

# RISK MANAGEMENT IN MINING AND MINERALS ECONOMICS AS WELL AS MINERALS RESOURCE MANAGEMENT

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**A thesis submitted to the Faculty of Engineering,  
University of Witwatersrand, in part fulfilment of the  
requirements for the degree Master of Science.**

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## **Declaration**

I declare that this thesis is my own, unaided work. It is being submitted for the degree of Master of Science in the University of Witwatersrand, Johannesburg. It has not been submitted before for any degree or examination in any other University.

Signed \_\_\_\_\_

Dated this .....day of .....2004

## ABSTRACT

The field of risk management has been growing in popularity over the last few years. Risk management is not a new concept but is becoming more important since the release of the Turnbull report.

This research reviews all the risk management systems currently available in the mining industry. The focus of this research is from a Mining Economics as well as a Minerals Resource Management perspective.

It is the Mineral Resource Managers primary task to ensure that the orebody is extracted in the most optimum method to ensure the maximum return for the shareholder. In order to do that, the Resource Manager needs a good understanding of the ore body as well as the extraction methods and the cost of mining. Recently it has become important to understand the risks around the mining process as well.

It was found that the principal risk associated with mining is extracting the orebody sub economically and hence the research focus was on optimisation. Three tools have been designed to facilitate the determination of optimisation. The above three tools have been tested in practice.

The first section of research focuses on how risk is defined in the industry. There is also an analysis what a Mining Economist and A Mineral Resource Manager will encounter in terms of risk.

The second section covers the Basic Mining Equation (BME) and its uses. The research looks at using stochastic methods to improve optimisation and identifying risk. The @Risk software was used to analyse 5 years of historical data from an existing mine and predicting the future, using the distributions identified in the history.

The third section is based on the use of the Cigarette Box Optimiser (CBO), where the cost volume curve and the orebody signature are used to determine optimality in returns. It also looks at various forms of the BME and how it can be used to identify risk. The section also covers quantification of risk from a probability perspective using systems reliability logic.

The fourth section centres on the Macro Grid Optimiser (MGO), which considers the spatial differentiation of the orebody and determining the optimality through, an iterative process.

The last section analyses risk from a Mining Economics perspective. It considers the use of the 'S-curve' to determine risk. The section also includes a high-level shaft infrastructure optimisation exercise.

As an overall conclusion, it was found that the biggest risk associated with mining could be to extract the orebody sub economically. Some ore bodies could yield double the return that they intend to extract. In order for that to happen, the extraction program should undergo some form of optimisation. This will ensure that the optimal volume, cut-off, selectivity and efficiencies are met. **There is no greater risk**

**than to mine an ore body out without making an optimal profit.**

We are in mining to make money! Cash is king!

## ACKNOWLEDGEMENTS

I acknowledge Anglo plc for giving me the opportunities to develop my skills over the last 33 years. They have afforded me many opportunities and this document would not have been possible without those opportunities.

Mr Dave Diering who has enhanced the Minerals Resource Management discipline by quantum leaps. His reinvention of the Basic Mining Equation (BME) has grown in popularity and can now be found in all the divisions of our company. His passion for MRM has given birth to this research.

Professor Dick Minnitt for supporting and motivating me over the last three years.

Mr Bill Abel for supporting me and challenged my thinking. I believe he added significant value to this research... Yes Bill, it is now finished!

Dr Christina Dohm for the support and I hope we will be able to take some of the thinking in this document to new heights. I believe the Macro Grid Optimiser will still add significant value. We should move away from modelling grade; we should model profits!

Thanks to Fiona Donnan, Meera Jainarian, Debbie French and Beverly Clark for assisting me with my two finger keyboard trouble and fixing my tenses and grammar.

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Rocco Adendorff, probably one of the most experienced shaft specialists in the world, for his motivation and help with the thinking concerning shafts. I believe ultimately Mining Economics will determine all shaft depths and other parameters.

## GLOSSARY OF ABBREVIATIONS, SYMBOLS AND TERMS

BGE	Basic grade equation
BME	Basic Mining Equation
CBO	Cigarette box optimiser
Cu	Copper
GLE	Greatest living expert
IRR	Internal rate of return
ME	Mining Economics
MGO	Macro grade optimiser
MRM	Minerals Resource Management
Ni	Nickel
NPV	Net present value
SG	Specific gravity
m	Metres
cmg/t	centimetre gram per tonne
m <sup>2</sup>	square metres
m <sup>3</sup>	Cubic metres
kg	kilograms
ktpa	kilo tonne per annum
R/kg	Rand per kilogram
%	Percentage
g/t	Grams per tonne



## DEDICATION

I dedicate this research to my wife Anne de Jager who has stood by my side throughout my career. She took a big risk by marrying a young sampler and hopefully it paid off in the end. She mitigated some risk and controls the residual risk through her own Risk Management program.

She continuously inspired me and supported me in spite of many hours she spent on her own whilst I was learning. I must also declare that she is a wonderful cook and ensured I was well fed and happy all the time.

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# **1 OVERVIEW OF THE RESEARCH**

- *Overview of the research.*
- *Methods and tools used*
- *What is risk?*
- *What is Minerals Resource Management?*
- *What is Mining Economics?*
- *Mining Economic risk*
- *Risk in the mining industry*

## **1.1 Background**

I have had the good fortune to spend time on many mines during the last 32 years, starting as a sampler (highly skilled at swinging a hammer in confined areas) to being a Mining Economist, (auditing and high level Minerals Resource Management and Mining Economics functions at our operations, as well as reviewing new projects). There have been significant changes in the way the orebody is managed over the years and the levels of professionalism have increased by quantum leaps over the last five years. Pockets of excellence in the application and practice of Minerals Resource Management were observed, as well as some very poor approaches to this discipline within some departments. It is the Mineral Resource Manager's primary task to ensure that the ore-body is extracted in the optimal way to ensure the maximum return for the shareholder. In order to do that, he or she needs a good understanding of the

orebody, the best extraction methods, and aspects of the costs involved. In recent years, it has become important to understand the risk associated with the process of mining as well and the management of this process forms the most significant branch of this research.

During the past six years, I have been involved in auditing and played a part conducting feasibility studies of around 80 mines including gold, copper, diamond, coal, iron ore, nickel and platinum mines. Understanding the orebodies in these mines has become increasingly sophisticated, with the introduction of 3D modelling and conditional simulation techniques. There is a concern that in some cases advanced software is being applied without the operator having a good understanding of the underlying theory and techniques associated with them. However, this problem is disappearing rapidly.

The Mining Design function is also improving rapidly but lacks the capacity to compare the efficiency and returns on different mining layouts. Mine designers are not yet able to compare the impact of a cheaper layout against the increased risk profile such a layout might have. The planning process is a concern, as it appears that short-term thinking drives the process. With the huge capacity of computerised planning packages it is possible to extend short-term planning with all its attendant detail into the long-term mine plan. However there is a danger that the short-term imperfections and problems could be compounded as they are extrapolated into the future.

For example, the current reserves shortages may prohibit selective mining in the current time period, but the danger is that this could become a norm for the future. One mine in particular was designed in such a way that it would realise R2 billion over the life of mine based on an NPV of 10%. Detailed interrogation of the mine and milling stages indicated that there was spare capacity that could be filled with very little effort. A simple redesign resulted in an improved realised value of around R4 billion. Selective mining could have added another R0,5 billion. Such uncritical approach's to mine planning could have destroyed some 60% of the financial potential of the mine!

Application of 'Whittle' and NPV scheduler software to open-pit operations has greatly improved the efficiency of mineral extraction and some mining houses are currently designing software tools for long-term and strategic planning.<sup>1</sup> However, the simple methods that are the focus of this research do provide assistance in the optimal extraction of valuable ore bodies

## **1.2 Risk in mineral resource punishment and mining economics**

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<sup>1</sup> Whittle J. (1989). *The facts and fallacies of open pit optimisation.*



The statement that risk, is poorly understood and poorly developed in most operations requires some qualification. The concept of risk in mineral extraction is appreciated and allowances have been made for it for over many years. However the weakness in the understanding of risk as it applies to Minerals Resource Management is around formal quantification of risk and appropriate mitigation techniques.

This report describes how risk can be identified, quantified, mitigated and controlled from two perspectives. These are:

- Mineral Resource Management (MRM); and
- Mining and Minerals Economics (MME).

Much research has been carried out around risk, and from different perspectives. In the mining industry, the bulk of the risk analysis has been on Safety and Health issues. There have been significant attempts to quantify risk in terms of the orebody, considering the introduction of the **SAMREC and JORC Codes** that are now commonly used in South Africa.

These codes focus principally on the classification of Resources and Reserves, but very little research has been done in terms of risk associated with the planning process, an area that impacts the overall profitability of most mines. The planning stage provides the opportunity to pro-actively mitigate risk and benefits can be

measured in the billions rather than the millions of Rands, especially when optimisation is considered.

There are two main areas in which risk can be identified and mitigated, and these include:

- Mineral Resource Management (MRM) where the practitioner quantifies and classifies the orebody, and then plans and controls systems around its extraction.
- Mining and Minerals Economics, a discipline that involves marketing, pricing, modelling and financing mineral extraction. Although these two fields have much in common, for the purpose of this analysis, they are considered to be two separate fields.

The final step is to produce a generic approach that quantifies and evaluates risk. This is based on spreadsheets that are practical and simple. Experience shows that the more complex a method is, the less the understanding of it, and the smaller the likelihood of acceptance of such a method. This research focuses on the simple issues and is not a definitive all-inclusive system.

### **1.3 Methods and tools utilised**

This research puts different techniques and ideas together in a systematic approach to identifying and mitigating risk and includes software (spreadsheets) that can be used to optimise the extraction of a typically tabular orebody.

The research uses a fairly simple equation relating to the value that accrues to a mining operation through the extraction, processing and sale of a single unit of ore. This equation, which is shown in Table 1 has in recent years been popularised by Mr Dave Diering and is the basis for the development of the software tools that are used later in this research. The two main spreadsheet based tools are:

- Cigarette box optimiser (**CB0**); and
- Macro grid optimiser (**MGO**).

### **1.3.1 The Basic Mining Equation (BME)**

The importance of the BME in valuation and optimisation of mineral extraction can be attributed to Mr Dave Diering, one of the world's foremost experts in practical Minerals Resource Management of Wits-type gold deposits. The BME as shown in Table 1-1 is essentially the algorithm of the operation combining the critical variables in order to determine the expected profit. The importance of the BME is that it provides a means of measuring the impact of changes in the variables on the value of the mine. The value derived using the BME is a snapshot in time and does not consider the impact of the time value of money on the decision. It is mainly used to prioritise and identify critical risks. The BME shown in Table 1-1 in its simplest format is typical of a Wits-type gold mine. This equation and its derivatives are discussed and used in chapters 2 and 3 of this report, and refer to '**Stochastic analysis**' and '**Optimisation**', respectively.

1		FACE LENGTH	m	2,656
2	x	FACE ADVANCE	m	9.66
3	=	TOTAL m2	m2	25,647
4	x	ON REEF PERCENTAGE	%	95.54%
5	=	REEF m2	m2	24,502
6	x	ON REEF cmg/t	cmg/t	1,556
7	x RD =	kg GOLD EX STOPES	kg	1,060
8	+	VAMPING kg	kg	40
9	+	REEF DEVELOPMENT kg	kg	20
10	=	TOTAL kg BROKEN	kg	1,120
11	x	MINE CALL FACTOR	%	97.50%
12	x	RECOVERY FACTOR	%	97.20%
13	=	GOLD RECOVERED	kg	<b>1,061</b>
14	x	GOLD PRICE	R/kg	R72,102
15	=	REVENUE	R ('000)	R76,498
16	-	PRODUCTION COST	R ('000)	R60,662
17	=	CONTRIBUTION	R ('000)	R15,836

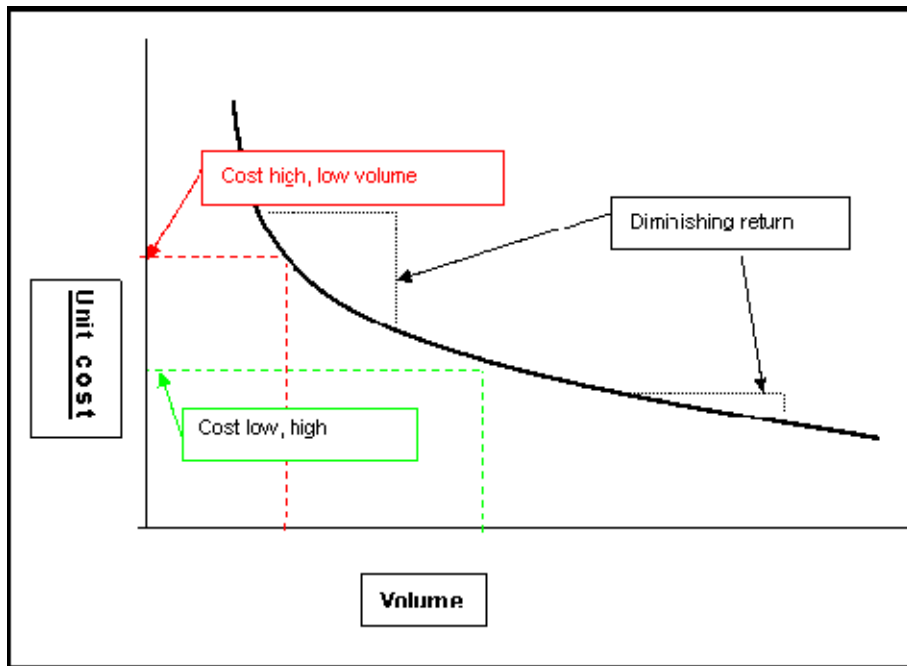
Table 1-1: A typical Basic Mining Equation (BME)

### 1.3.2 Cigarette box optimiser (CBO)

The Cigarette box optimiser was originally designed to provide 'the back of a cigarette box' estimate of value that could be derived from the extraction of certain grades of ore from a specific area of a mine. Over the years since its first formulation by the author the concept has grown and become progressively more sophisticated. In its current form optimiser uses a combination of the cost-volume curve, the grade-volume signature of the orebody, the BME, and an NPV calculation.

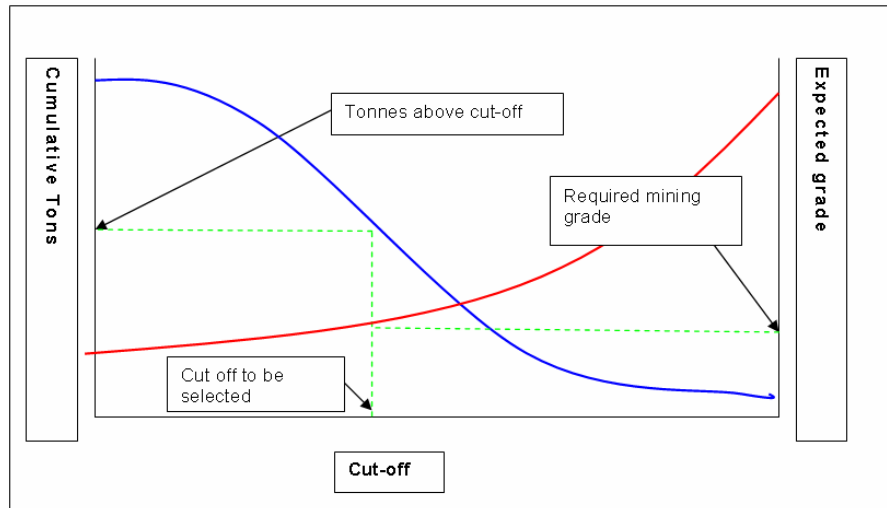
The CBO works on the premise that increasing or decreasing the mining volumes; will move the position on the unit cost line of the cost-volume curve. Economies of scale dictate that the higher the volume of ore mined, the lower will be the unit cost as a result of not having to increase the fixed costs of mining.

The effect of diminishing returns and an increase in risk with higher volumes also needs to be considered. If you reduce your mining volume the opposite comes into play. The purpose of varying the mining volume is to determine the cost volume relationship, which determines the unit cost. This in turn impacts on the cut-off grade and pay-limit.



**Figure 1-1: The cost-volume curve showing the economies of scale derived by increasing output without increasing fixed costs**

A second aspect of this model is related to the orebody signature. Figure 1.3 depicts how the cut-off grade can be graphically determined from the grade-tonnage curve.

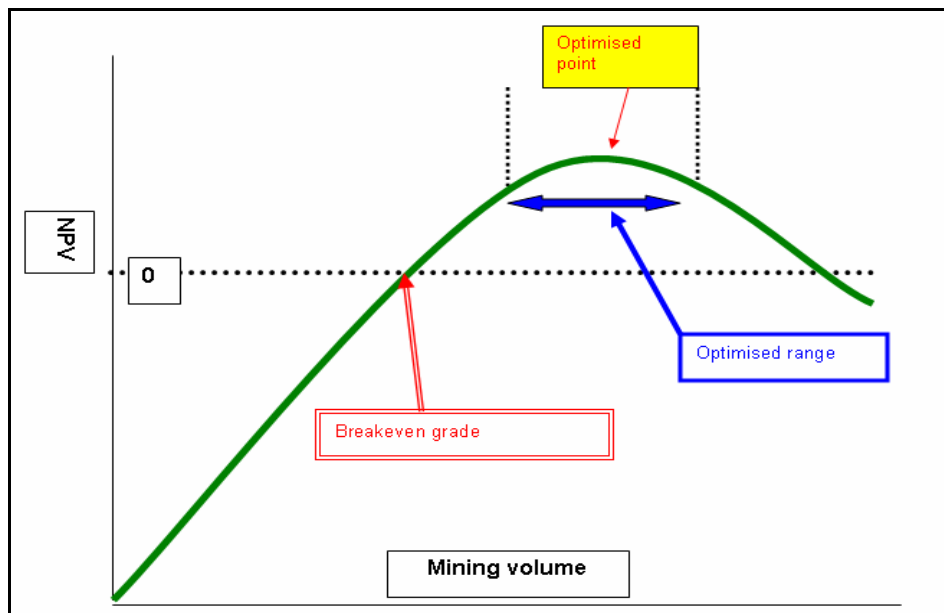


**Figure 1-2: A typical grade-tonnage curve showing increasing expected grade and decreasing tonnages as the cut-off grade increases**

Figure 1-2 also illustrates that increasing the grade of ore sent to the mill through selective mining, means that the volumes available for extraction will decrease. Conversely the life of mine can be increased if the cut-off grade is reduced and volumes available for mining are increased. Increasing the volumes mined introduces risk at other downstream positions in the extractive process. These risks include the impact of increased volumes on mining and milling capacity and on the market in which the final product is sold. There is no simple indication of how increased volumes will impact the risk profile of the mining operation, but 'expert opinion' based on past

experience would be a valuable input. It should be remembered that the mining grade is determined by the required profit margin, while the optimum cut-off grade should be chosen at the point that the NPV is maximised.

