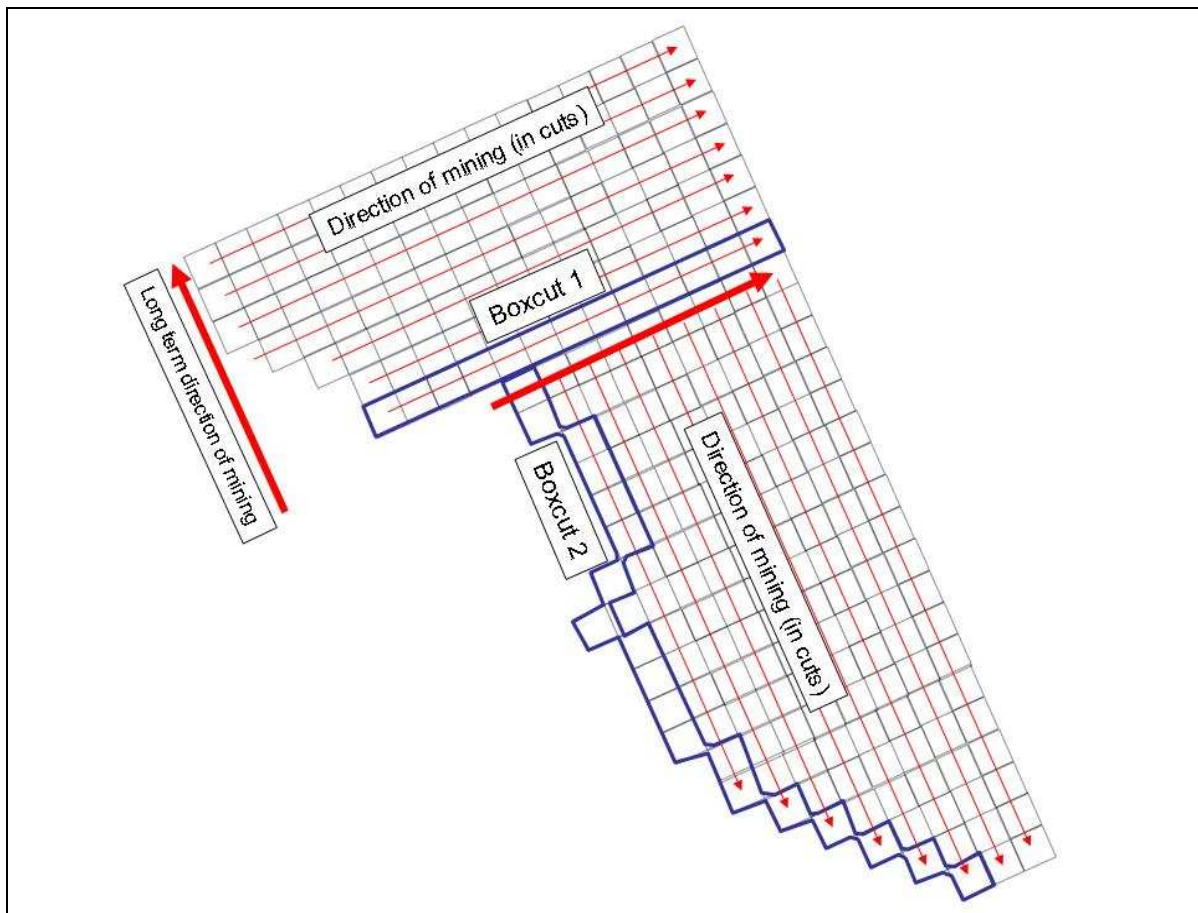
- Boschmans North of Dyke
- Boschmans South of Dyke
- Makoupan
- Klipplaats
- Zaaiwater



**Figure 5.8 The Zaaiwater pit design indicating the boxcut areas with the mining direction, as well as the advance direction of the cuts.**

## 5.3 Mine scheduling

The mine schedule, i.e. the sequence in which the mining activities take place, is broadly based on the saleable strip ratio as this will allow low cost mining during the start-up phases of the operation, thereby maximizing profitability and the rate at which the capital is

paid back.  Another advantage of this approach is that should commodity prices outgrow the cost of mining as the demand for energy increases and the easily accessible global resources decline, the life of the mine can easily be extended into previously un-profitable areas by expanding the end-wall.

A drawback of this approach is that the volume of the overburden spoiled from the current operating cut will always exceed the volume available from the previous void as the coal seams deepen.  This can be corrected with additional rehabilitation efforts, but will be more costly than mining from deeper areas towards the sub-outcrop.

Another major consideration that was taken into account is pit length.  If the cuts are too short, the phased nature of the mining operation will result in certain processes not being able to function until all the activities in that area have been completed.  An example of this would be if the overburden removal operation needs to spoil the waste from the current block into the previous cut, but the coal has not yet been removed to create the void, then the overburden operation is delayed.  In this instance the waste removal activity is said to be coal-bound.  This will reduce the efficiency of the mine plan greatly.  As these operations do not consider draglines for waste removal a pit length of only 500meters is required to ensure proper unhindered activities.
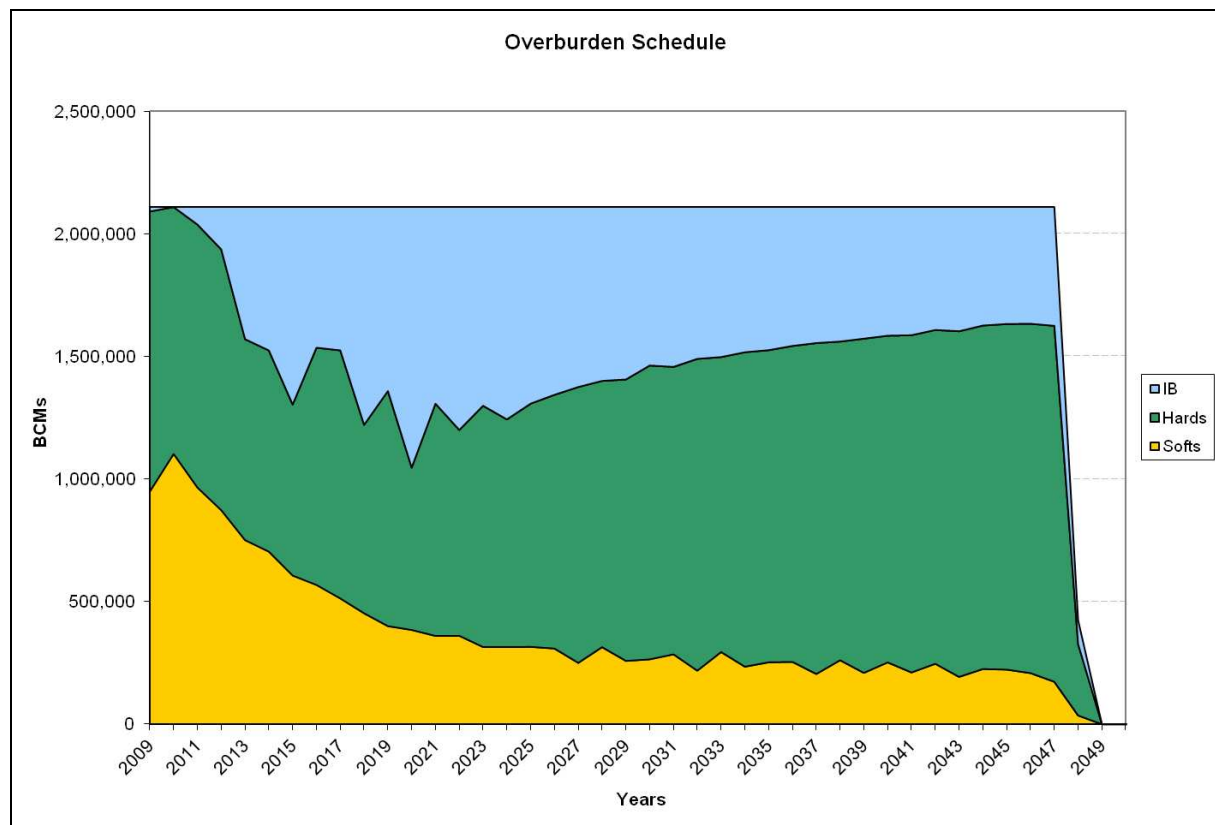