A typical representation of the relationship between the grade-tonnage curve and the cost-volume curve is shown in Figure 1-3 that plots the NPV versus the volumes mined. The mine is profitable in any region above the NPV = 0 line but the NPV is maximised over a range of mining volumes. The main drawback of this method is that it cannot incorporate the dynamic changes in volumes mined from year to year, nor does it account for the special physical characteristics of the orebody such as variations in grade from place to place.



**Figure 1-3: A NPV versus mining volume curve that identifies the optimal NPV over a range of mining volumes**

A critical component of selective mining of an ore body is the ability to control the grade accurately. The simple plot of NPV versus mining volume shown in Figure 1-3 is too inaccurate to make it a definitive method for establishing appropriate mining volumes determining the right size of the operation, identifying the appropriate cut-off grades and achieving the best profit margin. Rather than selecting one specific mining volume to mine it is probably better to consider a range of solutions. A method for achieving optimality is discussed in detail in Chapter 3 (**Optimisation**).

### **1.3.3 Macro grid optimiser (MGO)**

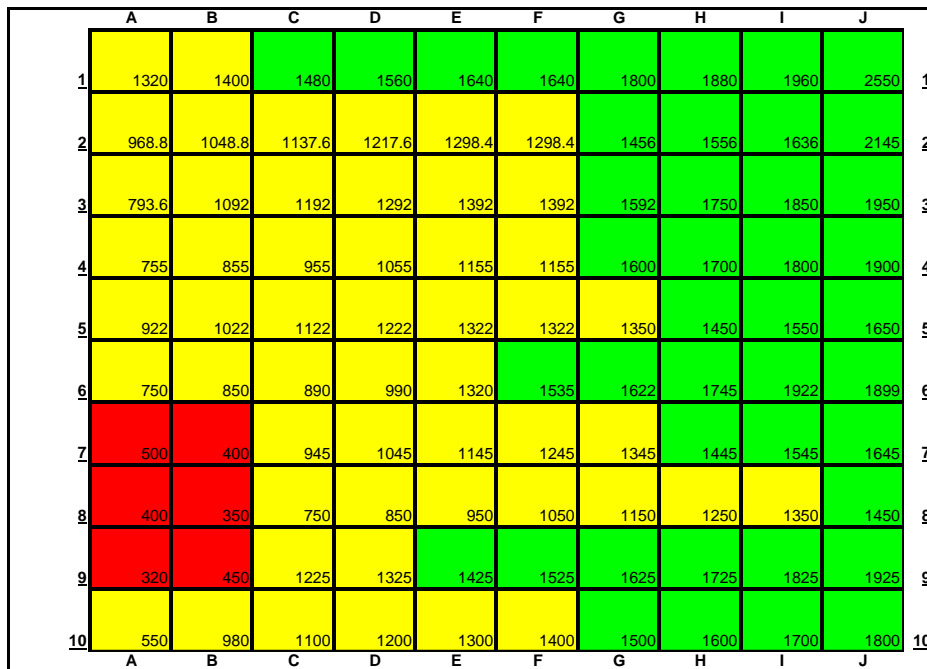
The second spreadsheet-modelling tool takes the spatial distribution of reserve and resource blocks into account and provides a visual means for optimising the mineral extraction. A decision to mine specific reserve blocks depends on the cut-off grade since this is determined by costs, which are themselves determined by the mining methods, the distances from shafts, access and required services. By definition reserves are supported by appropriate infrastructure and access, whereas resources often require significant capital expenditure before they can be transferred to the reserves category.

The grade of each block is determined from the macro Kriging model of the ore body and is shown in Figure 1-4. The breakeven cut-off grade is determined using the BME. Blocks below the cut-off grade are shown in red, whereas blocks with grades that lie between the



cut-off and the pay-limit are shown in yellow; green blocks are those above the pay-limit, i.e. the profitable ore.

A visual inspection of the output from the macro-Kriging (Figure 1-4) suggests that the top right-hand corner of the area holds the best potential, and the principle of the time value of money requires the highest-grade ore to be extracted as swiftly as possible.



**Figure 1-4: A macro-Kriging output shown on a square grid**

At this point there is no way to distinguish one green block from the next, but depending on geological complexity and the cost of access and associated infrastructure the total cost of extraction for each block will be different. The development costs for each block (i.e. the costs required in order to convert a resource block to the reserve category) are determined and then converted to an equivalent gold

grade. The cost in terms of grade is then subtracted from the kriged grade for each block, giving a residual grade attributable to the block after it is fully developed.

The process of reducing the grade by subtracting costs as equivalent grades is repeated until all standard and anomalous costs items are accounted for. The cost items to be accounted for include:

- Major infrastructure (split back to the blocks serviced);
- Development;
- Services;
- Mining method;
- Balance of overheads; and
- Risk.

Figure 1-5 shows the residual gold grade or 'profit grade' in blocks that are accessible to mining and classified entirely as reserves. The individual blocks are now identical in terms of mining potential and 'deliverability' to the mill and are therefore financially comparable on an equal basis.

	A	B	C	D	E	F	G	H	I	J	
1	1223	1303	1383	1463	1543	1543	1703	1783	1863	2453	1
2	872	952	1040	1120	1201	1298	1456	1459	1539	2048	2
3	696	995	1095	1195	1295	1295	1553	1653	1753	1853	3
4	658	758	926	1016	1106	1058	1542	1603	1742	1803	4
5	825	925	1099	1222	1303	1264	1331	1353	1501	1553	5
6	653	753	793	990	1320	1535	1622	1726	1893	1802	6
7	403	390	906	1045	1048	1148	1316	1348	1448	1548	7
8	303	321	750	850	853	953	1053	1153	1253	1353	8
9	223	353	1128	1228	1328	1428	1528	1628	1728	1828	9
10	453	883	1003	1103	1203	1303	1403	1503	1603	1703	10
	A	B	C	D	E	F	G	H	I	J	

**Figure 1-5: Residual gold grades in reserve blocks that are fully developed.**

As the size of each block is known the profit attributable to it can be determined. The value of each block is discounted for time and the percentage of extraction of each block. The sum of the discounted values of all mining blocks is equal to the NPV of the mine before taxes and finance.

Once the basic inputs (kriged block values) have been made the spreadsheet can be used to experiment with a variety of 'What if' scenarios can be examined in order to determine the best mining strategy. The best mining strategy will in turn provide the basis for planning the programme of underground development and access to

the ore. This macro plan is then entrusted to the planner, who can turn it into a more accurate reality. Figure 1-6 shows both what blocks should be mined and the expected NPV associated with each block.

NPV of operation										322
A	B	C	D	E	F	G	H	I	J	
0	0	0	1	4	3	13	14	14	27	
0	0	0	0	0	0	3	3	5	20	
0	0	0	0	0	0	6	10	13	15	
0	0	0	0	0	0	11	13	9	15	
0	0	0	0	0	0	-2	-5	-5	5	
0	0	0	0	33	30	32	28	21	17	
0	0	0	0	0	0	-3	-2	-3	2	
0	0	0	0	0	0	-8	-8	-12	-2	
0	0	0	0	0	-2	2	5	-12	6	
0	0	0	0	0	0	-1	1	3	5	
A	B	C	D	E	F	G	H	I	J	

**Figure 1-6: The distribution and expected NPV of ore blocks for extraction.**

The current development and mine planning procedures require several days of the mine planners time and there is little scope for comparing a wide range of scenarios or for making significant changes after the mining layout has been accepted. The preferred method for maximising the extraction in any area is to increase the Mine Call Factor until an enhanced profit is achieved. This is a

dangerous practice as the planning credibility is jeopardised in the process.

The macro grid optimising methodology referred to as '**Macro Mining Grid**' is described in detail in Chapter 5 of this report, but the value of the technique has been demonstrated in practice at a number of mines.

#### **1.4 Risk Elements In Mineral Extraction**

##### **1.4.1 Definition of Risk**

Risk is about uncertainty and the likelihood that things will turn out differently from what **we anticipated they would**. This last statement suggests that after the event we have the benefit of hindsight and are then in a position to compare what we thought would happen and what actually happened.

Clark (2000) states; "What is obscure is seldom clear". This often leads to questions about trust. How good is the plan, how achievable is it? The principal dilemma in mine planning is that future extrapolations of what can be achieved in terms of the percentage extraction and the efficiency of extraction are mostly driven by historical performance, i.e. past achievements are used as a basis for predicting what can be achieved in the future. The problem is that yesterday's records become tomorrow's standards but the use of these standards does provide a benchmark against which

performance can be measured. Such an approach involves risk and uncertainty.

The Concise Oxford Dictionary (1985) defines the term "risk" as:

*A venture undertaken without regard to possible loss or injury. (Example: "He saw the rewards but not the risks of crime")*

The Thesaurus build into Microsoft software gives the following similes for risk:

*Chance, uncertainty, peril, gamble, venture, danger, jeopardy and hazard.*

The Concise Oxford Dictionary (1985) defines the term "trust" as:

***noun:** the trait of trusting; of believing in the honesty and reliability of others (Example: "The experience destroyed his trust and personal dignity")*

Another concept that is associated with risk is chance, which is defined as follows The Concise Oxford Dictionary (1985):

***noun:** a risk involving danger (Example: "You take a chance when you let her drive")*

The primary risk systems in the mining industry suggest two categories of risk - pure risk and speculative risk.

**Pure risks** are those risks that offer only the prospect of loss, in other words zero to negative ranges, **while speculative risks** are

those that offer a chance of gain and loss. The latter are sometimes referred to as “operational risks”. This research report focuses primarily on speculative risks.

### **1.5 Minerals Resource Management And Mineral Economics**

Two distinct but closely related fields associated with the optimal extraction of the orebody, include the relatively new disciplines of what has become known as Minerals Resource Management (MRM) and the older perhaps more widely understood fields of Mining and Minerals Economics fields (MME). Practitioners in each of the two fields usually have a good understanding of the required skills base of the other discipline. As the name suggests the Mineral Resource Manager focuses primarily on the management of the company’s most valuable asset namely the ore body, with a strong appreciation of the business aspects of ensuring that profits are returned to the operation. Mineral economists and mining engineers concern themselves mainly with the business aspects of mineral extraction, but will also have a good understanding of the ore body being depleted. The MRM is specifically concerned with the details of the ore body under his managerial control. Both the nature of the ore body and the potential to be added to the operation through the application of good MRM knowledge and practices is essential.

The following definition by Diering (2002) captures the essence of the MRM and his functions.

*'MRM is the subtle art and gentle science of ensuring optimal exploitation of orebodies and ensuring optimal means and effecting positive outcomes. This requires planning and effective management control.'*

MRM is not an entirely new concept and has been practised in isolation sometimes for many years in several mining related disciplines. This is still the case at some operations but its success is hastening the change. The skills of the geologist, surveyor, mining engineer and metallurgist have been combined at many operations to form an MRM department to facilitate synergies and break down the walls of misunderstanding between the various skills and disciplines. The following events in the minerals and mining industry led to the consolidation of MRM as a more clearly defined discipline.

- Static or declining commodity prices;
- Squeezed profit margins;
- Mining houses becoming mining companies with their own listings and accountabilities;
- Primary objective being to increase shareholder value;
- Mines being considered business units in their own right;
- Primary output KPIs for mine management having become: - Contribution, - Break-even price;
- Depleted reserves, lower -grade ore bodies, and more complex ore bodies;



- Skills shortages and increased demands on MRM departments and personnel;
- Production personnel stressed and being required to 'work smarter';
- Availability and application of Computer-aided mine design and scheduling tools - very powerful; and
- "Because we can!"

MRM models are diverse in the extractive industries because of the differences in commodities and the differences in the way business is run for different mineral types. For an example gold producers can send as much as they can produce to the market without influencing the price whereas nickel miners need to be aware of competitor production volumes how the market will react to additional supply. Hence marketing will constitute a significant proportion of the Mineral Resource Management function in market dominated extractive industries.

The following diagrammatic representations of the Mineral Resource Manager functions are taken from different mineral industries. They differ markedly from each other but there are a few common elements that are evident. This commonality suggests that some generic thinking could be transferred between industries but that much will have to be developed within individual industries. Figure 1-7 reflects the different aspects and methods that AngloGold-

Ashanti sees in their MRM functions. It is a model that is continuously updated because the evolution of the MRM functions is still in progress.

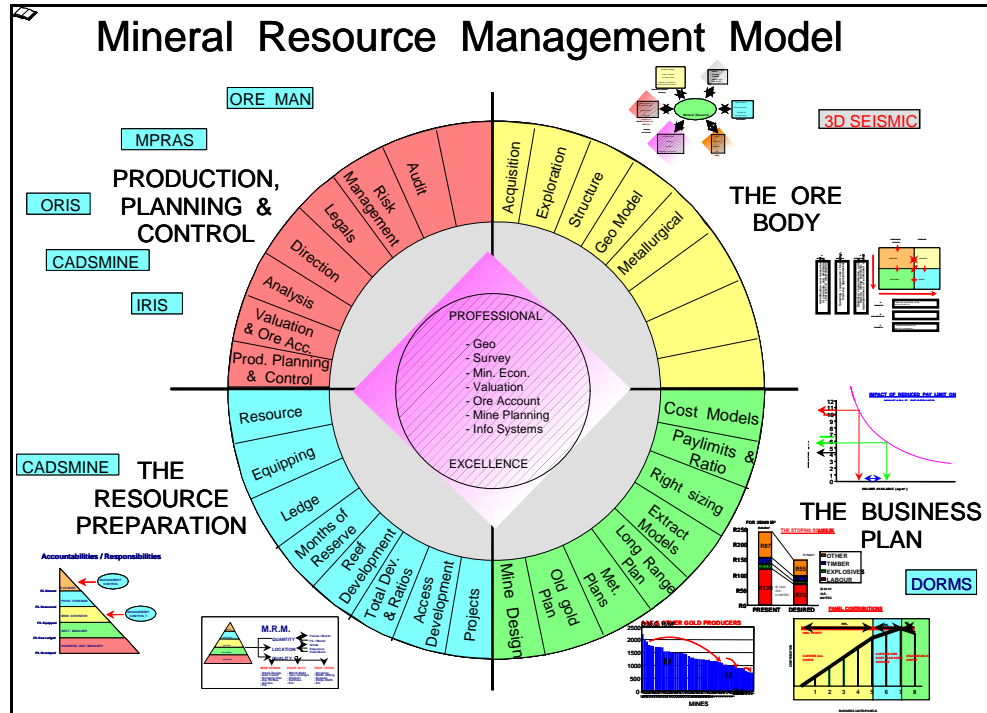
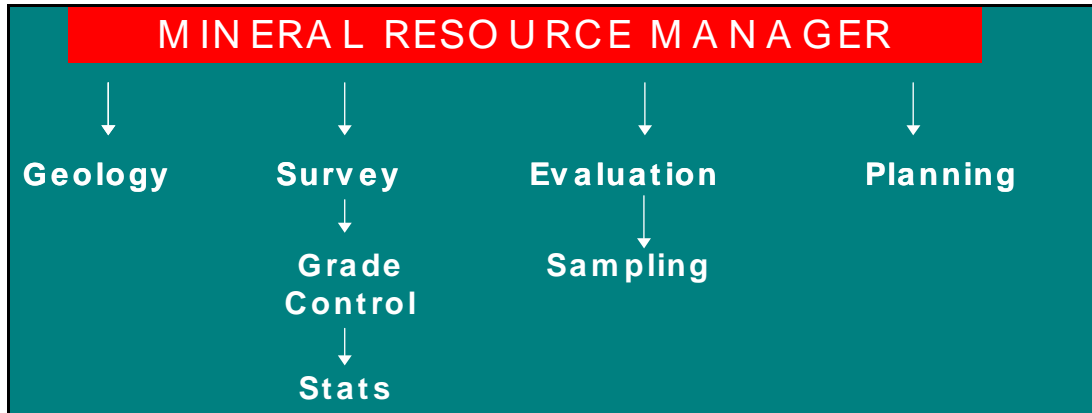


Figure 1-7: The AngloGold-Ashanti mineral resource management model

The MRM model in Figure 1-7 has been split into four quadrants, namely:

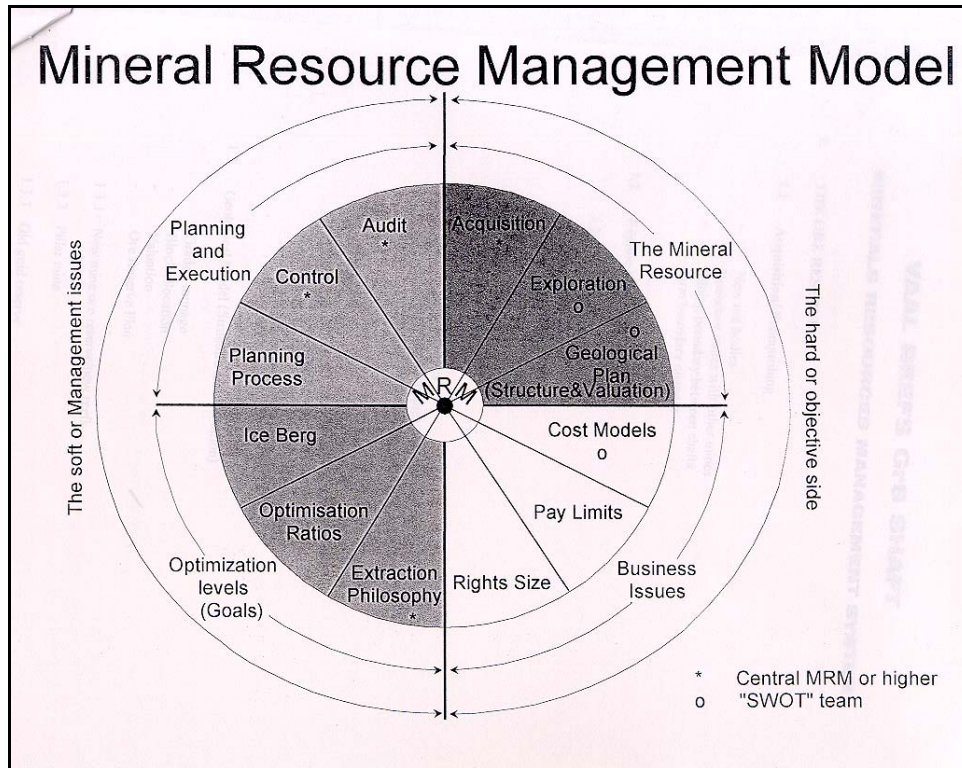
- Tasks that establish the quantity and quality of the ore body;
- The business plan and value aspects of the ore body including the Mining and Minerals Economics skills of the MRM Manager
- The extraction programme; and
- Planning and control systems.

Figure 1-8 shows the typical structure in a Minerals Resource Manager's domain. The synergy achieved because of the removal of the political lines between the different functions is enormous.



**Figure 1-8: Typical structure and functions accountable to the MRM**

Although the origins of Figure 1-9 is unknown, it is believed to be a diagrammatic representation of MRM functions / tasks as required by a coal mining operation.



**Figure 1-9: Diagrammatic representation of MRM functions in the coal industry**

This is similar to the wheel designed by AngloGold-Ashanti in that its quadrants are similar:

- Tasks around the quantification and qualification of the orebody;
- The business side of the equation;
- Optimisation; and
- Planning and control systems.

The second and third quadrants could, to some extent, be considered the Mining and Minerals Economics skills of the MRM

Manager. Figure 1-10 is attributed to an oil mining operation but its origin is unknown.

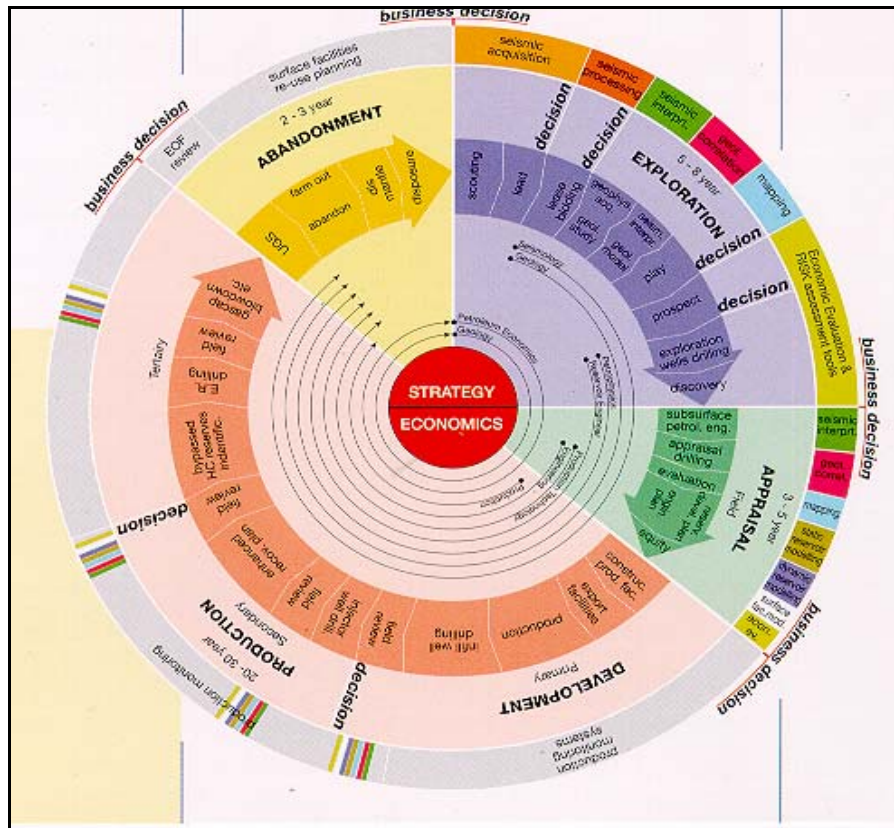


Figure 1-10: MRM functions in the oil industry. (Source unknown)

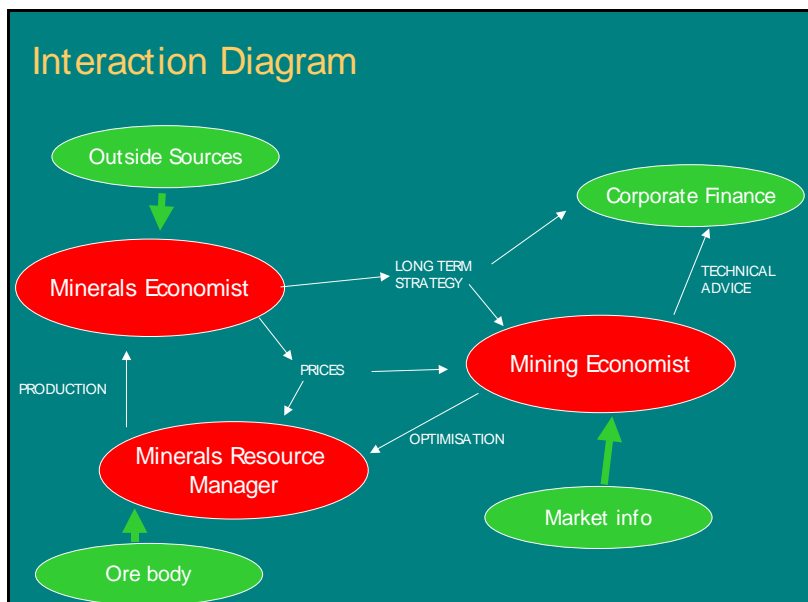
This wheel is designed from a sequential perspective. Again, there is a large common series of elements in the process. Unfortunately, the origin of this wheel is not known.

### 1.6 What is Mining Economics?

The world of the Mining Economist finds its roots in the domain of the MRM, Corporate Finance, Marketing, Accounting, Capital and Project Management disciplines. The practitioner tends to be a Jack-of-all-

trades who has the ability to cross the borders into other skills to ensure that a holistic picture of the business at hand emerges. In the South African mining context Mining Engineers, Project Managers and Minerals Resource Managers perform a significant number of Mining Economic tasks.

Another branch of the economics business has been entrusted to Minerals Economists. These practitioners tend to employ macro concepts including metal prices, metal supply and demand issues. Their key output is a view on the future of the product and the expected price forecasts. Figure 1-11 shows the interaction between Mineral Economist, Mining Economist and Minerals Resource Managers. It focuses on the inputs and outputs for each of the disciplines.



**Figure 1-11: Inter-relationships between the Mineral Economist, Mining Economist and the Mineral Resource Manager**

The Mineral Economist, Mining Economist and Mineral Resource Manager may function across the boundaries of the disciplines and may have different titles. The functions are also significantly different in the different fields / minerals. For example, Minerals Economists may find themselves in a Minerals Marketing department in some minerals, like coal and base metals.

Those charged with the responsibility of marketing the mineral products tend to specialise in that specific field and market. In some minerals the cost of transport is very high and the Marketing Department specialises in transport, contract, and supply and demand functions. The risks involved in these functions fall outside the scope of this research.

Figure 1-12 reflects the level of operation for the different specialists. The Minerals Economist functions at industry level whereas the Minerals Resource Manager functions at operational level.

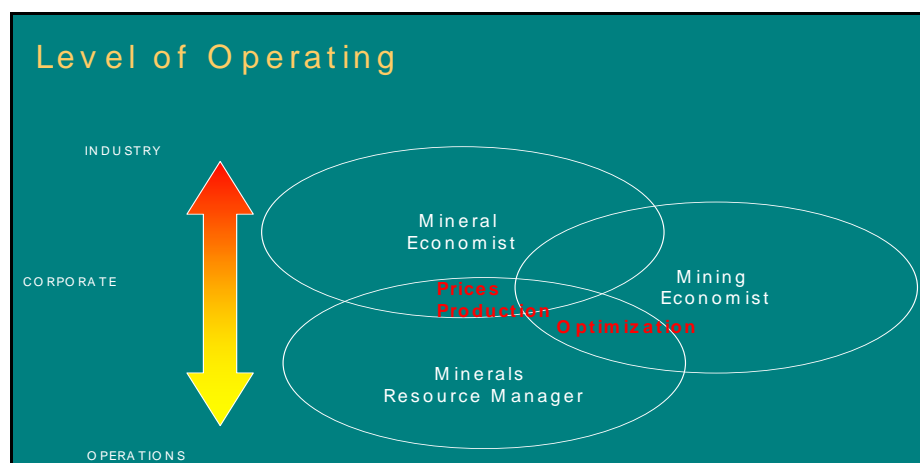
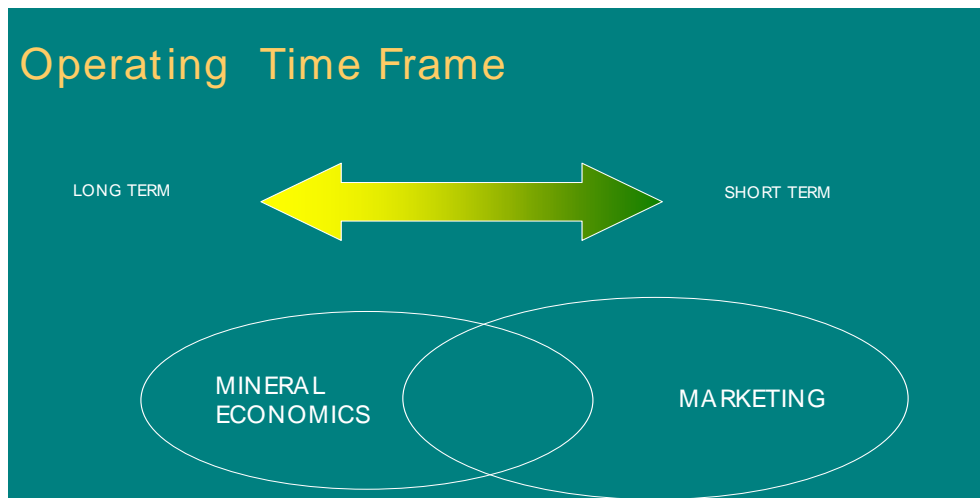


Figure 1-12: Operating level diagram

This research is limited to the lower half of the spectrum as reflected in Figure 1-12 above. Figure 1-13 shows another dimension where a difference exists between the marketing practitioner's and the Minerals Economist's approach – namely the time horizon.



**Figure 1-13: Time horizon**

The Minerals Resource Manager and Mining Economist tend to span the full spectrum of time. The input for the Minerals Economist involved in the sale and marketing of the products is the supply and demand curve. These are the critical tools of his trade. The output is a strategy and a price forecast that becomes the input for the Minerals Resource Manager and the Mining Economist.

A review of a many projects suggests that price of the metals and the grade of the ore body are the main risk parameters and account for a large percentage of project failures. Thus the biggest risk falls outside the domain of the Mining Economist and Minerals Resource Manager. Much research has been undertaken on the mitigation of



risk including hedging and the use of other derivatives in the gold markets.

## 2 STOCHASTIC ANALYSIS

- *Variables and risk*
- *The basic algorithm*
- *The BME*
- *Using the BME to determine risk*
- *Analysis of a gold mine algorithm*
- *@ Risk analysis*

There are a multitude of variables that influence a mining operation, all of variable importance. This study focuses on the financial risk to which a company is exposed. The process considers the key variables in the basic algorithm determine their variability and then finally apply them in a stochastic analysis. The probabilistic outcome shows the most likely achievement as well as the risk associated with it.

### 2.1 Basic Algorithm

The first step is to develop an algorithm that reflects the key first-level variables that leads to profit. The algorithm for a gold mine is simply as follows:

$$\text{Contribution} = [(F/L * F/A * OR * RD * \text{cmgt}) + DG + VG] * MW * REC * GP$$

- F/A = face advance;
- OR = percentage on reef mining;
- RD = relative density;

- Cmg/t = cmg/t value of the ore mined;
- DG = development gold;
- VG = Vamping gold;
- MCF = percentage Mine Call Factor;
- REC = Recovery factor;
- GP = Gold Price; and
- Cost = total cost.

Each of the above first-level variables is a combination of many other variables, which are referred to here as the second and third-level variables. In order to determine the impact of a second-level variable, it is suggested that a new algorithm be constructed for the first-line variable. Determining the combined effect of all the contributing variables will form this secondary algorithm.

For an example, face advance is a combination of the blasting cycle and the advance per blast. The blasting cycle variable could be made up of several tertiary level variables. (i.e. drilling, blasting, cleaning and support). For the purpose of this research, the focus will stay on the basic first level algorithm.

## **2.2 Basic mining equation (BME)**

The basic mining equation (BME) is translated into a spreadsheet model comprising 11 variables shown in Table 2.1.

A BME is defined as a tool that allows simple mining activities, processing activities and sales of the metal produced, to be combined in a single equation that says what the financial contribution of the mining operation for a given period of time will be.

		BUDGET		Planned
1		FACE LENGTH	m	2,656
2	x	FACE ADVANCE	m	9.66
3	=	TOTAL m2	m2	25,647
4	x	ON REEF PERCENTAGE	%	95.54%
5	=	REEF m2	m2	24,502
6	x	ON REEF cmg/t	cmg/t	1,556
7	x RD =	kg GOLD EX STOPES	kg	1,060
8	+	VAMPING kg	kg	40
9	+	REEF DEVELOPMENT kg	kg	20
10	=	TOTAL kg BROKEN	kg	1,120
11	x	MINE CALL FACTOR	%	97.50%
12	x	RECOVERY FACTOR	%	97.20%
13	=	GOLD RECOVERED	kg	1,061
14	x	GOLD PRICE	R/kg real terms	R72,102
15	=	REVENUE	R ('000) real terms	R76,498
16	-	PRODUCTION COST	R ('000) real terms	R60,662
17	=	CONTRIBUTION	R ('000) real terms	R15,836

Table 2-1: Simplified gold BME

The BME is an important tool to evaluate changes in mining variables and identifying risks associated with the mining operation.

### 2.3 Using BME to determine risk

The following analysis is aimed at determining which of the first line variables is most exposed and could most seriously impact on the financial contribution of the mining operation have the most impact from a risk perspective.

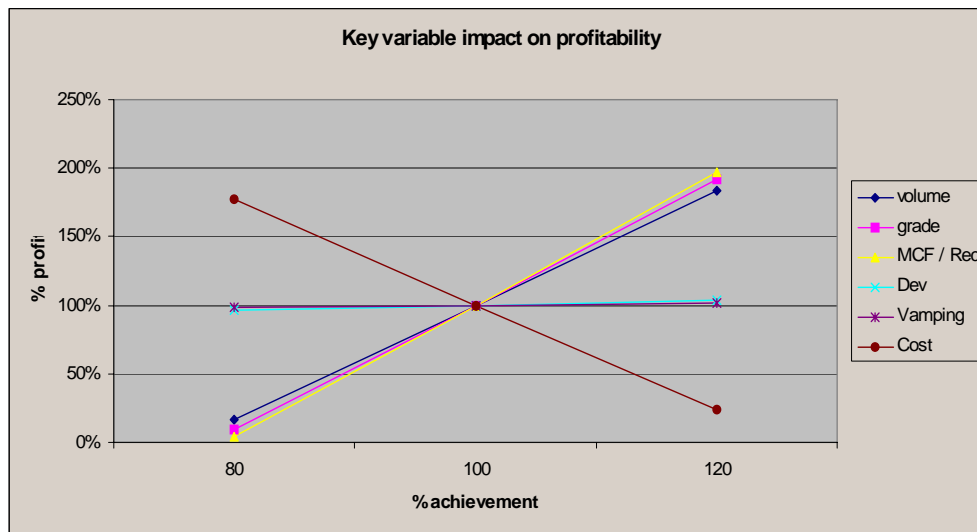


Figure 2-1: Sensitivity" spider chart"

Each first line variable highlighted in Table 2-1 is discussed in turn together with the risks associated with each of these variables. Note that the highlighted items are the variables. The above graph (Figure 2-1) shows the sensitivity of the variables based on a 20% change in both directions. This shows that reducing the face length by 20% results in a decline of 83% of the profit

The following table (Table 2-2) is an example of using the BME to vary variables. It is a derivative of Table 2-1 above. In this case, the

face length was reduced by 20%. It is clear that this swing destroys 83% of the profit.