**Figure 5.9  Graph indicating the total overburden production schedule of the Boschmans North of Dyke pit.  The constant total waste indicates that the process has succeeded by mining the soft, hard and inter-burden independently whilst achieving the total waste target as set out by the scenario parameters.**
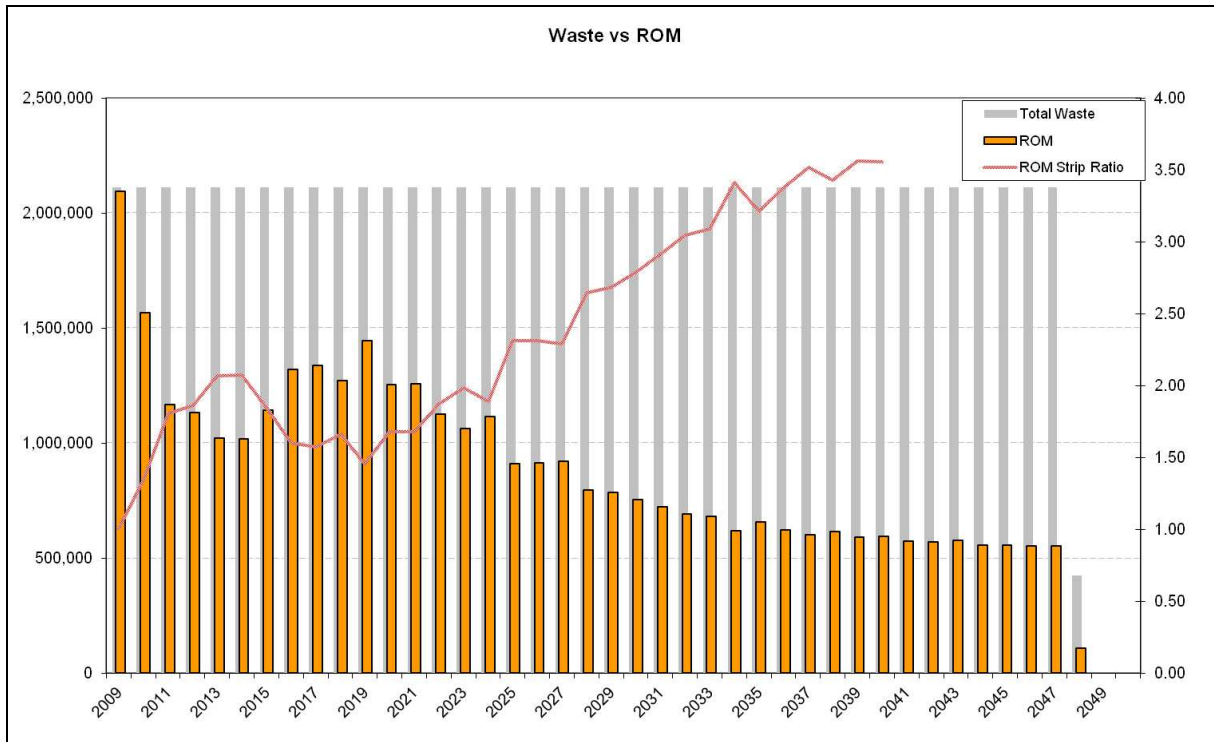
**Figure 5.10 Graph indicating the total overburden production schedule of the Boschmans North of Dyke pit versus the total run-of-mine tonnages (ROM). Plotted on the graph is thus the ROM strip ratio, confirming that the pit layout design achieved the criteria of mining from a low strip ratio to a high strip ratio over the life of the mine.**
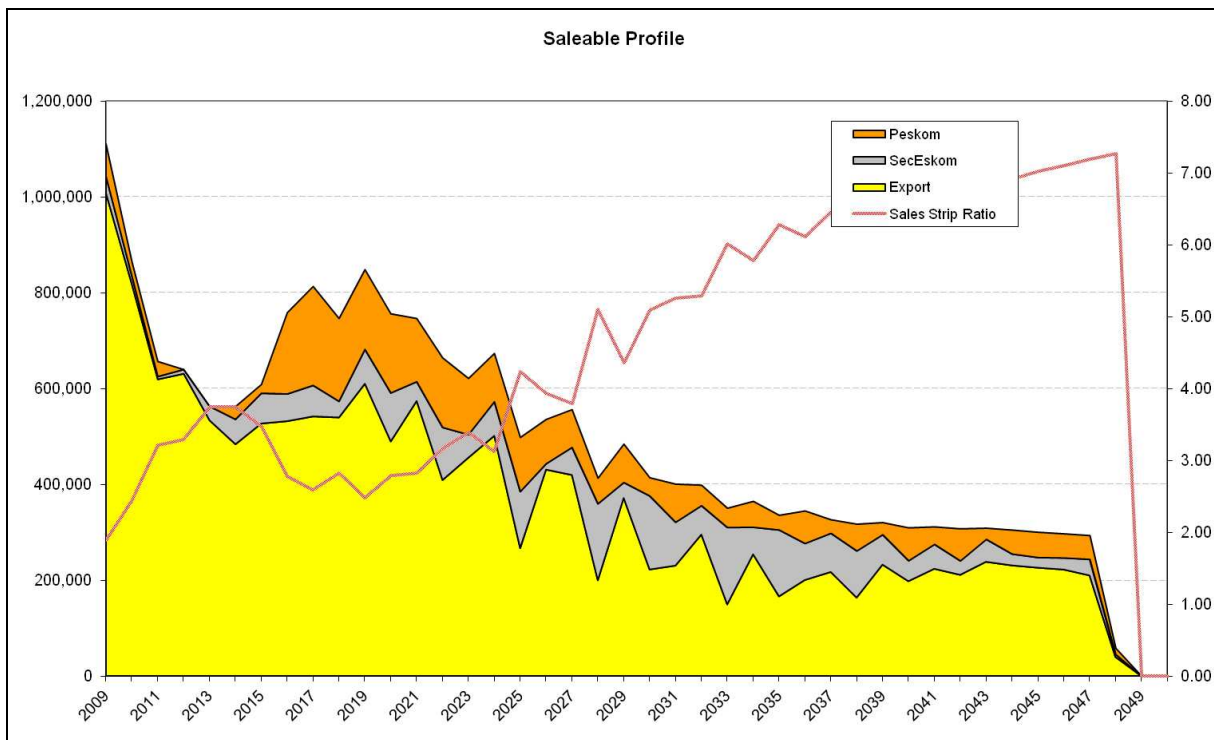


**Figure 5.11 Graph indicating the total product production schedule of the Boschmans North of Dyke pit. All the possible products are plotted, from Export, primary Eskom to secondary Eskom. Also indicated on the graph is thus the saleable strip ratio, also confirming that the pit layout design achieved the criteria of mining from a low strip ratio to a high strip ratio over the life of the mine.**

The biggest challenge of the study was to create a system that would allow mining operations to be simulated continuously whilst changing the mining parameters and seeing the impact on the outcome in real-time. For this a database was created for each operation within the Tweefontein complex. The database contained all the parameters required to create a detailed mine plan within each single mining block. As the operation is waste driven the total waste in each mining block was inserted into a matrix. The Y-axis of the matrix contained each mining block with its corresponding total waste in cubic meters. The X-axis contained the time periods, in this instance years, as well as the target waste removal for that scenario as a variable that can be altered for evaluation purposes. A complex decision based algorithm then allocates the blocks to be mined each year and even calculates how much of a certain block should be carried across to the following year should the required waste be met for that year. This algorithm is a key cornerstone of the project as it allows numerous scenarios to be repeated with small changes in the parameters testing the effect of those changes on the final outcome.

From the blocks that are simulated by the algorithm, the coal tonnages can be added creating a ROM profile and totals for each year. By multiplying the ROM tonnages with the corresponding yield, the saleable profile and totals can also be calculated. By adding the totals for each year a comprehensive production schedule is derived which forms the basic input into the financial model. The overburden is also divided into soft and hard overburden, and scheduled separately before being added in the final production schedule.

## 5.4 Financial Model

The aim of the financial model is to determine the value of each mining scenario by applying costs to each facet of the operation and calculating the revenue gained from the product coal. These values can be adjusting for risk and compared to determine an optimal solution. The volumes and tonnages produced by each mine schedule were used to calculate the costs and revenue from the operation in the following manner:

### 5.4.1 Topsoil removal

As a standard practice, one meter of topsoil is pre-stripped before the soft overburden is removed. The disturbed area (multiplied by 1m) was then multiplied with the current price for the activity and thus the costs were derived over the life of the mine.

### 5.4.2 Pre-strip removal of soft overburden

The soft overburden volume is multiplied with the cost to derive the total cost per annum for soft removal.

### 5.4.3 Hard overburden drilling

As this is a cost per meter drilled, a basic 8m x 9m staggered drilling pattern was assumed, used to calculate the total meters to be drilled for each cubic meter of waste.

### 5.4.4 Hard overburden blast

The hard overburden is then blasted and the cost per cubic meter is calculated and added for the life of the mine.

### 5.4.5 Bull dozer pushover of two thirds of the blasted waste

An assumption was made that two thirds of the blasted hard overburden can be dozed into the void, thus two thirds of the cubic meters of hard overburden are multiplied with the dozer pushover cost.

### 5.4.6 Truck and shovel removal of blasted waste

The remaining one third of the hard overburden is multiplied with the truck and shovel removal cost.

### 5.4.7 Parting drill, blast and removal

All interburden is costed slightly differently, thus the interburden drill, blast and removal cost is calculated separately from the overburden cost.

### 5.4.8 Coal drill, blast and removal

Coal drlling, blasting and removal are calculated on the Rand per ROM ton basis and thus the ROM tonnages can be used to calculate the total cost over the life of mine.

### 5.4.9 Pit services

Additional pit service costs are added which will account for small activities in the pit as a basic function of the ROM tonnages produced in that year.

### 5.4.10 Coal transport costs to wash plants

Various beneficiation options can be considered as well as various transport options, but for the sake of consistency all transport distances are kept constant at 1km for all operations.  The costs are thus calculated on the ROM tonnages.

### 5.4.11 Washing cost

The ROM tonnages are fed directly into the plant and thus the cost of beneficiation is calculated based on the ROM tonnages.

### 5.4.12 Product handling and discard handling

Product and discard tonnages are multiplied with the costs and added to produce the costs over the life of the mine.

### 5.4.13 Rehabilitation cost

The area disturbed as calculated from the mine schedule is multiplied with the rehabilitation costs.

### 5.4.14 Labour costs

A general labour profile is utilized and which increased the labour as the project commences. The labour figures are not adapted as a function of the mining rate as the equipment capacities are assumed to merely increase not requiring additional personnel, which might not be the case, but without extensive equipment utilisation study a more accurate approach is not available.

### 5.4.15 General overheads

General overheads are added to the operation, perhaps not impacting on the comparative nature of the exercise, but ensuring it is done as completely as possible.

### 5.4.16 Capital

In order to calculate the start-up capital required for each operation there are two approaches that can be taken. First is a complete equipment investigation where the capacities of the equipment need to be studied as the mining rates are adjusted, for constraints in the system whilst matching the equipment in order not to over-capitalize the project. This type of study will also require a full resource based mine plan where the activities for each equipment piece are simulated in conjunction with the entire fleet to determine utilisation. The actual cost of each piece of equipment can then be added to the capital cost as the mining rate is varied. This method is extremely tedious and will require extensive study outside the basic scope of this project. Also as this mine schedule is based on a cookie cut principal the equipment activities cannot be tracked to determine true capacities. The second approach is a generalised approach where the most likely set of equipment is used to determine mining rate as well as the resulting capital required. To

investigate the impact of changing the mining rate on the value of the operation, multiples of these equipment fleets are then used.  This method is based on fairly accurate assumptions whilst allowing for uncomplicated simulation iterations.

The various capacities of the equipment were considered when determining the numbers required for a full operational fleet, but as the waste removal is the driver of the mining operation, the capacity of the waste removal fleet is used in the mine rate calculation. The following list of equipment was used:

**Table 5-8: List of the equipment used in the production schedule**

| | |
|---|---|
| PitViper | Overburden Drill |
| EX5500 | Hydraulic face shovel |
| EX2500 | Coal escavator |
| CAT789 | 200t Burden removel dumpers |
| CATD11 | Pushover dozer |
| CATD10 | Stockpile management |
| DM30 | In-pit drill - drilling parting and coal |
| CAT992K | Front -end loader |
| CAT785 | Rear Dumper - 140t for coaling |
| CAT834B | Rubber tyre dozer |
| CAT16G | Grader |
| CAT777C | Water tankers |
| CAT769 | Diesel Bowser |
| CAT988 | Cable reeler |
| CAT980 | Tyre handlers |

### 5.4.17  Cash flow

The costs and revenue were input into a basic income statement to determine the cash generated each year.  Direct mining, labour overheads and rehabilitation costs are deducted from the revenue to determine the earnings before interest and tax (EBIT).  The taxable income is determined by capitalizing the start-up equipment costs immediately. The company tax is then calculated as 28% of the taxable income respectively, with the formula stated in the Royalty Act (DMR, 2008) to determine the royalty percentage.  Cash is then calculated by subtracting the tax and royalties from the EBIT with the initial capital investment as a negative value in year zero.

Other approaches can be considered to optimize the cash flow by adjusting the amortization rate, but the decision was made to keep that policy simple and constant for this exercise to ensure true comparison between the operations.

From the cash flow statement the NPV and IRR can easily be calculated.  As NPV considers the cost of capital specific to the company, it will serve as a better indicator of value than IRR.  Generally in the industry NPV is preferred as a better measure of value

compared to IRR (Investopedia, 2011). A similar cash flow model was used as described by Pincock (2005).

## 5.5    Mine rate fluctuation

By altering the mining rate in the system the production schedule adjusts accordingly, impacting on the cash flow and the resulting NPV. The faster the mining rate, the more equipment is required, the higher the capital cost. As the mining rate increases the rate of income is higher allowing for re-investment of the funds, but this value is offset at a specific point by the increase in capital. By completing many iterations using multiple mining rates this point of maximum value can be determined. This graph for the Boschmans North of Dyke is illustrated by figure A.



**NPV for each mining rate**
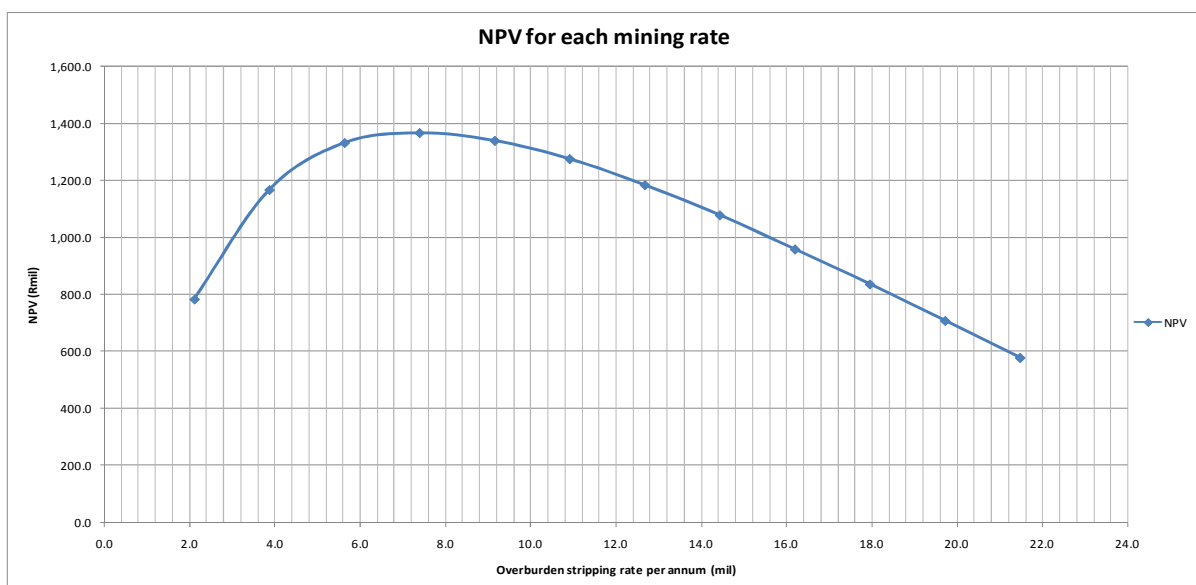
**Figure 5.12 Graph indicating the NPV values at various mining rates. A maximum point is clearly visible at 7.4 million cubic meters per annum.**

From this it can easily be seen that the maximum value (NPV) is gained by mining at 7.4 million cubic meters per annum, giving an NPV of R1.36 billion. The actual NPV value is not as much of importance as the relative number that is used to derive the optimal mining rate.

## 5.6    Price and cost fluctuations

The first step in considering the risk at each mining rate is to complete a sensitivity analysis. A basic sensitivity analysis producing a 'spider diagram', such as described by Pincock (2005), will have little use in this type of application as a method is needed to modify the optimal mining rate by considering the sensitivity of the system. Thus the sensitivity analysis is based on the following requirements:

- Price and cost will be fluctuated independently in order to assess the individual impact of these two parameters

- Uncertainty regarding the parameters increases with time thus price and costs will be escalated per annum by certain amounts rather than applying a constant increase value over the life of the mine.

Price and cost escalation is varied from -10%, -5%, +5% and +10% and the resulting NPV is recorded. This gives a range of NPVs for each mining rate and the distribution of these NPVs will give an indication of the sensitivity of the NPV to the parameters at that specific mining rate. These values for Boschmans North of Dyke are compiled in table 5.9.

## 5.7    Risk adjustment factors

The impact of the cost and price fluctuations needs to be utilized in a manner to modify the optimized mining rate to reflect the impact of risk. Calculations (1) and (2) were used to determine the risk adjustment factors. Thus for the mining rate of 2.1 million BCMs, the risk adjustment factors would be the following –

Price (1) –
*Standard deviavation (NPVi) / NPVbase = Xsensitivity*
Stdev(-107, 248, 783, 1 735, 3 773) / 783 = 1.99

Price (2)
*NPVbase / average (NPVi) = Xskew*
783 / ((-107 + 248 + 1 735 + 3 773)/4) = 0.55

Cost (1)
*Standard deviavation (NPVi) / NPVbase = Xsensitivity*
Stdev(1 116, 993, 783, 325, -950) / 783 = 1.07

Cost (2)
*NPVbase / average (NPVi) = Xskew*
783 / ((1116 + 993 + 325 – 950)/4) = 2.11

The average Cost risk rating is 1.59 with the average price risk rating 1.27. The average total risk rating for this specific mining rate is 1.43.