	BME	BUDGET		Planned	Probability	Likely outcome
1		FACE LENGTH	m	2,656	80%	2,125
2	x	FACE ADVANCE	m	9.66	100%	9.66
3	=	TOTAL m2	m2	25,647	80%	20,518
4	x	ON REEF PERCENTAGE	%	95.54%	100%	95.54%
5	=	REEF m2	m2	24,502	80%	19,602
6	x	ON REEF cmg/t	cmg/t	1,566	100%	1,566
7	x RD	kg GOLD EX STOPES	kg	1,060	80%	848
8	=	VAMPING kg	kg	40	100%	40
9	+	REEF DEVELOPMENT kg	kg	20	100%	20
10	=	TOTAL kg BROKEN	kg	1,120	81%	908
11	x	MINE CALL FACTOR	%	97.50%	100%	97.50%
12	x	RECOVERY FACTOR	%	97.20%	100%	97.20%
13	=	<b>GOLD RECOVERED</b>	<b>kg</b>	<b>1,061</b>	81%	<b>860</b>
14	x	GOLD PRICE	R/kg	R72,102	100%	R72,102
15	=	REVENUE	R ('000)	R76,498	81%	R62,016
16	-	PRODUCTION COST	R ('000)	R60,662	100%	R59,397
17	=	CONTRIBUTION	R ('000)	R15,836	17%	R2,619
		<b>RD Factor = 3E-05</b>				
		Fixed	90%	54540		
		Variable	10%	6060		
		Base	25600	20,518		
		Base cost	60600	59397		

**Table 2-2: Risk adjusted BME**

In order to determine the risk associated with each variable, data from a mining operation were applied in the analysis. The statistical function of Excel and Palisade @Risk software were used to analyse

each variable. The statistics associated with each variable were subjected to a Monte Carlo simulation in order to evaluate the overall risk associated with the mining operation as reflected in changes in the overall financial contribution.

### **2.3.1 Face length worked (F/L)**

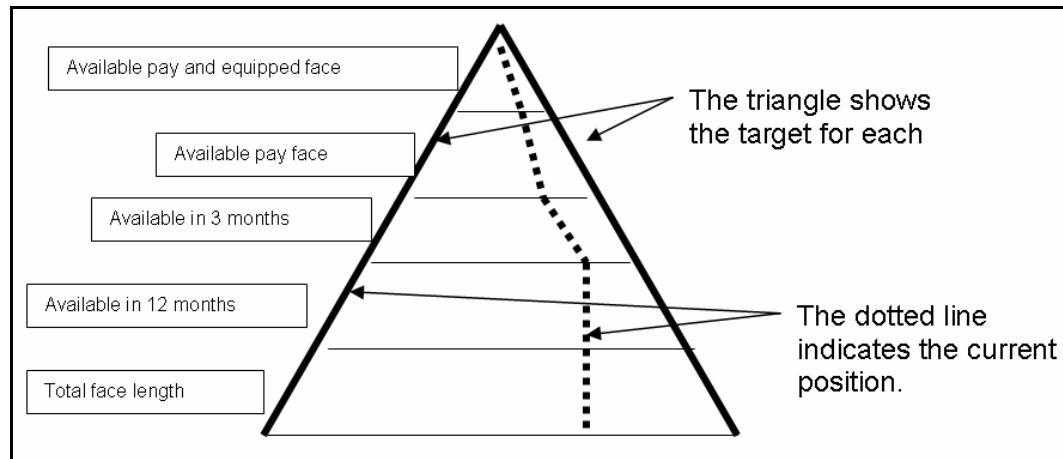
The face length worked is a design parameter whose risk is a function of the quality and quantity of face length available. The quantity is a function of the design, mine layout and rock mechanics while; the quality is a function of the variability of grades and payability of the reef. Payability in turn is a function of cost of extraction and price of gold.

In order to mitigate this risk, a detail analysis of development and available face length is required. Each development end should be prioritised and monitored. It is suggested that a critical path analysis (CPA), be conducted in order to improve the development planning. Latest starting dates and critical paths should be determined for each development end considering the risk in the mining methods as well as the local nature of the ore body. For example, in a mine with a low profitability and low confidence in predictability of grade, a higher percentage of proven reserves is essential. In more geologically complex areas longer lead-times and more exploration will be required, in order to mitigate the risk of not opening the reserves in time. It is also important that the investments in reserves be brought

to account as soon as possible. It is imperative that the balance between risk and return is maintained.

The probability of achieving a prescribed face length needs to be established through the use of control systems like the 'iceberg' and 'candy bars'.

The iceberg is the ratio of pay face to total face. The dotted line shows the current position.



**Figure 2-2: Iceberg" diagram**

The above diagram (Figure 2-2) should be designed according to the mining method and nature of the ore body. The ratio between "available pay and equipped face" and "available pay" face is determined by the equipping programme and the rate of face depletion. The ratio between "available pay face" and "total face length" is determined by the percentage payability. The development of the iceberg is not a precise science and the approximate ratios will



be determined over time. Changes in economic assumptions will also affect the above ratios.

The diagram below (Figure 2-3) is an example of the 'candy bar' and should be used to monitor the quality of the ore-body, whereas the 'iceberg' (Figure 2-2) is used to monitor the quantity of face length.

The first column shows the current position of the reserves, whereas the second column shows the required reserve position that will facilitate the selected optimised plan.

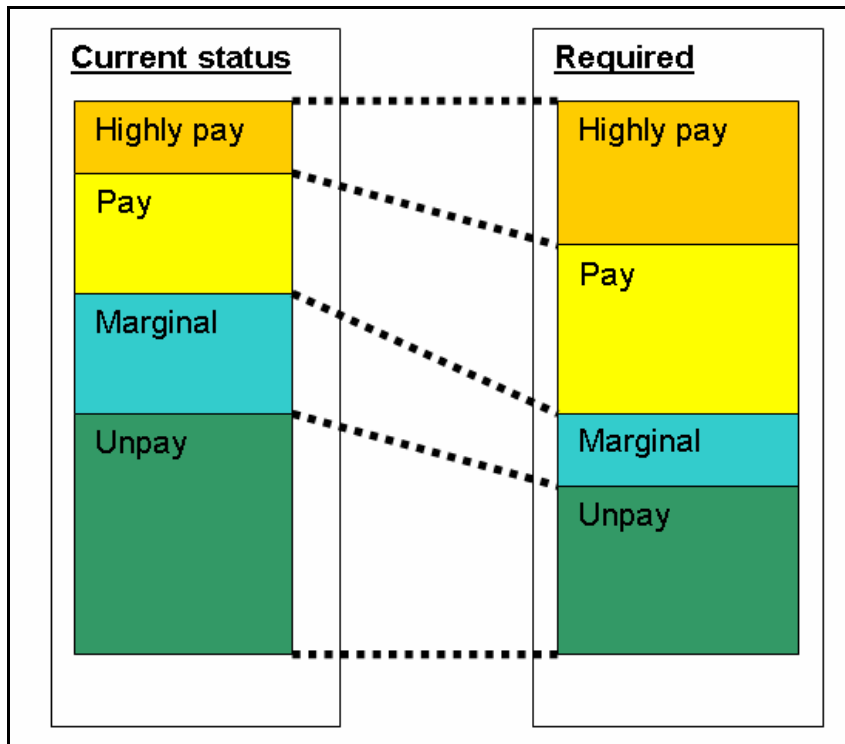


Figure 2-3: "Candy bar "indicating face length distribution.

The above diagram should also be designed according to the mining method and nature of the ore body. The diagram is constructed

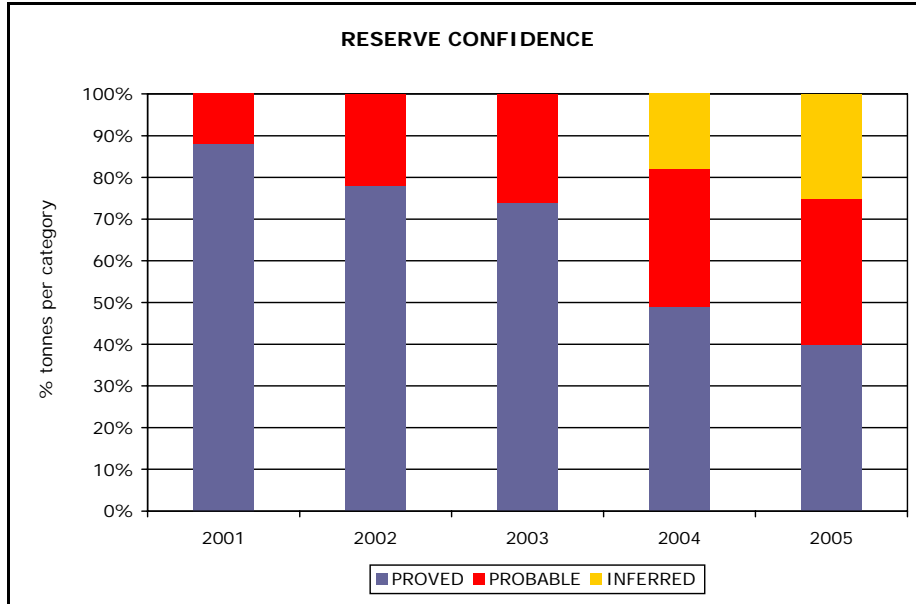
utilising historical data, the ore body signature as well as economic cut-offs.

These controls will provide an indication of the type of development program required to meet the minimum financial contribution of the mine. The deviation from the required standard could be used to quantify the risk associated with face length. For example, if there is less face length available than required, as indicated in the above sketch, there will be a risk associated in achieving the correct mix of face length. On the other hand, an excess of available pay face length will reduce the risk associated with the face length variable.

Mining operations generally have poor levels of control on the face length. Moreover, many mining operations do not mine at the designed because of the lack of proven reserves. The reason offered for the poor production rates problem was management cut backs on development over the last few years, as their margins were low. The net result of inadequate controls is that unrealistic production rates are extrapolated at the current levels and when the expectations are not realised, this can affect the life of mine.

As Mineral Resource Manager, you must give an understanding of the cost-volume curve and the ore body signature that will prove that significant value is destroyed in the process. The optimisation of the orebody extraction should be the number one KPA. This optimisation process is discussed in the next two chapters. Ensuring

that sufficient tonnes of ore at the right grade are delivered to the mill is the most important factor in maintaining the financial contribution.



**Figure 2-4: Graph showing resource / reserve confidence**

Strategic design indicates that the first 3 years in the life of a mining operation should come from proven and probable reserves as shown in Figure 2-4 above.

The comparison between planned and achieved face length is shown in following graph (Figure 2-5).

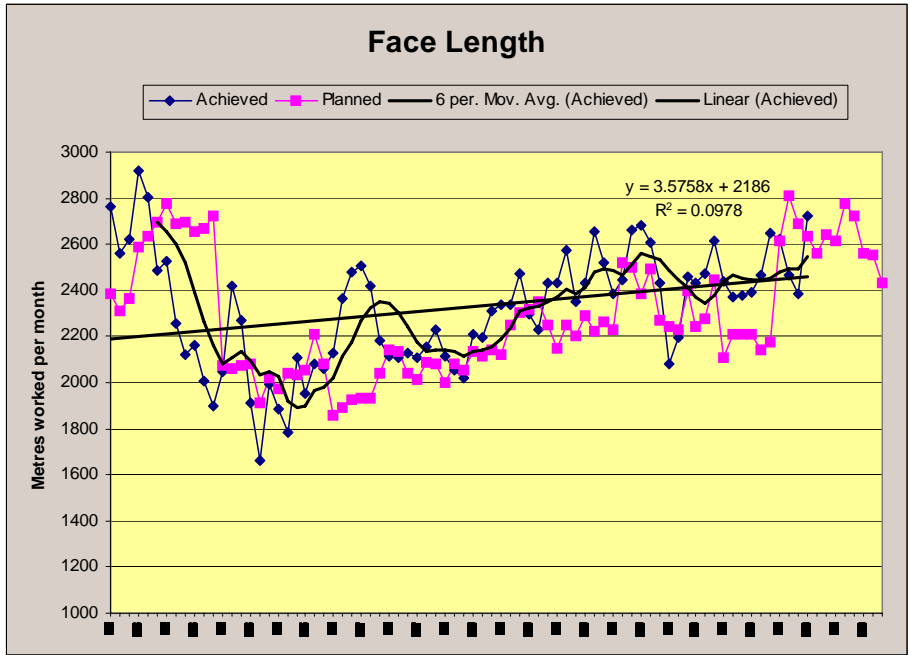


Figure 2-5: Average face length per month analysis

The regression line through the six-year period indicates that there has been a consistent increase in face length worked.

	Face length worked(m)	Stope Face Length planned (m)
<b>Mean</b>	2324	2256
<b>Standard Deviation</b>	253	242
<b>Range</b>	1252	953
<b>Minimum</b>	1665	1855
<b>Maximum</b>	2917	2808
<b>Count</b>	76	76

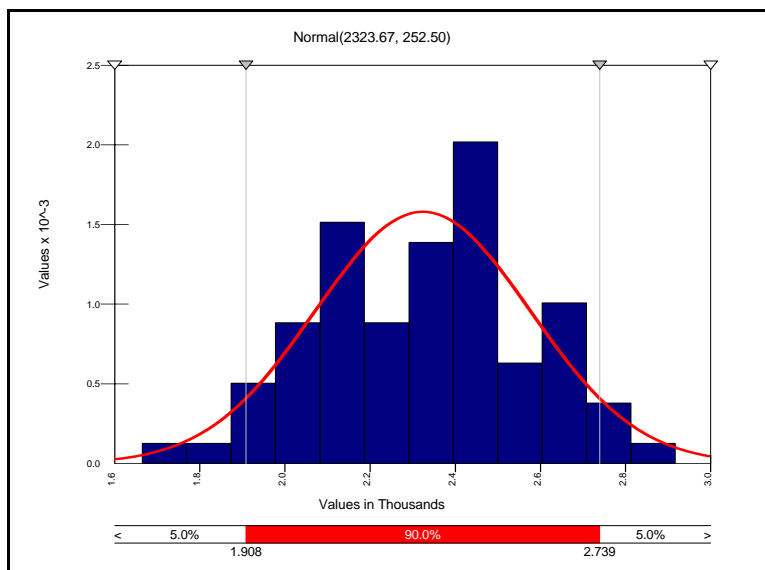
Table 2-3 Face Length Worked

From the above statistics, it appears that over a period of 76 months the planned face length (2256) is lower than the actual face length worked (2324).

The face length worked should stay fairly constant on a mine where a certain production level is maintained. The correct way to determine the face length is to physically measure it on plan.

The wrong way to determine the face length planned is to divide the square metres mined by the planned face advance. The planned face advance is often not derived from the plan but is estimated as a strategic target. It is suggested that controls and estimation techniques be improved, as it is the first critical variable in the mining operation. It is important to do have confidence in listed districts of face length so that the risk associated with this variable can be managed or mitigated.

The following graph (Figure 2-6) indicates a normal distribution for face length as the best estimate. The historical data is reflected in the histogram.

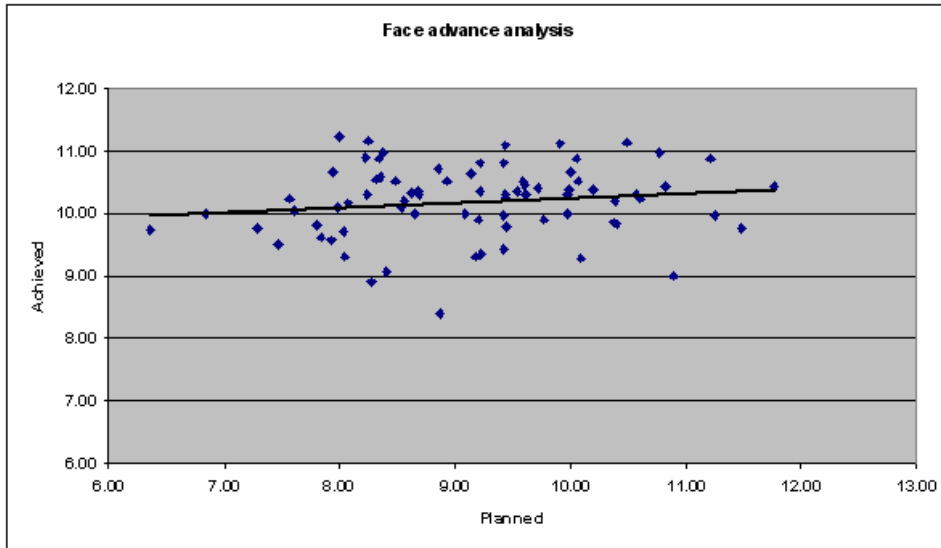


**Figure 2-6: Distribution of face length**

In reality, the risk is a function of the availability of proved reserves. The less the reserves, the less the flexibility, the higher the risk associated with achieving face length planned. The Iceberg and Candy bars could be used to estimate a risk factor. For this exercise however, a normal distribution with a mean of 2323m and a variance of 252m will be used in the @ Risk simulation.

### **2.3.2 Face advance (F/A)**

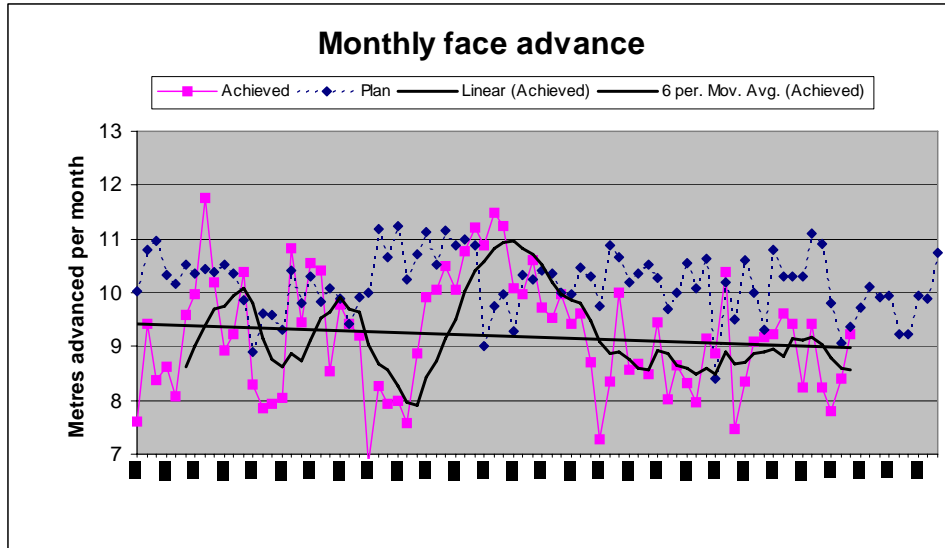
The face advance is a function of the number of times a face is blasted and the advance per blast. Advance per blast is a function of drill steel length, the quality of drilling, as well as the explosive type and efficiencies. The advance per blast could be determined and / or measured for each team or working place. The ratio between the advance per blast achieved compared to the planned advance per blast will result in the probability of the advance required. Very few mining operations have comprehensive control systems that monitor, control and determine face advance.