The basic NPV (prior to any fluctuations applied) is divided by this risk factor to adjust the NPV relative to the NPV at other mining rates. The exercise is completed for all mining rates and the NPVs are normalized to allow for easy comparison to the un-adjusted NPV profile. This can be done as the adjustment factors impact on the NPV relative to other mining rates and the absolute value of the NPV points have become meaningless. The price and cost factors for the various mining rates can be seen in table 5.9. The colour spectrum

applied purely indicates where in the table the highest (blue) and the lowest values (red) are obtained. Much can be interpreted from this table. Should the coal price rapidly increase over time, the model indicates that it becomes more valuable to mine at slower mining rates as to benefit from these higher prices. Should coal prices decrease over time it seems it remains the safest option to mine at close to the optimal mining rate, perhaps somewhat faster. Similarly it can be noted that with a significant growth in the mining cost it becomes more profitable to mine at much faster mining rates and again close to optimal should costs decrease over time. From the table it is also clear that the value of the resource is much more sensitive to fluctuations in the price of coal than the costs. This can give justification for price hedging when the company commences the process of contract negotiation (Kernot, 1970).
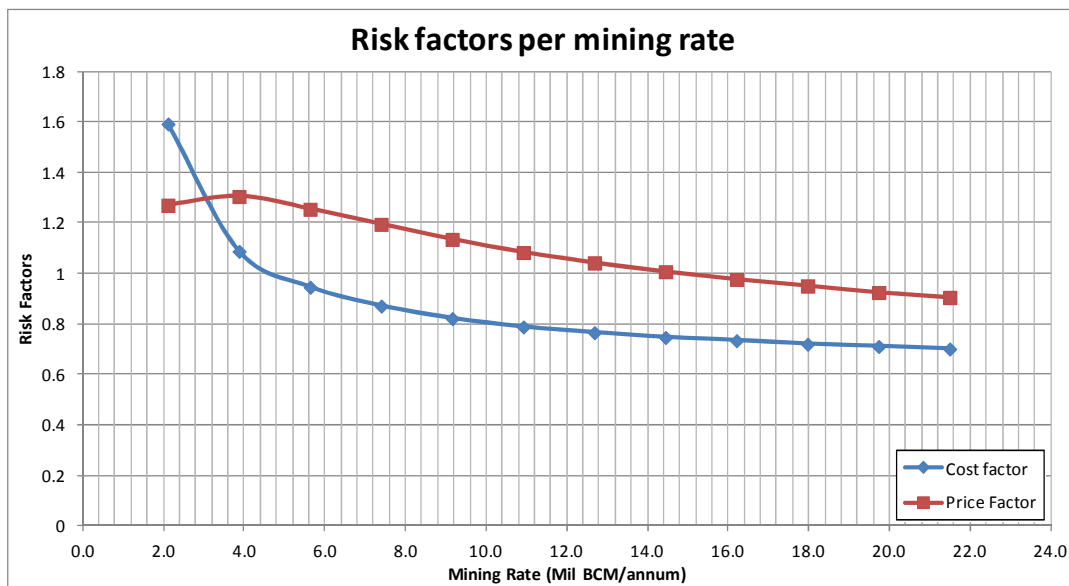


**Figure 5.13 Graph indicating the risk adjustment factors for the Boschmans North of Dyke pit for various mining rates. It can be seen that both the price and cost risk reduces for faster mining rates.**

By adjusting the NPV with the risk factors a new adjusted profile can be seen in figure 5.14a.
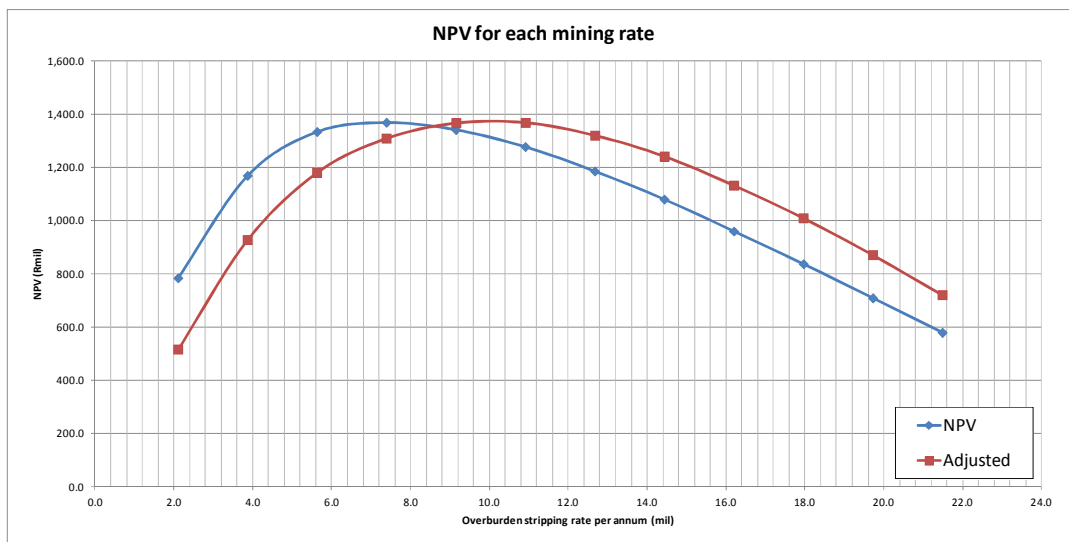
**Figure 5.14a Graph indicating the original NPV values for the specific mining rates and the adjusted NPV for the specific mining rates after the risk factors have been applied.**

**Table 5.9** The table indicates the NPV ranges achieved when applying the price and cost fluctuations for each of the mining rates.

| Waste Removal Rate (Mil BCM) | Costs | | | | | Price | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | -10% | -5% | 0% | 5% | 10% | -10% | -5% | 0% | 5% | 10% |
| 2.1 | 1,116 | 993 | 783 | 325 | -950 | -107 | 248 | 783 | 1,735 | 3,773 |
| 3.9 | 1,603 | 1,429 | 1,167 | 730 | -135 | 27 | 509 | 1,167 | 2,154 | 3,734 |
| 5.6 | 1,796 | 1,600 | 1,332 | 955 | 316 | 142 | 673 | 1,332 | 2,232 | 3,490 |
| 7.4 | 1,830 | 1,628 | 1,366 | 1,025 | 539 | 206 | 732 | 1,366 | 2,177 | 3,224 |
| 9.2 | 1,791 | 1,589 | 1,339 | 1,030 | 641 | 239 | 741 | 1,339 | 2,069 | 2,962 |
| 10.9 | 1,706 | 1,509 | 1,275 | 996 | 662 | 235 | 714 | 1,275 | 1,937 | 2,719 |
| 12.7 | 1,596 | 1,405 | 1,183 | 926 | 627 | 196 | 655 | 1,183 | 1,791 | 2,490 |
| 14.4 | 1,472 | 1,288 | 1,078 | 839 | 568 | 135 | 578 | 1,078 | 1,641 | 2,277 |
| 16.2 | 1,338 | 1,159 | 958 | 733 | 478 | 56 | 485 | 958 | 1,483 | 2,066 |
| 18.0 | 1,199 | 1,026 | 835 | 624 | 391 | -29 | 383 | 835 | 1,332 | 1,876 |
| 19.7 | 1,058 | 890 | 707 | 507 | 288 | -120 | 277 | 707 | 1,174 | 1,679 |
| 21.5 | 914 | 752 | 577 | 388 | 184 | -225 | 163 | 577 | 1,023 | 1,501 |

NPV (R mil)

Low                                                                                           High

From this it can be seen that the risk adjustment indicates a new optimal point at 10.9 mil BCMs/annum relative to the originally estimated 7.4 mil BCMs/annum. This does not give a clear indication of the actual mining rate to be used, but merely a range in which the highest probability of maximum value exists. These ranges will now become useful when combining the schedules for the entire complex.

Rules of thumb exist in the industry for evaluating the performance of a life-of-mine plan. Such as Taylor's Law (Taylor, 1986) which states the estimated ROM tonnes per day can be calculated in the following manner –

$$t/d = 0.014 * (Reserves)^{0.75} \text{ …………………………………….(3)}$$

This rule is not focussed on waste removal but can still be compared to the findings of the thesis by using Boschmans North of Dyke as an example, where the reserves equate to 35.8 million tonnes. By utilising Taylor's Law the following rates are calculated –

$$t/d = 0.014 * (35.8 \text{ million})^{0.75}$$

t/d = 6488 tonnes per day

t/a = 6488 * 365 days = 2.37 million tonnes per annum

The 2.73 million tonnes per annum calculated from Taylor's Law compares reasonably with the average ROM mining rate of 2.98 million tonnes per annum calculated in this paper. Other rules of thumb exist for the economic characteristics of the project, such as that the cashflow must be sufficient to repay the capital twice (Smith, 1997). By comparing the total cashflow to two times the capital, and plotting them on the same curves as seen in Figure 5.14a it is clear from figure 5.14b that the optimised mining rate suggested in this paper is well within the acceptable ranges.

## 5.8 Company scheduling

The study thus far focussed on the evaluation of the individual resource areas in isolation, but many companies will have various of these isolated resource areas and the estimated mine schedules, at optimal rates, can be utilized to determine the companies tonnage profiles when combined. The range between the original optimal mining rate and the risk adjusted mining rate gives opportunity for adjustment of the combined schedule to meet specific short to medium term requirements. The requirements can include plant feed capacity, rail allocation and stockpile capacity amongst others.
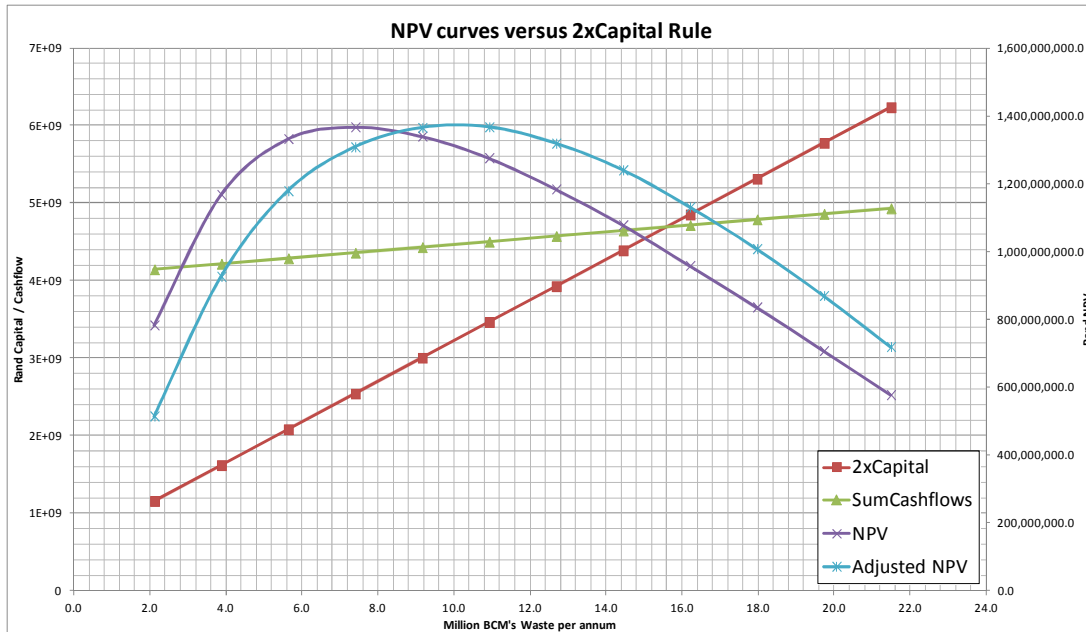
**Figure 5.14b Graph indicating the original NPV and adjusted NPV now on the secondary axis compared to the 2-times-capital rules (Smith, 1997). It is clear that the sum of the cashflows become less than two times the capital at approximately 16 million BCM's per annum, much higher than the suggested 7.4 and 10.9 million BCM's per annum.**

In this study, the total operation has a rail allocation of 4.1 million tonnes per annum of high grade export coal and the beneficiation plants are capable of feeding 10 million tonnes per annum. The schedules are paired in various combinations and the constraints are evaluated. This process is then repeated until the most constant total profiles are acquired. The following graphs show the waste removal, ROM production, export and Eskom grade product tonnage profiles for the entire company.

Table 5.10 shows the optimal rate, risk-adjusted rate and final utilised rate for each of the operations.

**Table 5.10 The table indicates the optimal, risk adjusted and actual utilised mining rate (as per the combined mining production schedule).**

| Operation | Optimal Base Rate | Risk Adjusted Rate | Utilised Rate |
|---|---|---|---|
| Zaaiwater | 4,646,400 | 5,808,000 | 6,969,600 |
| Makoupan | 16,281,760 | 18,934,080 | 16,281,760 |
| Klipplaats | 12,100,000 | 16,940,000 | 9,680,000 |
| Boschmans NOD | 7,395,520 | 9,157,280 | 7,395,520 |
| Boschmans SOD | 7,986,000 | 4,356,000 | 15,972,000 |

The optimal range for each of the deposits can also be compared graphically to the utilised rate as seen in figure 5.15. A 1:1 line is plotted on the graph where the area above this graph indicates possible over-utilisation (the mining rate is too high) whereas the area below the line indicates possible under-utilisation (the mining rate is too low). In order to complete the combined schedule, it is clear that the Klipplaats pit is under-utilised. This was done to maintain a constant 25 million BCM's per

annum later in the company's life, but with Boschmans SOD under-utilised in the same period, the rates can be adjusted closer to the optimal levels.
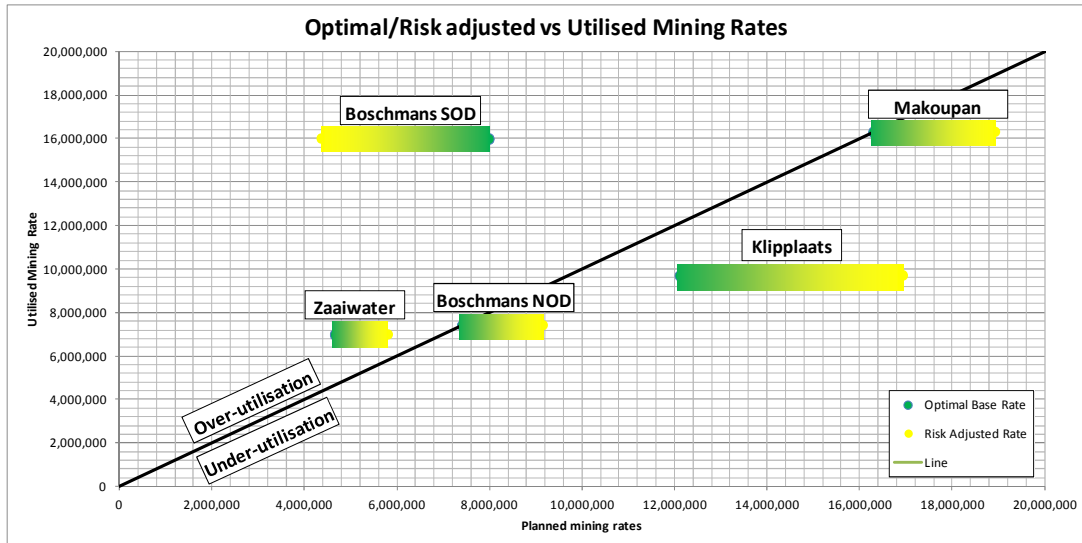


**Figure 5.15 Graph illustrating the range of suggested mining rates versus the utilised mining rate in the combined schedule.**

By increasing the mining rate of the Klipplaats pit to better align with the suggested optimal mining rate, a more profitable solution is gained for the company. With the initial planning Boschman's SOD was over-utilised to a large extent, but with the Klipplaats mining rate increased the Boschmans SOD rate can be slightly decreased whilst considering all the constraints and requirements of the system. Figure 5.16 illustrates adjustment made to the Klipplaats and Boschmans SOD pits.
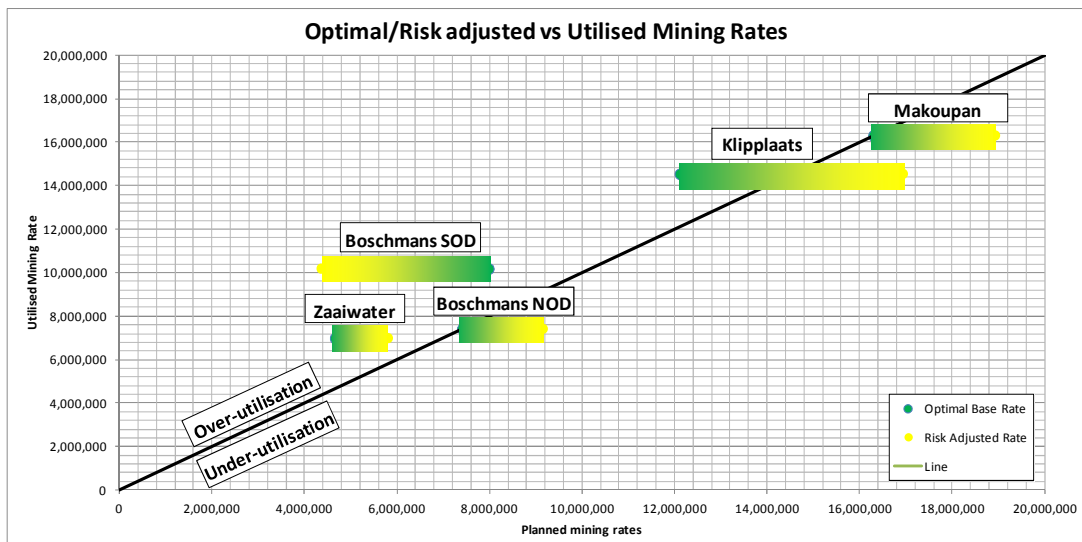


**Figure 5.16 Graph illustrating the changes to the utilised mining rate in the combined schedule.**

With the combined schedule now optimised the production schedules can be inspected in the following figures:

### 5.8.1   Waste Removal

By combining the schedules in this manner (Figure 5.16) the waste is kept relatively constant, with a slight increase towards the middle of the life of the mine. This configuration also ensures that as few operations run in parallel as possible, raising the opportunity for equipment to be carried across to new operations, thereby reducing the capital expenditure.
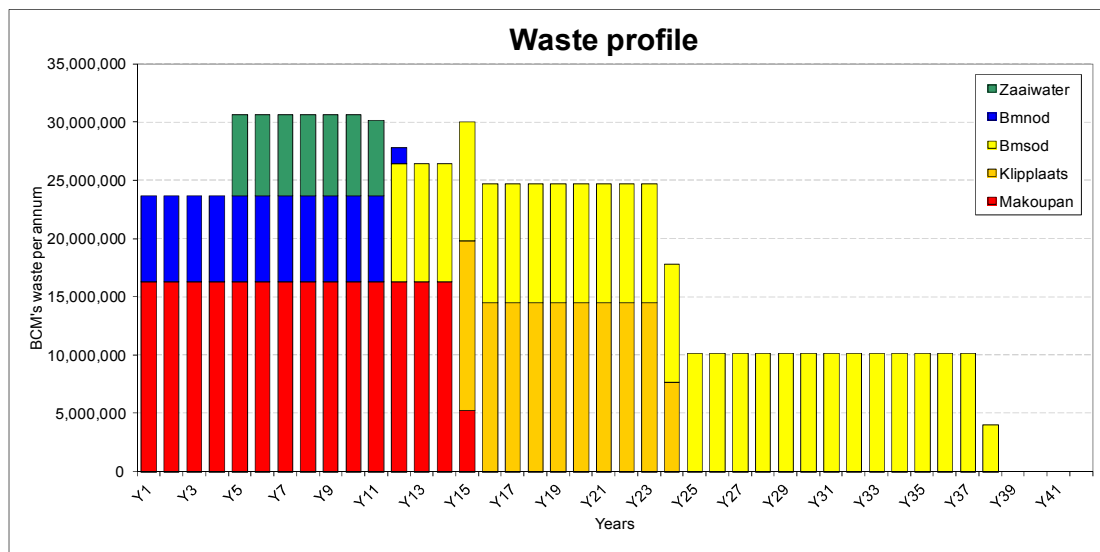