**Figure 2-7: Scatter plot of face advance planned versus achieved.**

The number of times the face is blasted is a function of the adherence to the mining cycle. The cycle will depend on the ability to adhere to the designed programme. Historical data can be analysed in order to calculate the probability achieving the planned cycle. The ratio between the number blasts achieved versus the number blasts planned as per the blueprint could be a proxy for the blasting probability. Figure 2-7 above shows a typical scatter plot between face lengths planned and face length achieved. Note the wide spread of points.

The regression line over the 28-month period (as seen in Figure 2-8) shows a decline in face advance. However, the correlation between the fitted line and the actual data ( $R^2=0.0148$ ) is too poor to make a significant deduction from this linear fit line.



**Figure 2-8: Face advance analysis**

The following statistical analysis shows that the face advance (10.16m) planned is almost 1 metre more than what is achieved (9.16m). This again bears out the discrepancy between what is planned and what is actually achieved in a mining situation.

	Face advance achieved (m)	Face Advance planned (m)
<b>Mean</b>	9.16	10.16
<b>Standard Deviation</b>	1.12	0.57
<b>Range</b>	5.41	2.84
<b>Minimum</b>	6.36	8.40
<b>Maximum</b>	11.76	11.24
<b>Count</b>	76	84

**Table 2-4: Face Advance**

Figures 2.7 and 2.8 shows that there was probably very little constructive basis for face advance prediction.



The following analysis (Figure 2-9) compares the relationship between these two mining parameters and indicates a strong inverse relationship between the two variables.

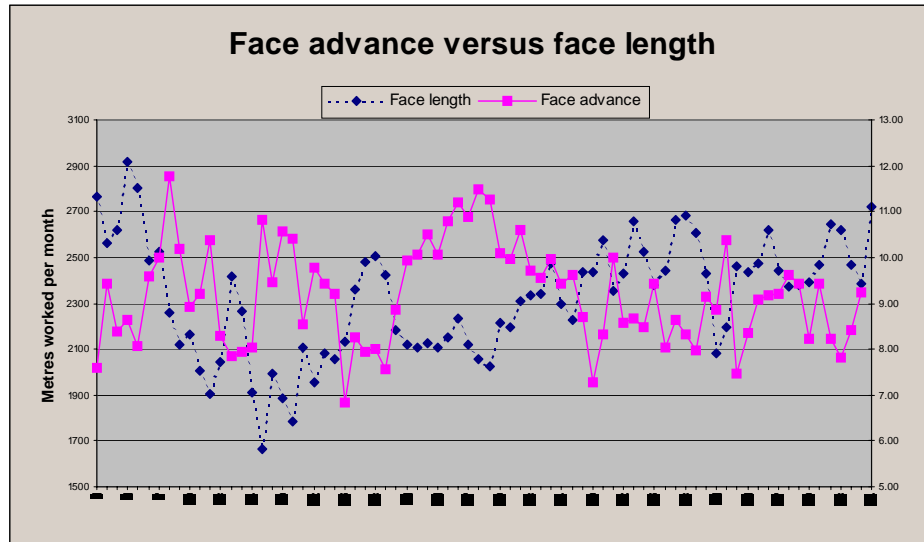
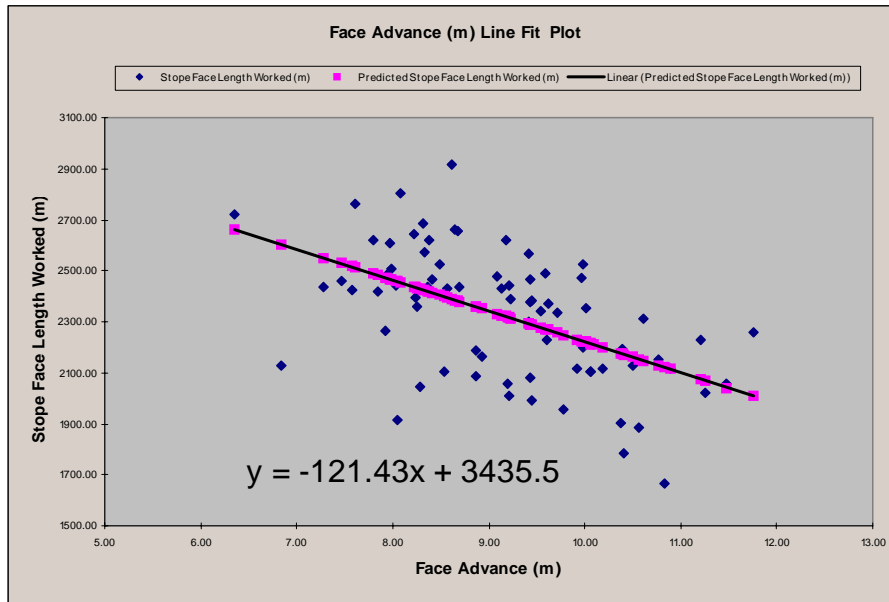


Figure 2-9: Face advance versus face length analysis

In reality there should not be such a strong inverse relationship between the two variables, as a standard face length is allocated to each team, and the number of teams does not fluctuate significantly on the month-to-month basis. Face length worked is thus expected to stay far more constant where is face advance is expected to fluctuate. It is suggested that the above mine verifies its methodology to determine face length and face advance. It appears that the current statistics are not reliable.



**Figure 2-10: Face length versus face advance regression**

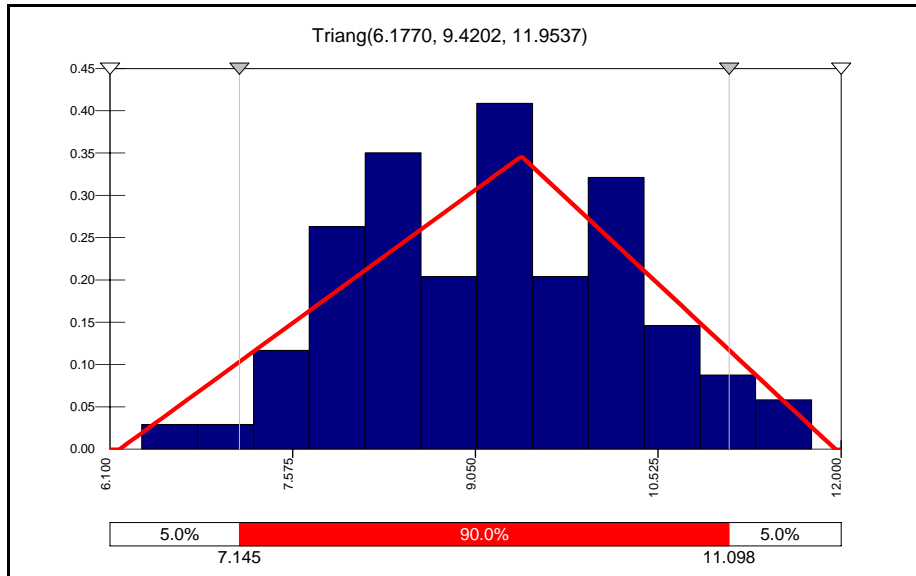
The above graph (Figure 2-10) shows a different analysis of the same two variables but in a scatter plot format. The line down the centre of the graph shows the BestFit correlation in order to predict the face advance or the face length from the other variable. This could be used if the statistics are accurate.

The BestFit line **face length = (-121\*face advance+3435)** could be used to predict the face length required. The face length planned is directly linked to the 'candy bars' and 'icebergs' as described in the face advance section.

In summary, face advance is not really a function of face length but rather of the cycle and the ability of the team to achieve the cycle. Establishing an appropriate face advance for any stope should be a function of face length, cycle, and drill steel length. The use of these

variables would provide a better estimate and would improve the control systems around face advance.

The dataset was further analysed to determine the best distribution of the historical data to use in the stochastic analysis. (Figure 2.14).



**Figure 2-11: Distribution of face advance**

The above graph (Figure 2-11) was generated using Bestfit software and the triangular distribution (with a minimum face advance of 6.2m, an most likely advance of 9.4m and a maximum of 11.9m) was used for the @ Risk Monte Carlo simulation.

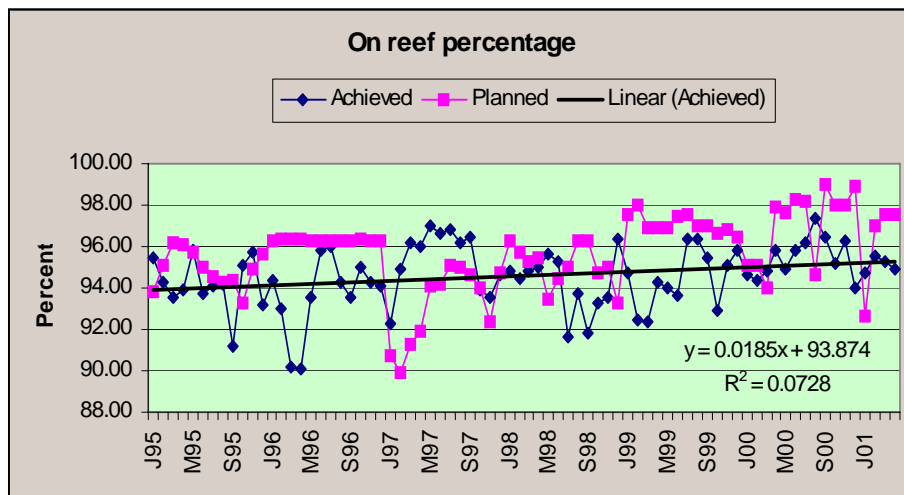
### **2.3.3 Percentage on reef mining (OR)**

The quality of the production needs to be assured through a number of parameters. The first of these qualitative variables is the percentage on-reef extraction. (The standard format used in the industry is the off-reef percentage. The on-reef percentage is simply

the complement to 100%. (i.e. if the off-reef percentage is 5% the on-reef percentage will be 95%).

This factor is often a historical number decided on by the surveyors that is kept constant for the year, where in reality it is a function of the geological complexity of the area mined.

The following graph (Figure 2-12) shows the on-reef percentage compared to the planned statistics.



**Figure 2-12: On reef analysis**

The slight increase in the on-reef percentage for the 19 quarters from January 1995 to January 2001 is shown in Figure 2-12. Considering the linear fit curve on the historical achievements, it appears that there has been an improvement in the last few years. However, the correlation ( $R^2 = 0.0728$ ) between the actual achievements and of the regression line appears to be fairly poor.

The following table shows the statistical analysis of the on-reef variable.

	Actual (%)	Planned (%)
<b>Mean</b>	94.59%	95.63%
<b>Standard Deviation</b>	1.51%	1.83%
<b>Range</b>	7.3%	9.1%
<b>Minimum</b>	90.1%	89.9%
<b>Maximum</b>	97.3%	99%
<b>Count</b>	76	76

**Table 2-5: On-reef analysis**

The difference between the planned and achieved on-reef percentage only differs by 1 percentage point. Overall there is little difference in these parameters.

It is important though that the necessary control systems are in place and that the off-reef mining is planned panel-by-panel and linked to the geological model. As a best practice, some operations ensure off-reef mining is approved during planning meetings. Penalties are included in the bonus scheme to address an authorised off reef mining. Good grade control practices are critical as far as this variable is concerned.

In summary, this variable mostly falls within a fairly narrow range of around 3%, resulting in minimum risk exposure.

It is important to note that the off-reef mining generates cost but no revenue. In addition, it displaces good ore from the mill if the operation is running on the upper limit of any capacity constraint.

To facilitate a stochastic analysis, a normal distribution (with a mean of 94.6% and a standard deviation of 1.5%) was fitted to the data and is reflected in Figure 2-13 below.

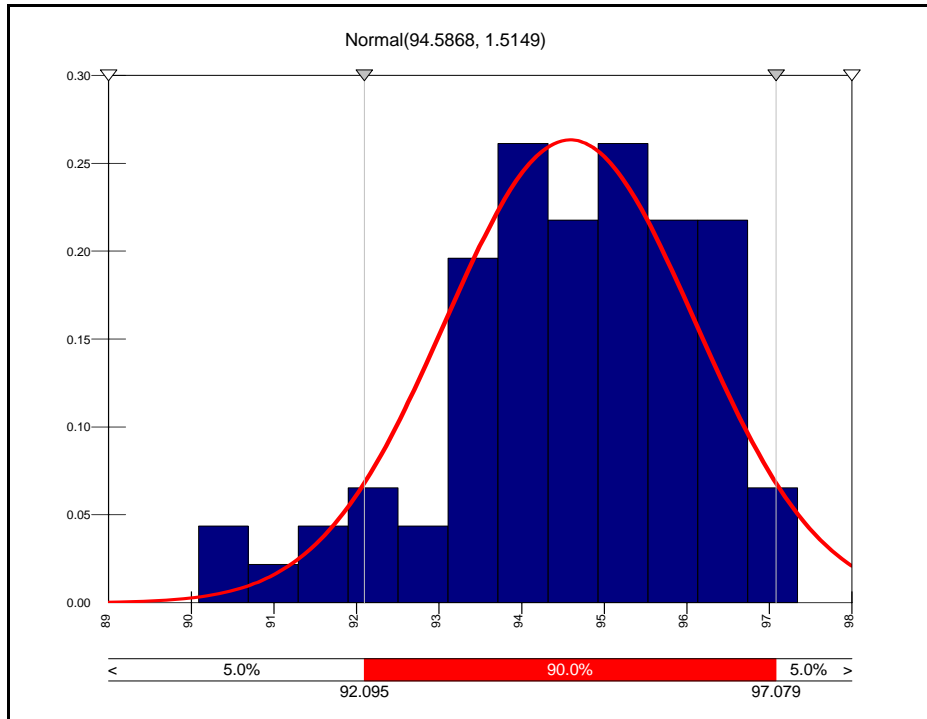
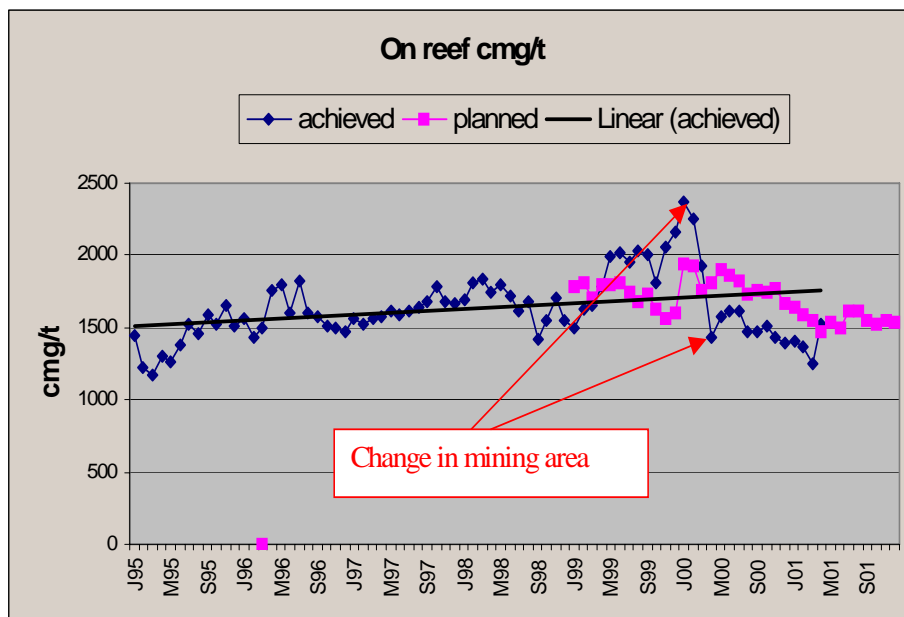


Figure 2-13: Distribution of off-reef

#### 2.3.4 Accumulation value of the ore mined (cmg/t)

The accumulation value is product of the grade (g/t) and stoping width (cm). The grade will change as different areas are mined. The statistics of the past may not be valid for the new part of the orebody that will be mined in the future. However, an analysis of past achievements versus the planned target does to some extent indicate one's ability to achieve set targets.

Unfortunately, the geostatistical parameters of the reserves were not available for this study. It is suggested that the evaluator who supplies the mean grade and variance for each mining block provide the valuation of a mining area, in terms of grade. These individual values are then entered into a Monte Carlo simulation so that the average mean and variance for the total area mined can be obtained. However, for the sake of this research, an analysis of the historical data was used, as it is all that was available. The following graph (Figure 2-14) reflects the on-reef cmg/t. It is unfortunate that the planned grade is only available for two years.



**Figure 2-14: Grade analysis**

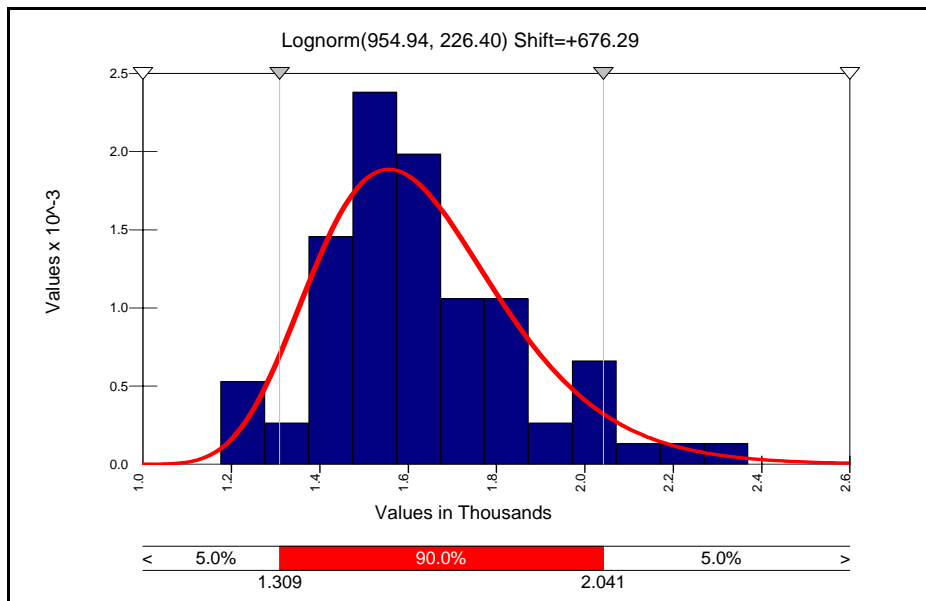
From the above (Figure 2-14) it is clear that there was a change in the grade between September 1999 and May 2000. This is as a result of mining moving into a low-grade area. This change is likely

to continue in the short-term. This mine has insufficient proven reserves and is now paying the price of the low levels of development achieved over the last few years.

The following analysis of the on reef cmg/t was conducted over the full population of grades, as well as the last 28 month's to account for the low grade area mining as explained above. The last 28 month's data parameters are used for the @ Risk analysis.

	<u>Achieved 6</u> <u>years</u>	<u>Planned 2</u> <u>yrs</u>	<u>Achieved 2yrs</u>
<b>Mean</b>	1631	1735	1721
<b>Standard Deviation</b>	230	115	304
<b>Range</b>	1193	470	1123
<b>Minimum</b>	1176	1468	1246
<b>Maximum</b>	2369	1939	2369
<b>Count</b>	<b>76</b>	<b>28</b>	<b>28</b>

**Table 2-6 Analysis of on-reef cmg/t**



**Figure 2-15: Grade distribution**



From the historical grade analysis (Figure 2-15) the best distribution is reflected by a log normal distribution with a mean of 954 cmg/t and a standard deviation of 226 cmg/t.

The variable of grade has by far the biggest impact on the BME. This indicates that the monitoring and control systems associated with grade issues should be of the highest priority.

### **2.3.5 Grade control issues**

The factors to be considered are discussed in detail in the optimisation section (Chapter 3). This selective mining capacity refers to the concepts of regression, variability, predictability and continuity.

There are two sources of error associated overall grade, namely errors in prediction and errors that occur during production, for example, mining sequence (i.e. planned not mined and mined not planned). In order to ensure proper control, an analysis of both errors that occur during evaluation and production has to be established.

The presence of discrepancies in estimation or evaluation was originally highlighted in the calculation referred to as the **block factor**. This factor compared the grade estimated with the grade achieved from a specific area or block of ground to be mined. The use of this factor has lost favour with the geostatisticians, because the block factor can only be calculated when the whole block is

mined out. With much more selective mining taking place, this measure is being displaced by alternate controls. The MCF is also a good proxy to indicate if problems exist and is discussed in Section 2.3.7.

The control on grade in the mining sequence is done through the **daily blast control** systems and the monthly reconciliation systems. A carefully controlled daily blast system that monitors the face advance and tonnage from different areas is critical for monitoring the production progress. Such a daily blast control system, to some extent; is a proactive system to rectify grade and planning problems up front. These systems are available at most operations and are used with various levels of success. They are often linked to the lost blast system (which endeavours to analyse the real cause of losses), with great effect. The biggest problem is around the integrity of data recorded in the lost blast system and the lack of analysis of the key contributors to losses.

The longer-term controls are seated in the **planning reconciliation** method. The "planned not blasted" versus the "blasted not planned" is the discrepancy between what is in the planning target and what is actually achieved underground. Whilst the planned and the actual outcome should be identical they rarely are, for a number of reasons. These may include:

- Unforeseen geological features;

- Poor planning;
- Poor discipline;
- Over achievements.

This analysis is usually conducted once a month. Variances are recorded and the low correlations between what is planned and what happens in reality are remarkable. Planning variances of up to 50% are not uncommon. The reasons for these variances are linked to inflexible planning systems, poor discipline, and poor planning techniques. Planning reconciliation is a field of study in its own right. The consequence of such a poor correlation is that it reduces the company's ability to manage their profitability.

### **2.3.6 Development gold (DG)**

The amount of gold recovered (referred to as development gold), is related to the amount of on-reef development. The gold produced by development is usually not significant in any operation that has reached design capacity as the development reduces.

The analysis is very similar to that for the previous variables and was not done for this variable, as in this specific case it is unlikely that it makes a significant difference to the profit and the decision making process. However, there are several operations were impact is significant and including this in the stochastic analysis is beneficial

### **2.3.6.1 Vamping gold (VG)**

Gold from vamping is a function of gold that has been lost in all areas through bad or unavoidable mining practices. A new mine usually does not have significant quantities of ore locked up in all areas. It is also unlikely that significant sustainable tonnage of locked up ore would be available in long-wall mines, mainly as a result of the backfill practices. Some of the older scatter operations may have significant lock-ups and cleaning up in these old areas may contribute additional gold that he may become a significant part of the revenue flow.

The analysis will be very similar to that for the previous variables and has not been done for this variable, as it is unlikely that it will make a significant difference, as this is a new long-wall mine.

Their recovery associated with vamping gold is as low as 1% on a new long wall mine and as high as 60% in some of the older "scattered" shafts nearing the end of its life.

### **2.3.7 Mine call factor (MCF)**

The mine call factor is the ratio expressed as a percentage of the specific product called for by the mines measuring methods to the specific product accounted for, inclusive of residues. The purpose of the mine call factor is to determine how much of the product was lost in the extraction programme.

This variable is probably the most discussed and most misunderstood variable in the whole mining industry. It is used extensively by the gold mines and is recently being considered at some platinum mines. A multitude of errors and inaccuracies with regard to the estimated gold content of the reef, including underground losses, inaccurate grade prediction and many more sources of losses are hidden in this factor.

There are essentially two sources of gold losses, namely real gold losses and apparent gold losses (gold that was not there in the first place). The apparent losses are usually a function of the sampling, assay, and valuation process. The real losses are attributed to losses in the back areas of existing stopes, along gullies, in ore passes, in haulages, in tips and in the shaft as well as losses in the plant. The MCF is often split into a plant call factor (PCF) and a shaft call factor (SCF) in order to more clearly apportion losses to different segments of the mining operation. This requires a go-belt sampler to be installed at the shaft head. This sampling device takes samples off the conveyor belt at regular intervals in order to determine more accurately the gold content of primary crusher ore about is sent to the plant. The SCF is thus a reconciliation of the product as determined by the mine's measuring methods and the product estimated over the conveyor belt system. The accuracy of the go-belt sampling has often been questioned. However, the size and frequency of the samples are determined from a detailed statistical

analysis. Moreover, if values are compared from month to month, the comparison is relative, as the go-belt sampling method stays constant. The use of the go-belt sampler as well as the SCF facilitates better controls where multiple shafts uses the same plant.

The plant call factor is the relationship of the product as estimated over the go-belt system to the product accounted for by the plant (recoveries and estimated residues).

Controls around the MCF are the function of the 'grade control department', and these controls include checking the quality of the sweepings to ensure that no blasted ore is left behind. The grade officers also have controls around old areas and vamping operations in worked-out areas.

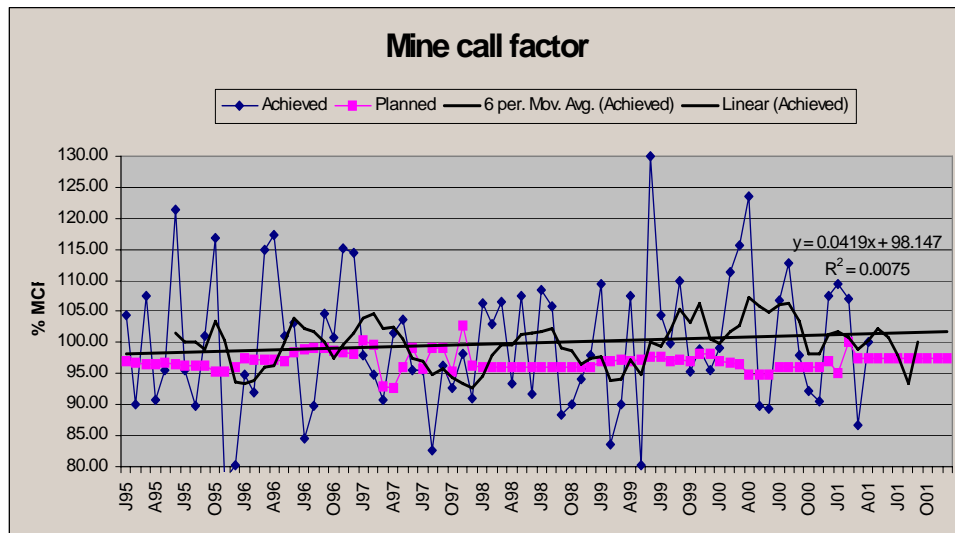


Figure 2-16: MCF analysis

The above graph (Figure 2-16) reflects the MCF over the six-year period and the following table the statistical analysis.

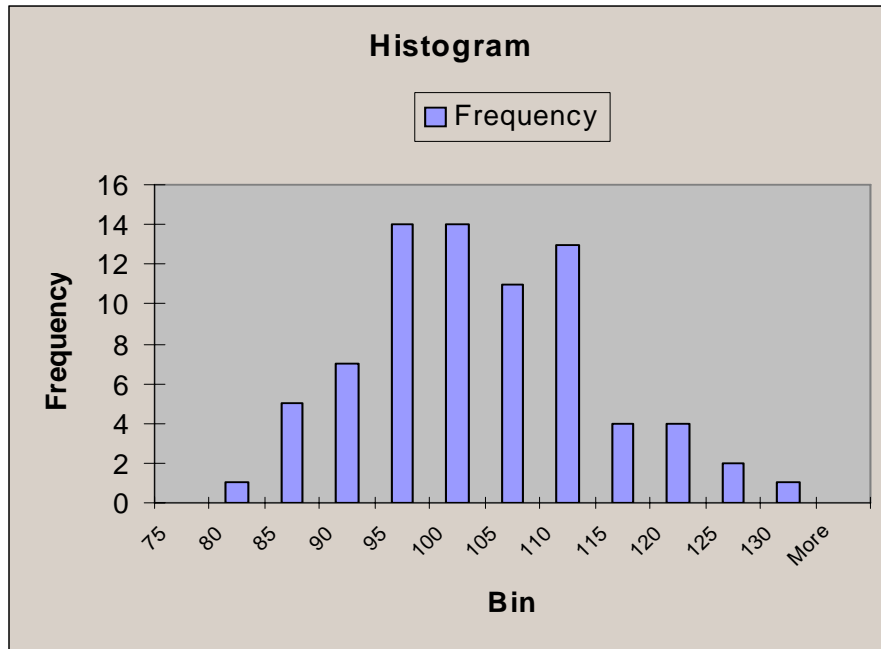
The mine has achieved a 100% MCF over a period of six years, which is three percentage points better than the target. Considering that apparent and real losses are expected, it is unlikely that a cumulative 100% MCF could be expected. This suggest that the grade is probably underestimated, resulting in an ‘under calling’. In addition, a range of 52% is very high and suggests that controls need to be improved; as such, a wide range of variation is disastrous for the profitability of the operation.

The six-month moving average shows a 7% variation around the mean and appears to have a cyclical pattern that cannot be explained.

	<b>Actual</b>	<b>Planned</b>
<b>Mean</b>	99.7	96.9
<b>Standard Deviation</b>	10.68	1.49
<b>Range</b>	51.9	10.2
<b>Minimum</b>	77.9	92.5
<b>Maximum</b>	129.8	102.8
<b>Count</b>	76	84

**Table 2-7: MCF Analysis**

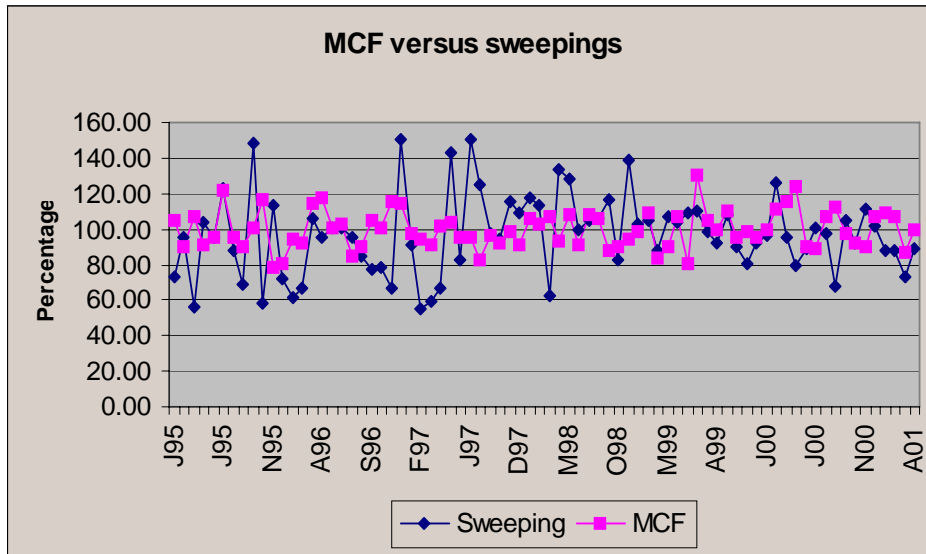
This histogram (Figure 2-17) is included to show the spread of the MCF as distributed over a 5% bin size distribution.



**Figure 2-17: Histogram of monthly MCF**

One of the first components of the MCF is the sweepings percentage. The following graph shows the correlation between the MCF and sweepings. It appears if there is some lag between the two variables. This is probably a function of the time that the ore takes to be transported and treated.





**Figure 2-18: Sweepings versus MCF**

The following table shows the statistical analysis of the sweepings, and the large fluctuations that appear suggest an inconsistency in the emphasis on getting the ore to the plant. Moreover, it also explains the large variations in the MCF. The fact that the MCF is averaging at 100% and the sweepings at 96% suggests that the losses are hidden by an underestimation in grade. These controls need serious attention.

<b>Mean</b>	96.86
<b>Standard Deviation</b>	22.29
<b>Range</b>	94.75
<b>Minimum</b>	55.53
<b>Maximum</b>	150.28
<b>Count</b>	76

**Table 2-8: Statistical Analysis of sweepings**

The following graph (Figure 2-19) shows the Bestfit analysis of the MCF data. The lognormal curve was selected and the massive

standard deviation reflects the variations in the MCF. This massive variation will be translated into the 'bottom line'.

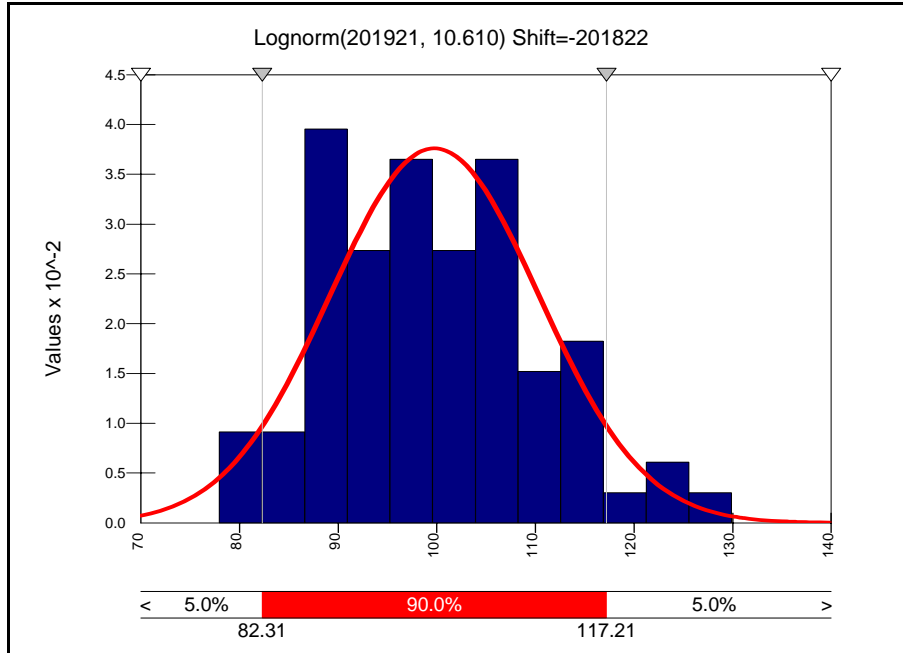


Figure 2-19: Distribution of MCF

### 2.3.8 Recovery factor (RF)

The following (Figure 2-20) variable shows a continuous improvement from 96% to 98%. The monthly variations are relatively small as can be seen in the descriptive statistics, which in turn suggests a low financial risk.