**Figure 5.17 Graph displaying the waste removal profile for the company up to year 38.**

### 5.8.2   ROM production

With the plant feed capacity at 10 million tonnes per annum, the ROM production performs slightly above this level for the first couple of years and then remains consistently around the target levels until the last three years of the product. This will ensure that the plant always has coal available to perform at optimal levels. Should the feed stock levels become very high, some of the higher quality Eskom product can be crushed and screened and sold as a low grade product.
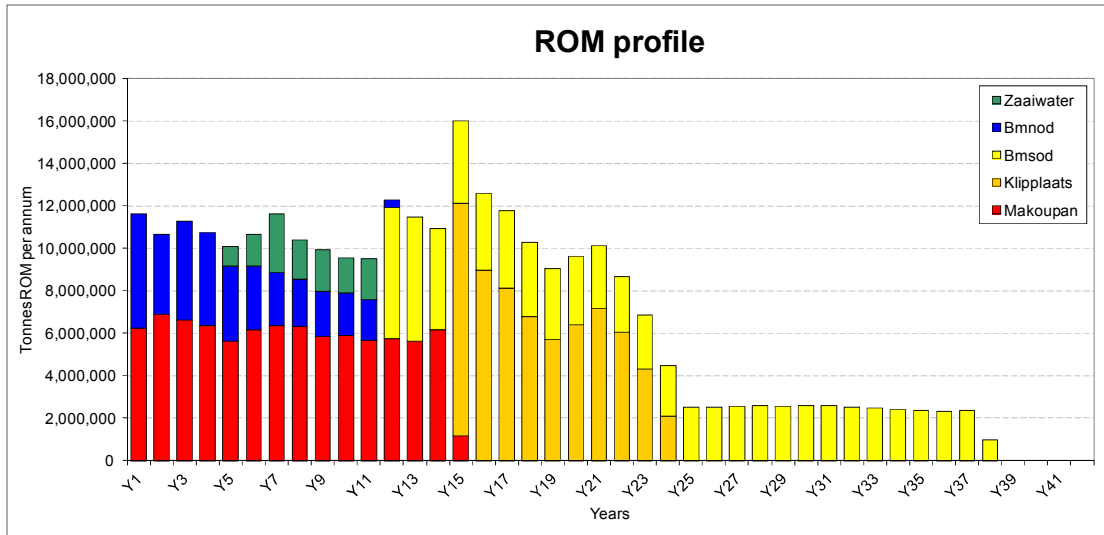
**Figure 5.18 Graph displaying the ROM production profile for the company up to year 38.**

### 5.8.3 Export Product

As much of this process is aimed at using the available rail capacity fully, great care was taken to tweak the schedule to ensure sufficient supply whilst keeping stockpile levels realistic. Figure 5.19 shows the export product profiles with Figure 5.20 illustrating the surplus/shortfall of export product relative to the rail allocation, indicating the required capacity of the product stockpiles.
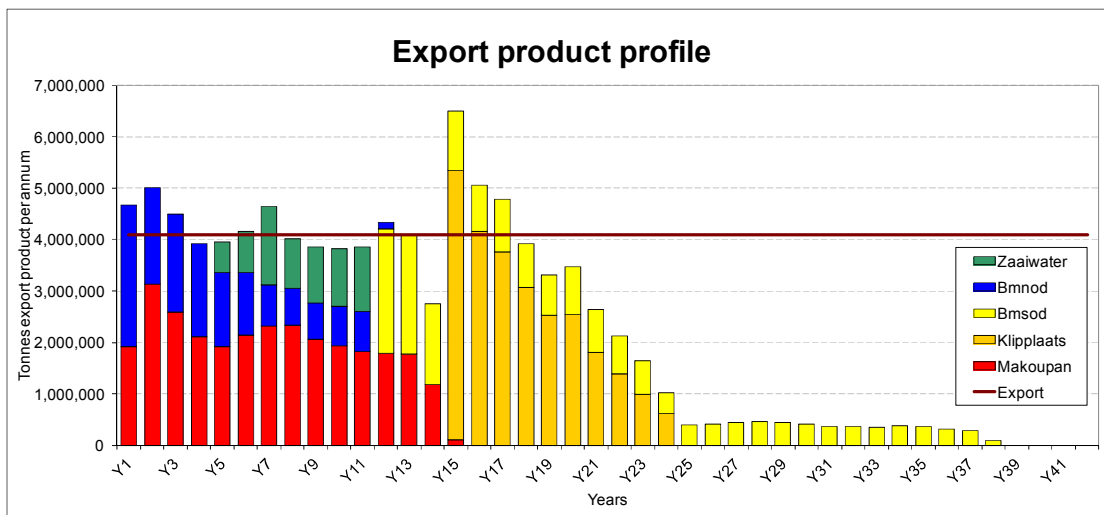


**Figure 5.19 Graph displaying the export product tonnage profile for the company up to year 38.**
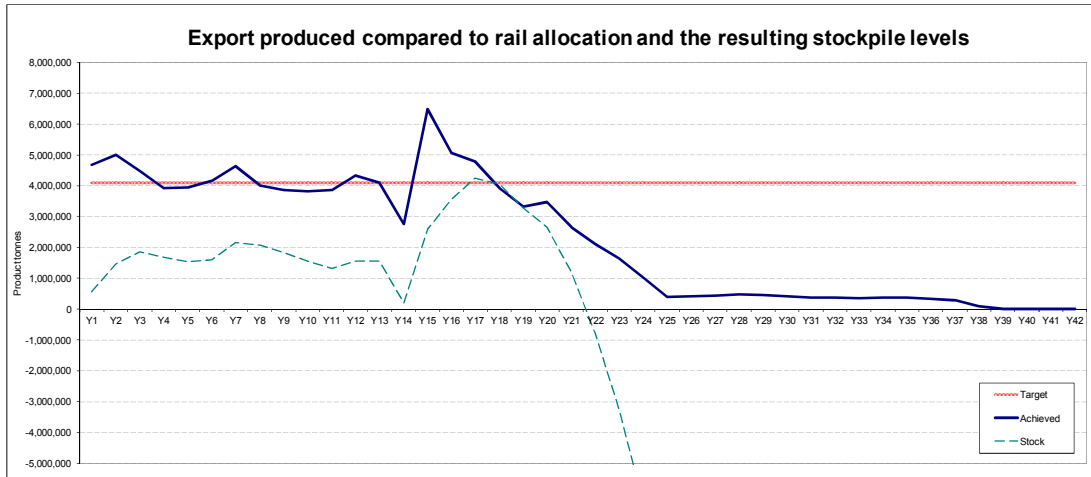
**Figure 5.20 Graph displaying the export product tonnage profile for the company up to year 38, with the rail allocation indicated by the red line. The resulting stockpile levels (indicated by the blue dotted line) peak at approximately 4 million tonnes in year 17. From there additional resources are required.**

### 5.8.4 Eskom Product

The Eskom product is not a primary driver of the process and thus results from the alterations to the other parameters. It is wise to monitor the expected levels as well in order to negotiate expected sales with the local power producer Eskom or any of the intermediaries.
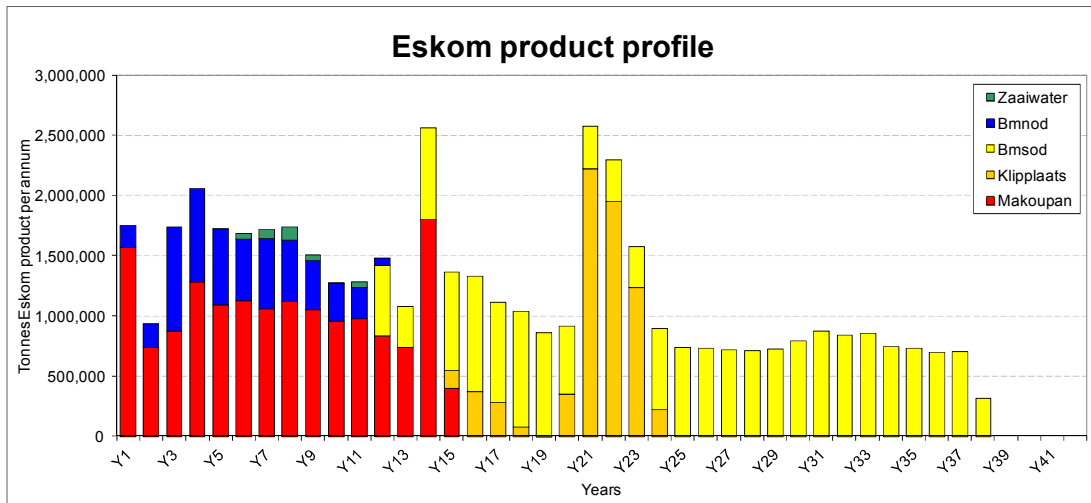


**Figure 5.21 Graph displaying the Eskom product tonnage profile for the company up to year 38.**

### 5.8.5 Tax Impact

As part of value, the value to the mineral owner's must be considered. With all mineral resources owned by the government in South Africa the value gain by taxation needs to be considered. This will also indicate the degree of responsible utilisation of the resource.