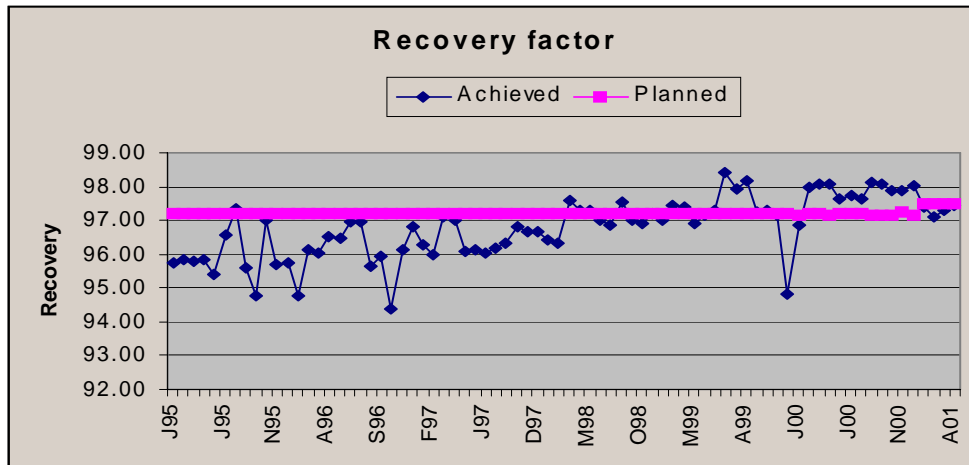


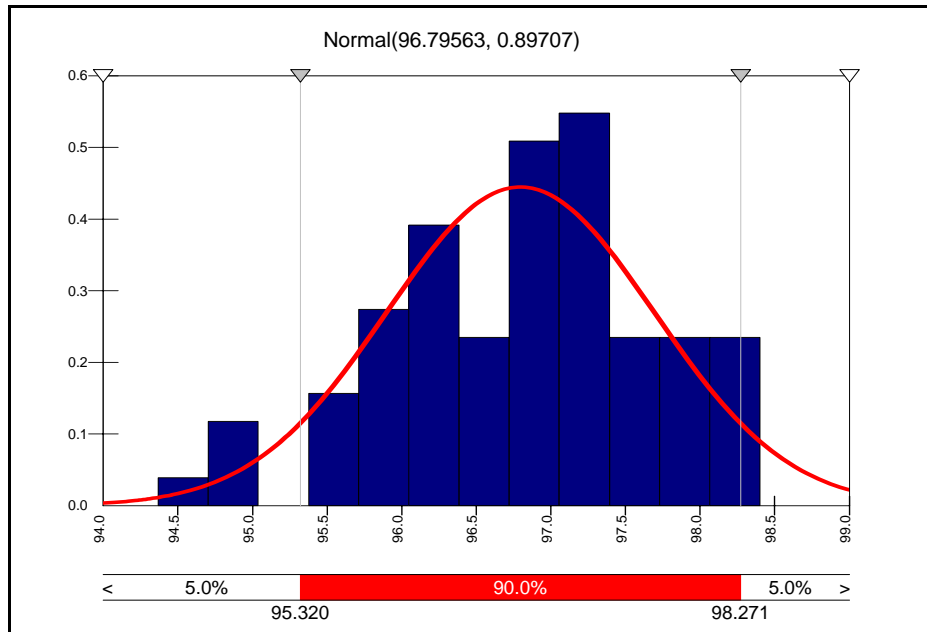
Figure 2-20: Analysis of recovery factor

The following table reflects the descriptive statistical analysis for the recovery factor.

	Actual	Planned
<b>Mean</b>	96.7	97.2
<b>Standard Deviation</b>	0.89	0.06
<b>Range</b>	4.0	0.3
<b>Minimum</b>	94.3	97.1
<b>Maximum</b>	98.4	97.5
<b>Count</b>	76	76

Table 2-9: Statistical Analysis (Recovery)

The following graph (Figure 2-21) shows the BestFit curve, consisting of a normal distribution with a mean of 96.8% and a standard deviation of a mere 0.9%.



**Figure 2-21: Recovery distribution**

### **2.3.9 Gold and price (GP)**

The gold price is the one variable that the operation has no control over. However, the operation may have the ability to reduce the risk around the price by entering into hedging or other derivative programmes. There is a wide range of tools available, ranging from put and calls options and the opportunity to sell or buy any of these derivative tools. The derivatives could be focused on the gold price or the exchange rate or both. They could also be structured to cover both upside and downside scenarios.

### **2.3.10 Cash cost (Cost)**

The unit cost benefit tends to decrease as the tonnage is increased, at a reducing rate (diminishing returns) and the risk disadvantage

may ultimately cancel out any benefits gained from scale of operation. This is discussed in detail in Chapter 3.

In order to determine the cost tonnage curve, some detailed analyses needs to be done. Mine costs and particular the cost and tonnage relationships are complex and often not well understood. Previous experience has shown that in excess of 70% of operating costs are fixed in a typical conventional underground mine. Trackless operations have a 50% fixed cost component, whereas open pit operations have a fixed cost component of less than 20%. The definition of fixed and variable costs may vary depending on who controls the cost. The director may be able to change cost that a foreman cannot. Moreover, what is fixed today may not be fixed tomorrow.

It is suggested and that a definition of fixed and variable cost in the mining industry could be penned as follows:

***"Cost tends to move from fixed to variable, depending on your ability to influence it".***

This could be clearly demonstrated in the field of labour. The General Manager may not have an agreement with his unions to enter into a retrenchment mode, and thus the labour could be fixed. On the other hand, labour could be transferred to other operations, which makes it more variable.

Another question that needs to be asked is: "How accurate does a definition of fixed and variable costs need to be?" It is probably better

to risk attempting to define the cost and do the exercises than to ignore the whole process, because the cost volume curve cannot be determined accurately. (See Chapter 3 for full discussion on cost.)

## **2.4 @ Risk Monte Carlo analyses using the BME**

### **2.4.1 Methodology**

The following section was compiled using the Palisade's @ Risk tools.<sup>2</sup> This software uses the statistical profile of each variable and runs simulations using either **Monte Carlo** sampling or **Latin Hypercube** sampling methods. Many people are familiar with the Monte Carlo simulations. However, the **Latin Hypercube** sampling was used to complete these simulations.

The **Monte Carlo** sampling refers to the traditional technique of random or pseudo-random numbers selection to sample from a probability distribution. The term "Monte Carlo" was introduced during World War II as a code name for simulation of problems associated with development of the atomic bomb. Today, Monte Carlo techniques are applied to a wide variety of complex problems involving random behaviour. A wide variety of algorithms are available for generating random samples from different types of probability distributions.

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<sup>2</sup> @Risk Help manual.

Monte Carlo sampling techniques are entirely random — that is, any given sample may fall anywhere within the range of the input distribution. Samples are more likely to be drawn in areas of the distribution that have higher probabilities of occurrence. In the cumulative distribution shown earlier, each Monte Carlo sample uses a new random number between 0 and 1. With enough iterations, Monte Carlo sampling "recreates" the input distributions through sampling. A problem of clustering, however, arises when a small number of iterations are performed.

**Latin Hypercube** sampling is a recent development in sampling technology designed to accurately "recreate" the input distribution through sampling in fewer iterations than the Monte Carlo method involves. The key to Latin Hypercube sampling is stratification of the input probability distributions. Stratification divides the cumulative curve into equal intervals on the cumulative probability scale (0 to 1.0). A sample is then randomly taken from each interval or "stratification" of the input distribution. Sampling is forced to represent values in each interval and, thus, is forced to "recreate" the input probability distribution. The result is that a smooth curve of potential solutions is found more easily and with less iteration.

#### **2.4.2 Simulation settings**

The following table 2.25 shows the simulation detail for the analysis that follows.

<b>Summary Information</b>	
Workbook Name	Risk bme.xls
Number of Simulations	1
Number of Iterations	5000
Number of Inputs	6
Number of Outputs	1
Sampling Type	Latin Hypercube
Simulation Start Time	06/05/2003 09:53
Simulation Stop Time	06/05/2003 09:53
Simulation Duration	00:00:04
Random Seed	117063383

**Table 2-10: Simulation parameters**

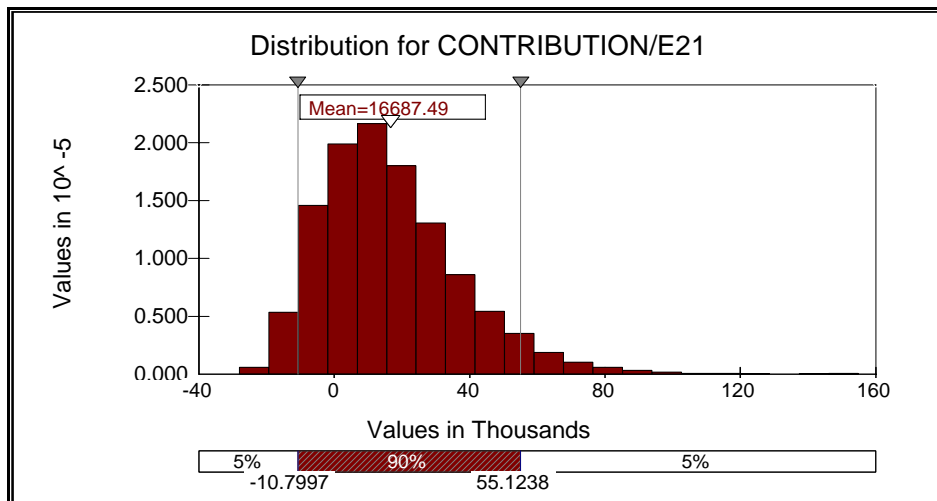
The results obtained from 5 000 and 10000 simulations are very similar and the histogram shown later shows that a reasonably smooth distribution has been obtained, which is a benefit of the Latin Hypercube sampling based on the stratification of the input probability distributions. Similar results were obtained using the Monte Carlo simulation; but more iteration was required to achieve a smooth distribution of answers. The simulation software and advanced computer hardware has facilitated the capacity to conduct multiple runs in short periods of time. The earlier simulation models needed excessive resources for even simple simulations, which made simulations unpopular.

### **2.4.3 Analysis of output**

The following graph (Figure 2.26) shows the final outcome of the simulation of the BME and its associated variables. The most likely outcome of the simulations is a profit of R16.7 million per month. This result yields the mean value for the contribution usually



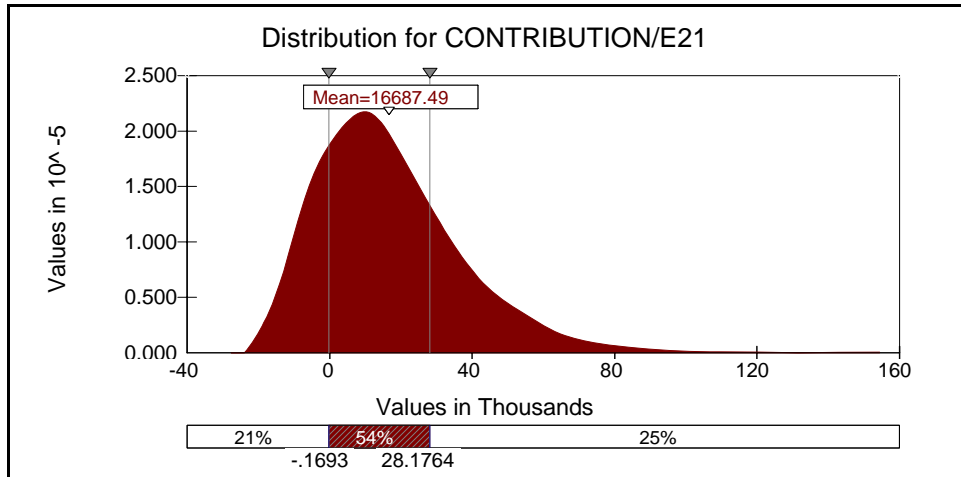
calculated as a single line calculation, as it is the current standard methodology applied in the industry. So why does one want to go to the trouble of simulating the answer? The graph also shows that in the distribution of possible outcomes, there is a 5% chance that losses in excess of R10.8 million could be made on the downside (roughly one month in two years). At the 90% confidence level, the profit could reach R55 million. This is a large range of potential outcomes and is not conducive for sustainable business. This analysis is based purely on the analysis of technical variances. The variance in the gold price is likely to compound the issue further. This wide range of outcomes suggests that better controls or improved planning should tighten the variables down. The identification of key drivers is discussed in section 2.4.4.



**Figure 2-22: Output distribution**

The following graph (Figure 2.27) shows the outcomes as a histogram with 75% upper confidence limits. From the analysis, it is

clear that there is a 21% chance of losses being made; i.e. one month in five, or roughly one quarter per annum. In interpretation is that the project has a 79% chance to break even or better.

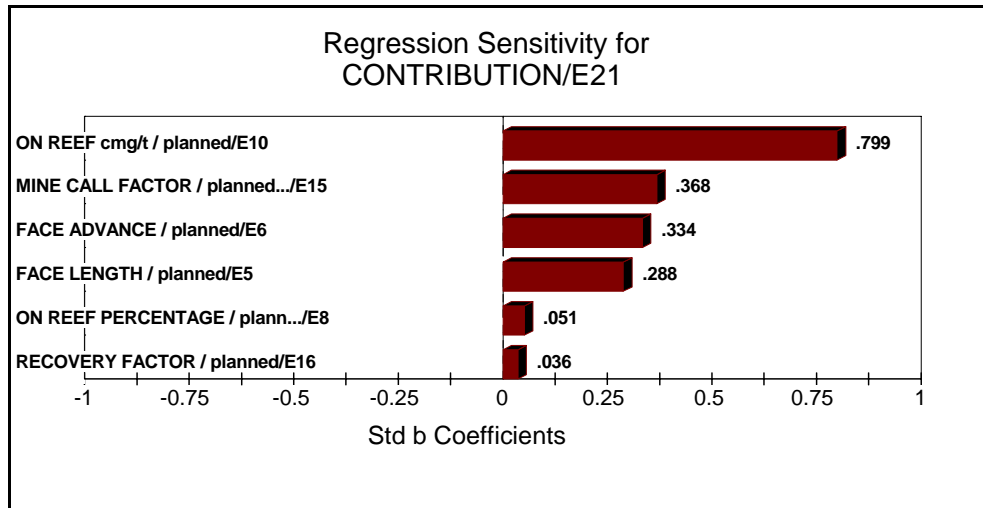


**Figure 2-23: Output distribution**

This is an unacceptable risk and needs to be managed. It would be better if the distribution could be shifted to a higher level and limited to a narrower range to reduce the monthly uncertainties. This is where the main benefit of risk analysis comes in because we are able to identify the risks and we can do something to reduce these risks. The control systems on a mine should be designed to control the ‘bottom line’, of paramount importance in the mining industry.

**2.4.4 Key driver analysis**

The following graph (Figure 2.24) is a Tornado graph of the main variables ranked in order of magnitude of impact on the ‘bottom line’.



**Figure 2-24: Tornado graph**

The biggest impact is related to the grade, with the MCF and face advance having similar impacts. It is clear that the best control and systems should focus around the forecast and control of the above variables if the risk around the making of profits is to be reduced.

The following table 2.24A shows the analysis of the outcome, as well as the input variables in the BME. This table also reflects the 5% and 95% confidence limits, as well as the range of movements within the confidence limits.

Output Name	Minimum	Maximum	Mean	Std Dev	x1	p1	x2	p2	x2-x1	p2-p1
CONTRIBUTION	-R28,031	R154,830	R16,687	R20,551	-R10,800	5%	R55,124	95%	R65,923	90%
Input Name	Minimum	Maximum	Mean	Std Dev	x1	p1	x2	p2	x2-x1	p2-p1
FACE LENGTH	1424	3354	2323	252	1908	5%	2737	95%	829	90%
FACE ADVANCE	6.2	11.8	9.1	1.1	7.1	5%	11.0	95%	3.9	90%
ON REEF PERCENTAGE	89.2%	100.4%	94.5%	1.5%	92.1%	5%	97.0%	95%	5.0%	90%
ON REEF cmg/t	623	3608	15631	366	1041	5%	2225	95%	1184	90%
MINE CALL FACTOR	68.9%	143.9%	99.6%	102.0%	83.7%	5%	117.2%	95%	33.5%	90%
RECOVERY FACTOR	93.1%	99.6%	96.7%	0.8%	95.3%	5%	98.2%	95%	2.9%	90%

Table 2-24A : Output table

### 2.4.5 Input analysis

The following range of graphs (Figures 2.25 to 2.28) shows the inputs as simulated over the 5000 runs.

The first graph (Figure 2-25) is the face length graph showing the mean of 2323 metres that generates a normal distribution in line with the inputs.