The true value to the government, and subsequently the country, can indeed become very complex when aiming to understand and quantify the degree to which the mining industry stimulates the economy apart from the direct value arising from taxation policies. The basic question that needs to be answered is whether the maximum value mining rate differs for the company and the country, and for a comparative analysis the direct value should be a good enough indicator, thus the complex stimulation effect will be ignored.

For the purpose of the exercise 28% of the taxable income was used to calculate the income tax from the company. To determine royalties the formula from the Royalty Act (DMR, 2008) is used –

Percentage royalty = 0.5 + [EBIT / (Revenue x 9)] x 100………(4)

VAT is calculated on all purchased equipment as 14% of the purchasing price. Roughly around 60% of all start-up equipment is imported (personal conversation with T. Howard, 2010), and these will be subjected to additional import duties as well. A figure of 20% is used on the original import price to calculate the import duties.

Average taxation values for each of the various working classes are applied, where officials are taxed at 35%, union men at 25% and Operators at 15% (Xstrata Coal, 2010). These are approximate values gained from the financial department, but will vary depending on the average salaries the companies offer in these classes.

As there is no capital investment from the government a net present value or internal rate of return will have no meaning in determining the value, thus the present values are calculated and added to produce a total present value tax. There might be an argument as to whether taxation value should be calculated using the time value of money principal, in the same manner as value to the company is determined. In the instance of mineral resources it is the opinion of the author that the time value of money plays a critical part in the value to the country, as the industry is utilising a non-renewable resource and any funds gained from this resource are crucial in researching and developing new technologies that will serve as a viable substitute should this resource be depleted. Thus funds early in the life of the mine are more valuable to the government than funds towards the end of the operation.

From figure 5.22 it can be seen that the total tax increases the lower the mining rate becomes. This is mainly due to the increasing company tax from the operation which is the biggest contributor to the total tax. The present value tax is

somewhat lower at lower mining rates as the tax gained from duties and VAT on the imported equipment is significantly more as the mining rate increases. With that tax available early in the life of the project is could potentially be more valuable as it can aid in the development of alternative sources of energy.
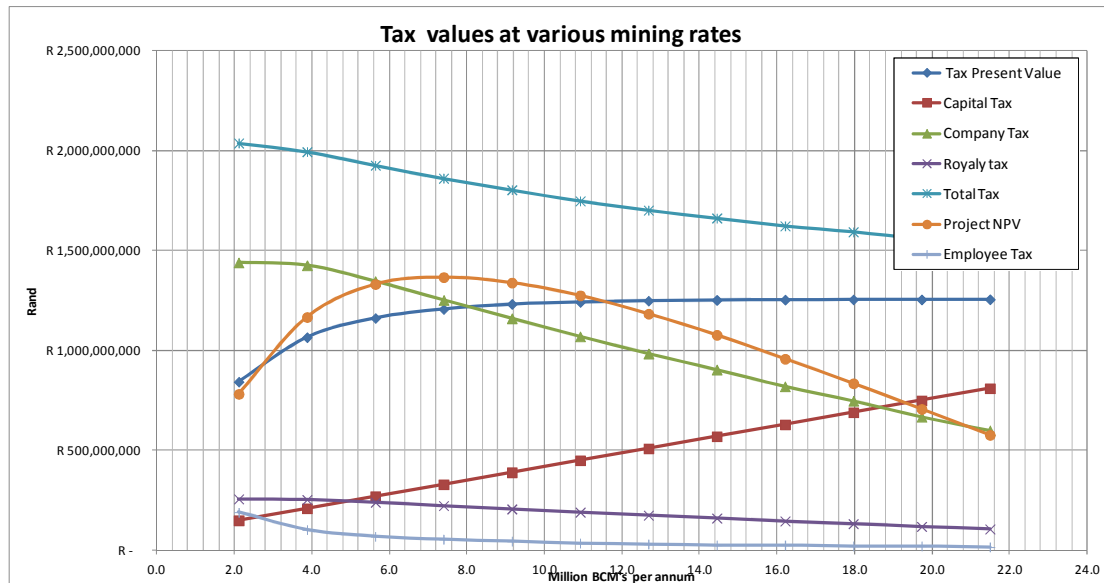


**Figure 5.22 Graph displaying the various tax values gained from the specific sources at different mining rates.**

# 6. Discussion and conclusion

Even though many projections of the coal price indicate a positive growth (McCloskey's. 2008), the risk of price reduction and/or significant mining cost growth needs to be considered when estimating the mining rate and scale of the operation. Other methods or applications propose optimisation through open pit design (Cacetta and Hill, 1999) or detailed sequencing (Minesight, 2009 and Fytasa *et al.*, 1993). This thesis provides a method to evaluate the optimal mining rate for an operation through comprehensive scheduling and financial simulation early in the life of the project, which will give an indication to current owners or prospective investors as to the scale and value of the operation. This will in turn give guidance as to the magnitude of the capital required to advance this project into execution. Advanced projects can also utilise this method to determine the optimal mining rate by expanding the basic waste removal capacity versus cost model used in this thesis. By determining the maximum value mining rate and risk adjusted mining rate, a range in which the operation should yield the best return is given and can be used to adjust the scale of operations in order to create a consistent and practical production profile for the company.

As this method provides a comprehensive financial evaluation grounded in the resource evaluation, the relationship between maximum value and characteristics

of the resources can thus be tested. By correlation the various characteristics of the resources to the maximum value, a trend indicating dependency would be useful as these can give basic rule's-of-thumb to evaluators when investigating operations on a high level.

The various parameters from raw qualities, in-situ resource size, processing yield, total saleable tonnage, average ROM and saleable strip ratios were calculated and compared to the maximum value for the operation.

As maximum value is gained mainly from the export product it is not surprising that the maximum value mining rate is related to the total tonnage of export available in the resource. This relationship indicates that the optimal annual (waste) mining rate equates to roughly two times (1.98) the total export product in the life of the mine. This 'rule' has only been applied in those instances where the average ROM strip ratio is less than 4 and would not necessarily apply in operations with ROM strip ratios higher than that.

Smith (1997) is hesitant to use the maximum NPV value as guidance towards optimising the mining rate as it "gives a very short mine life" which raises such concerns as little recovery time should the start-up incur problems, increased environmental disturbances and reduced social advantages. With a growing demand for energy (Blumenfeld, 2008) the growing need for coal supply remains the key factor, and with potential risk compensated for by adjusting the optimal mining rate, as well as the value to the country being addressed, it is the opinion of the author that by ensuring profitability, the mining companies ensure continual consistent supply of resources to an ever demanding world.

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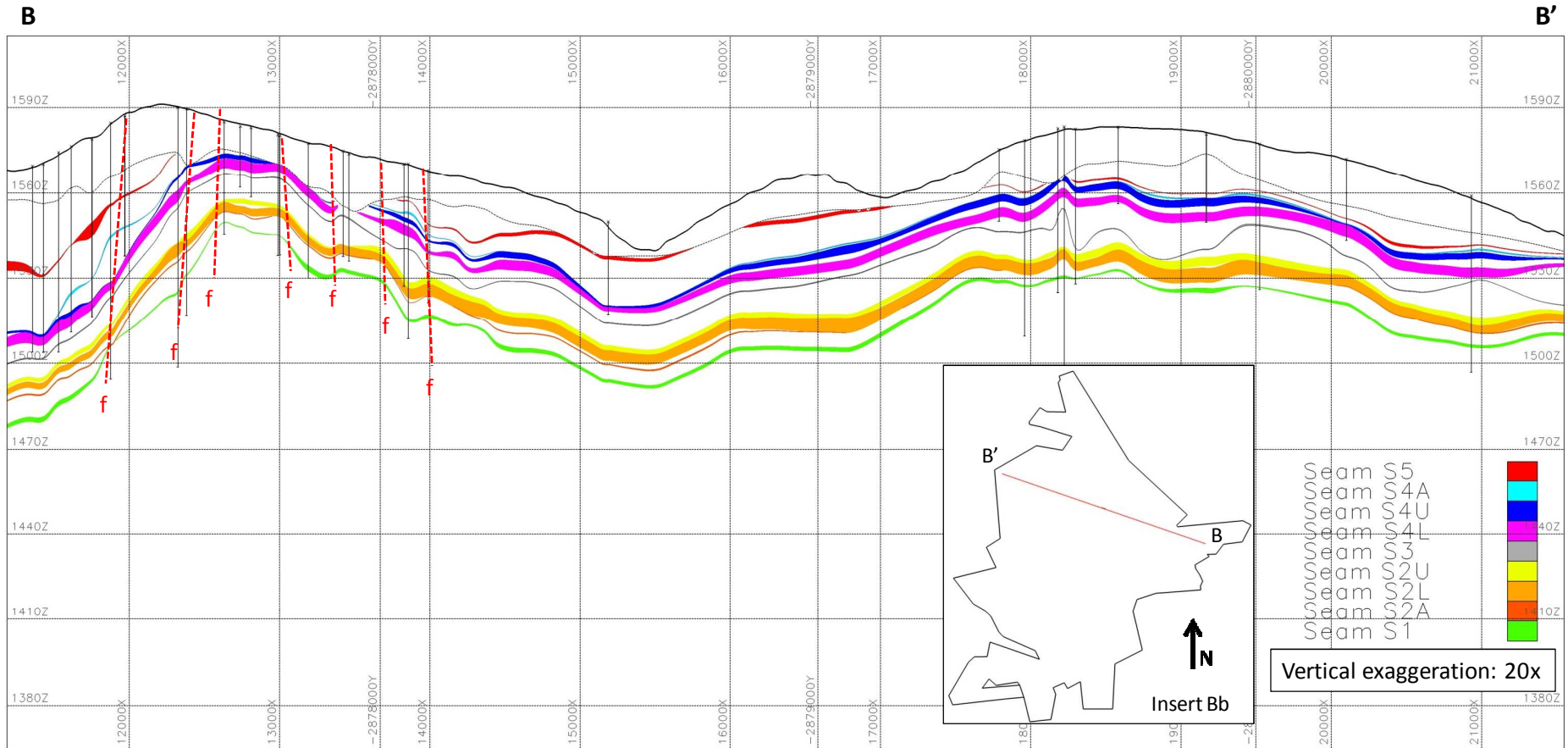Xstrata Coal (2010) – *Life of Mine financial parameters*. Internal Guide to LOM evaluations.
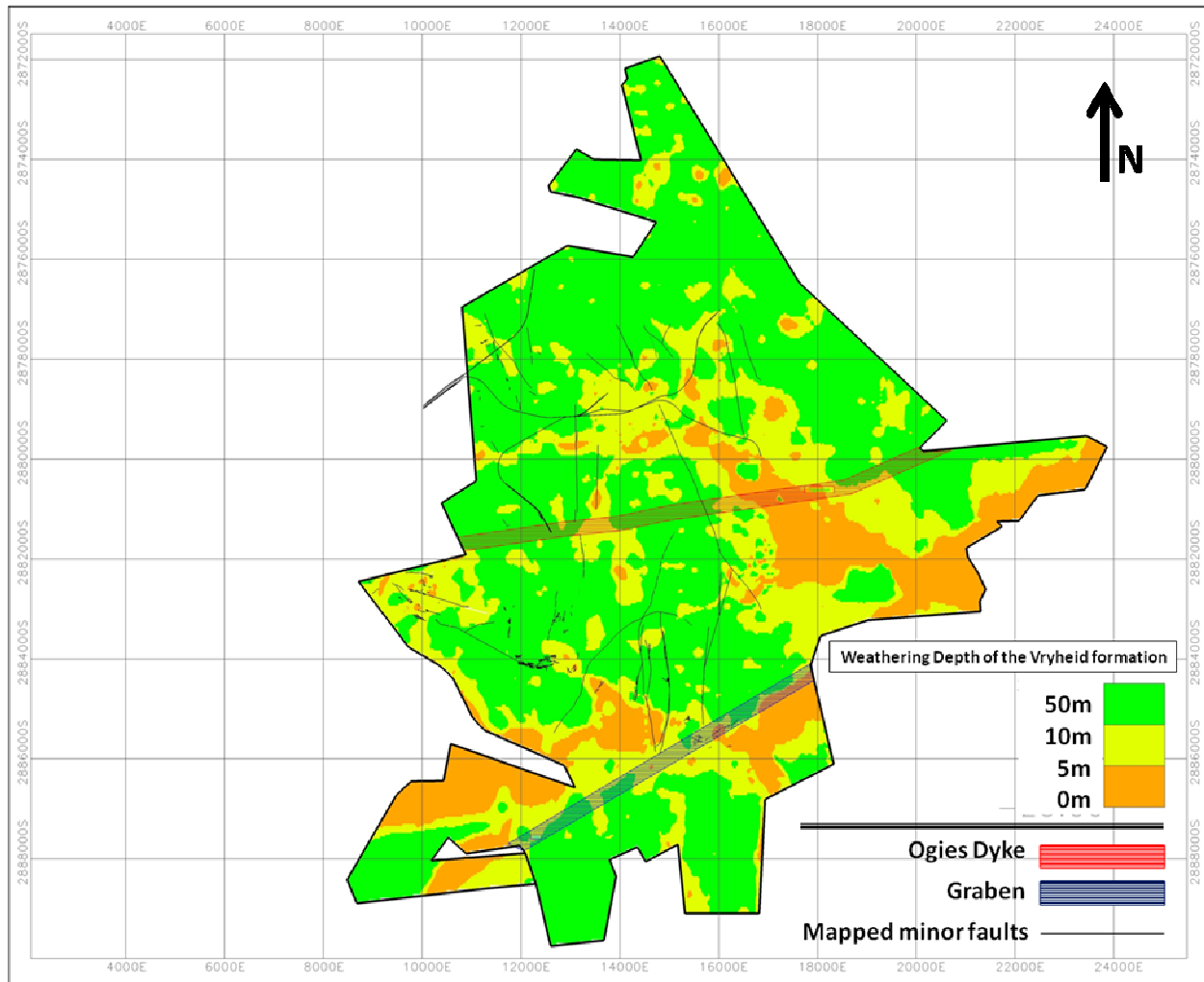
# 8. Appendices
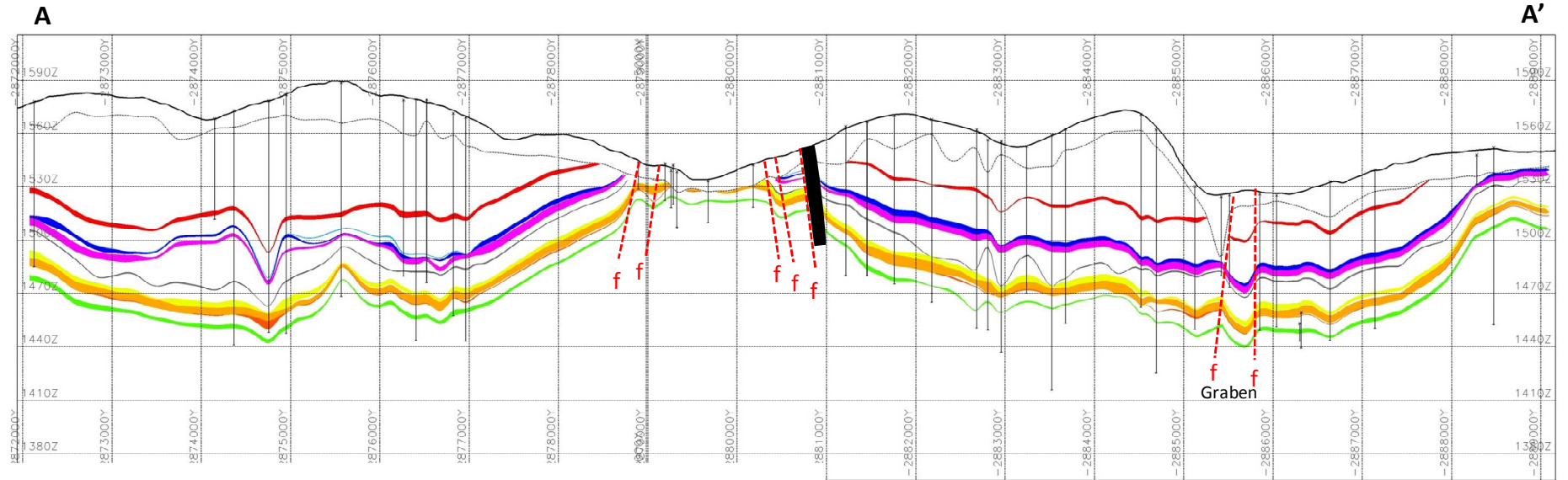


**Appendix A : North – South cross section as indicated on Insert Ab, illustrating the seam behaviour through the complex. The interpreted major faults are also indicated by the red dashed lines on the cross section.**

A
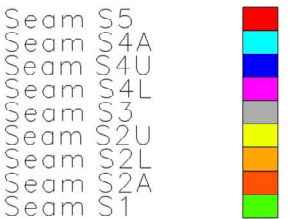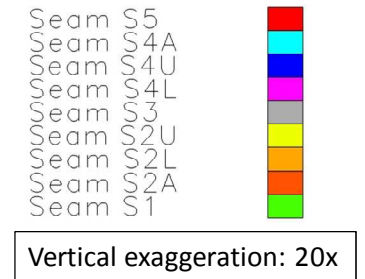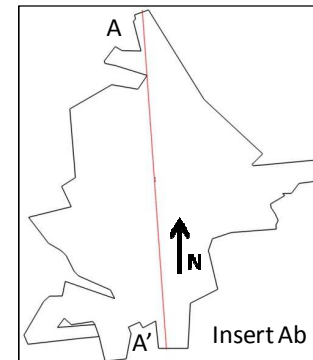
**Appendix B : East-West cross section as indicated on Insert Bb, illustrating the seam behaviour through the complex. The interpreted major faults are also indicated by the red dashed lines on the cross section.**

B

**Appendix C : Basic geology map indicating the weathering depth of the Vryheid formation that covers the entire study area.  The Ogies Dyke and the Graben are indicated on the plan, with minor faults as mapped in the underground operations.**

C