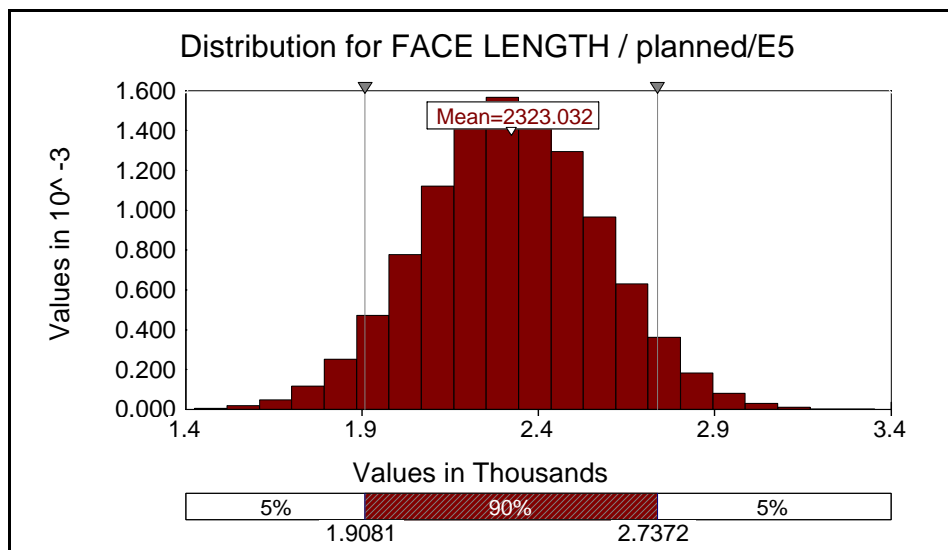
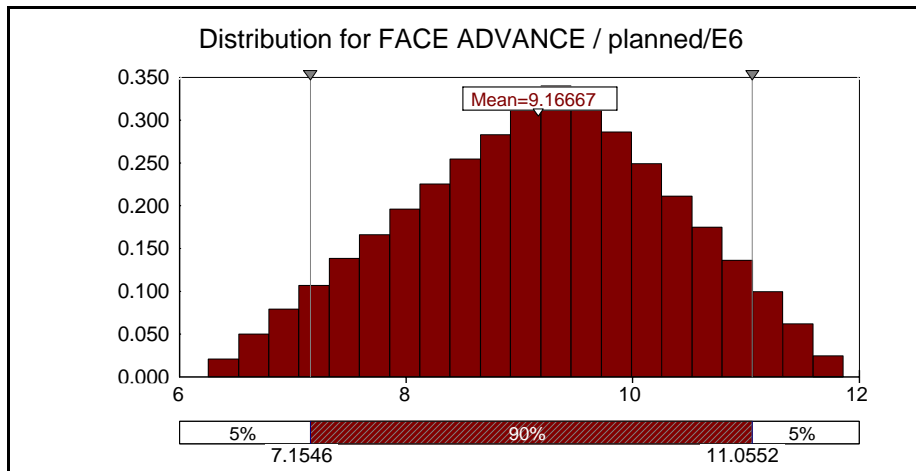


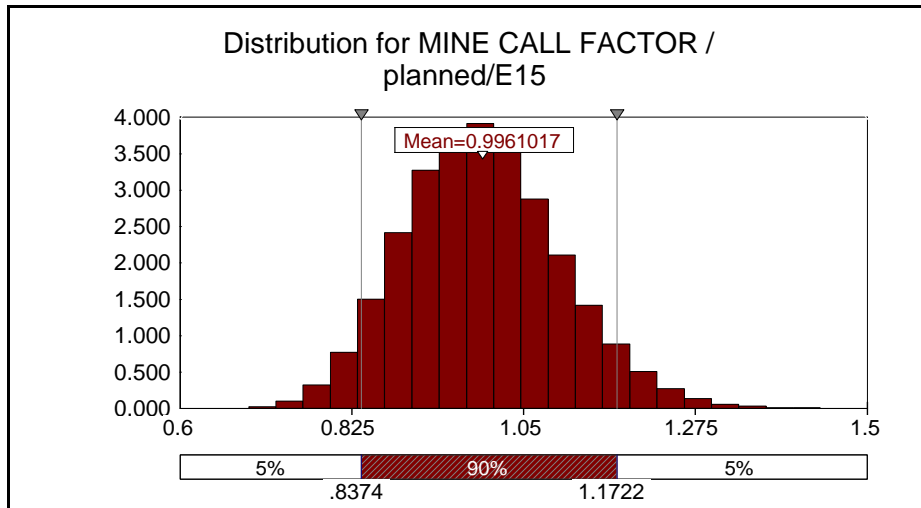
Figure 2-25: Simulated face length

The following graph (Figure 2-26) reflects the face advance simulations with a mean of 9.1 metres per month. The shape of the simulated outputs is in line with the triangular distribution specified as input.



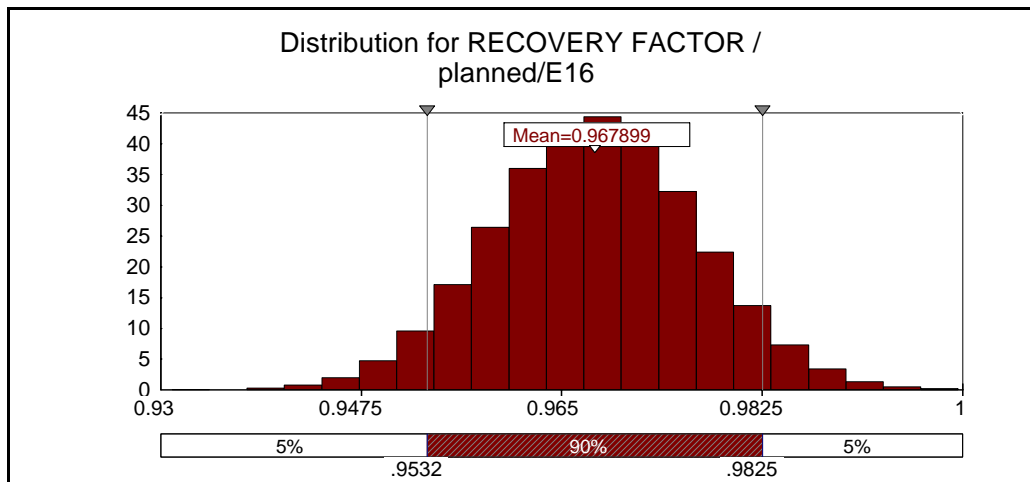
**Figure 2-26: Simulated face advance**

The next graph (Figure 2-27) shows the simulated output of the mine call factor and it reflects the normal distribution around the mine call factor as seeded in the simulation model.



**Figure 2-27: Simulated MCF**

The final graph (Figure 2-28) in the series reflects the normal distribution outputs of the recovery factor. Note that the distribution is rather tight and hence the low impact on the 'bottom line'.



**Figure 2-28: Simulated recovery factor**

#### **2.4.6 Conclusion of the stochastic analysis**

In summary, it is often better to be approximately right rather than precisely wrong. It is suggested that rather than considering single-line inputs for each variable, the variables must be understood and simulated, which results in a range of outputs and an understanding of the confidence limits around the outputs<sup>3</sup>.

Moreover, the relative risk around each variable and its impact on profitability must be understood and modelled. The control systems are then be aligned to focus on the biggest risk in the equation. This analysis is based on the technical risks in the BME and is but a part of the full analysis that can be carried out.

A further approach is to use neural networks to establish the relationship between the different variables and allow the network to 'learn' the patterns and then use the resulting model as a predictive tool. This has not been researched for this document but holds potential for future improvement in the professional conducting of Minerals Resource Management.

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<sup>3</sup> Davis G.A. (1994). Colorado school of mine. Mining Engineering, January 1995. *(Mis)use of Monte Carlo Simulations in NPV calculations.*

### **3 OPTIMISATION**

- *Definition of optimisation*
- *Basic mining equation*
- *Cigarette box optimiser*
- *Grade tonnage curve*
- *Cost tonnage curve*
- *Macro grid optimiser (MGO)*
- *Shaft design*
- *Systems reliability*
- *Case study*

#### **3.1 General observations**

The term “optimisation” has different meanings to different people. The thesaurus in the Microsoft software offers definitions such as 'best', 'most favourable', 'best possible', 'most advantageous', and 'finest'. None of these words are definitive by nature and hence the different interpretations. These differences also suggest optimisation is a relative concept and it is difficult to prove that the desired status is reached.

So the question remains, how optimisation is achieved, or for that matter, determined. It is easy to determine optimisation if the variables are few and crystallised. However, as the number of variables and their complexity increases, the optimisation process becomes more complex because of the increased number of



permutations and combinations. Confidence also decreases as the complexity increases. My favourite quotation of 'what is obscure is seldom clear' comes into play. (Clark 2000).

There are several tools available that can assist one to determine optimisation, including (the favourites) linear programming and decision trees. These have not really been used extensively in the mining industry and the reasons are often not clear. I assume it is mainly as a result of the complexity of the industry, which leads to many and often, clouded variables. It appears that most of the optimisation work is coupled to a simple iterative process driven by gut-feel. This may or may not lead to optimisation. It is suggested that a more structured process be followed.

Reviews and audits conducted at several mines (>100 to date) have indicated that optimisation is often lacking or outdated in many operations. This lack of optimisation does not mean that the operations are completely without direction, as many of them are restricted within their original strategies which (hopefully) did go through an optimisation programme. However, in some operations, the estimated NPV value of the operations could be increased in excess of 100%, if the production profile is optimised.

Optimisation is, in fact, very simple: there are two basic legs in optimisation process, namely 'tonnage changes' and 'selective mining'.

Several 'right sizing' exercises (mainly tonnage changes) have been conducted over the last few years at many operations. This is as a result of the squeeze of the profit margin. However, some healthy operations can benefit as much as their 'poor cousins' by right sizing these operations and / or optimising the grade. One can also argue that some of the older operations do not function under the same set of constraints as originally planned, as many variables have changed from the initial design days. Additional capacity might become available, market conditions might have changed, and technology may have improved. Often simple bottlenecks could be removed and capacities increased.

This section deals with the concept of "Doing the right thing versus doing things right".

### **3.2 Three step optimisation**

The level of optimisation depends on the ability to change the mining pattern and the variability of the orebody. A long-wall mine has far fewer opportunities to change than a scattered mine layout, as it is essentially restricted to long wall stopes and the mining configuration, as prescribed by the Rock Engineering department.

In addition, platinum mines are less likely to change the value of the operation by changing the mining plan compared to goldmines, as the orebody is usually highly consistent (or so it is believed).

There is nothing new in this research, but it puts different techniques and ideas together to optimise a systematic process. It also includes some software (spreadsheets) that can be used to optimise the extraction of the orebody.

A three-step process is considered, where the **first step** is to build up the algorithm of the operation, and it is called the **Basic Mining Equation (BME)**. This facilitates a good understanding of the optimisation process, as well as being a tool to measure the effectiveness of the optimisation.

The **second step** is to conduct a high-level optimisation exercise, using the **cost tonnage** curve as well as the **grade tonnage** curve that reflects the signature of the orebody. The spatial positioning of the orebody is not considered at this stage and the output is indicative of the right size as well as the impact of high- grading. The tool used is referred to as the **cigarette box optimiser (CBO)**.

The **third step** of optimisation now considers the spatial distribution of the orebody, utilising the macro block model, as well as the cost of extraction. The output is a strategy of mining as well as an indicative value of the operation. Several optimisation tools are used to facilitate the decision, ranging from an iteration approach, floating cone, linear programming etc. This system is referred to as the **macro grid optimiser (MGO)**.

This research was done using a simple series of spreadsheets but it is suggested that the Macro Grid Optimiser probably belongs in a 3-dimensional graphics environment.

### **3.3 The Basic Mining Equation (BME)**

The revival of the BME can to a large degree, be attributed to Mr Dave Diering<sup>4</sup>, whom I consider one of the world's most practical Minerals Resource experts from a gold perspective. The BME is essentially the algorithm of the operation, combining the critical variables in order to determine the expected profit. This is probably the most useful tool to measure the impact of changes in these variables.

The BME is a snapshot in time and does not consider the impact of time on the decision. It is mainly used to identify which risks are critical. The following diagram (Figure 3-1) shows a simple BME gold mine.

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<sup>4</sup> David H Diering (2001) Key note address: *The purpose and effectiveness of good Mineral resource Management.* : Colloquium for South African Institute of mining and Metallurgy.

1		FACE LENGTH	m	2,656
2	x	FACE ADVANCE	m	9.66
3	=	TOTAL m2	m2	25,647
4	x	ON REEF PERCENTAGE	%	95.54%
5	=	REEF m2	m2	24,502
6	x	ON REEF cmg/t	cmg/t	1,556
7	x RD =	kg GOLD EX STOPES	kg	1,060
8	+	VAMPI NG kg	kg	40
9	+	REEF DEVELOPMENT kg	kg	20
10	=	TOTAL kg BROKEN	kg	1,120
11	x	MINE CALL FACTOR	%	97.50%
12	x	RECOVERY FACTOR	%	97.20%
13	=	GOLD RECOVERED	kg	<b>1,061</b>
14	x	GOLD PRICE	R/kg	R72,102
15	=	REVENUE	R ('000)	R76,498
16	-	PRODUCTION COST	R ('000)	R60,662
17	=	CONTRIBUTION	R ('000)	R15,836

Figure 3-1: Basic Mining Equation

The BME is discussed in detail in the chapter 2.

### 3.4 Optimisation Process Using BME

The BME is an essential tool in testing the viability of a plan, by benchmarking against historical achievements and analysing variances. For example, tonnage targets not achieved. This may be traced back to insufficient available mining face, and point to a deficiency in the ore exposure strategy (inadequate provision of resources for ore exposure, or poor control of these resources, or both)!

### **3.4.1 Impact of sub optimal extraction**

An orebody is a non-renewable resource and you only have one chance of mining it. It is therefore crucial that the maximum value is locked in the other mining process.

The cross-subsidisation between different metals in an orebody often clouds the issue of optimisation. It is often been stated that the by-products pay for the operating cost, so therefore the main product comes for free. Could this be masking inefficiencies and sub optimality?

It has been argued by some economists, on a macro-economic level, that if producers continue to supply the market with sub-economic metal, for whatever reason they are suppressing the market price of the metal.

Thus, in principle, no metal should be mined or treated unless its cash cost of production can be covered by the price received. This means that even if the direct cost of mining and concentrating ore is met, it should not be mined unless the smelting, refining, other realisation costs (warehousing, freight, marketing and sales), overheads, interest, royalties and tax costs are met!

This principle is unfortunately difficult to follow, particularly where the metal is a secondary metal in a poly-metallic orebody.

Sub-economic metal could also reach the market via:

- Blending with economic metal to an average grade, which is economic;
- Processing of marginal ore (which excludes the cost of mining); and
- Entering into commodity price and / or exchange rate hedging contracts.

### **3.4.2 Multiple metal ore**

For a poly-metallic ore, the combined value of the ore should cover the cash cost of production (cost-to-concentrate, smelting, refining, realisation costs, etc.). In determining the combined value of ore in a poly-metallic orebody, account needs to be taken of:

The metal content in ore (in situ) of primary and secondary metals;

The mining factors;

- The metallurgical factors throughout the process in arriving at saleable metal;
- The cash cost of production; and
- The price received for each product sold.

This concept (pro-rata methodology) is demonstrated in the model set out below (Figure 3.2)

### Equivalent metal calculation

#### Metal content

	Mined	Recovered	Ratio	
<b>Nickel</b>	20000	10000	0.500	<b>A</b>
<b>Copper</b>	23000	8000	0.348	<b>B</b>

**Overall Factor                    1.438    A/B**

**This means 1.438 Unit copper will produce 1 unit of Nickel**

#### Equivalent Metal Price

**Nickel                                US\$3/lb**

**Copper                                US\$1/lb**

**Overall                                Factor3:1**

i.e. 3 saleable units of Cu will produce the same revenue as 1 saleable unit of Ni.

**Price factor = 3.**

#### Equivalent Metal Cost (\$lib)

**Nickel                                US\$2/lb**

**Copper                                US\$0.4/lb**

**Overall Factor                        Cu:Ni = 0.2:1**



i.e. the cost to produce 5 saleable units of Cu is equivalent to the cost to produce 1 saleable unit of Ni.

**Cost factor = 0.2**

**Equivalent Metal Contribution (US\$/lb)**

**Nickel** US\$3/lb - US\$2/lb = US\$1/lb

**Copper** US\$1/lb - US\$0.4/lb = US\$0.6/lb

**Overall factor** Cu:Ni = 0.6:1

i.e. the unit contribution of 1.67 units of Cu is equivalent to the unit contribution of 1 saleable unit of Ni.

**Contribution factor = 0.6**

**Aggregated Equivalent Metal in-situ (units)**

**Equivalent metal = Metal factor**

**Contribution factor**

i.e. 2.43 units of Cu in-situ (1.46/0.6) have an equivalent value to 1 unit of Ni in-situ. Aggregated equivalent metal factor = 2.43

**Cu equivalent metal (Ni) = 1**

**aggr.**

**Equivalent metal factor**

**= 0.41%**

**Thus 1% Cu in-situ has an equivalent value to 0.41% Ni in-situ!**

**Figure 3-2: Metal equivalent model**

Unfortunately, it is sometimes difficult to determine the equivalent in situ metal on a prorated basis, (as demonstrated in the preceding example) and sometimes the secondary product(s) are not metals (e.g. sulphuric acid) and / or are sold as a basket of metals (e.g. anode slimes). To overcome this difficulty, it is common to look at secondary product(s) simply as revenue that is credited to the cost of the primary product. This method provides a cost of primary product (net of by-product credits). This is the normal costing approach.

In the normal costing approach, secondary product(s) do not attract costs. The primary product attracts all the costs, but the revenue received from the secondary product(s) then offsets the costs. The cost of the primary product is thus net of by-product credits. Where there is a direct relationship between primary and secondary product(s) (i.e. if the primary metal produced increases or decreases, the secondary product produced increases or decreases proportionately), the cut-off grade can be easily determined, and based on the primary metal grades using the normal costing approach. Where this is not the case, the situation becomes more complicated.

Where there is a tenuous relationship between primary and secondary products, an alternative approach for converting secondary metal grades (in situ) to the primary metal equivalent grade must be followed. One such approach is the equivalent value (\$) method. In essence, this approach converts in situ metal to a

contained block monetary value (\$). Cut-off is then determined on block cost (\$) to produce the primary and secondary products.

Contained block value (\$) depends on units of metal (lbs) that can be recovered from the block and metal unit prices (\$/lb).

The BME has been discussed in detail in the risk analysis section. This is probably the most useful tool to measure the impact of changes in these variables. It is a snapshot in time and does not consider the impact of time on the decision. This BME could be used to determine the pay limit for single as well as poly-metallic deposits using the “goal seek” function in EXCEL. The tool is used to set the profit at zero by changing the grade of the orebody. Alternatively, one can decide what the required grade should be to achieve a certain margin, using ‘goal seek’.

### **3.5 Cigarette box optimiser (CBO)**

Optimisation could be done at a very high level using the two simple tools namely, cost tonnage curve and grade tonnage curve, as discussed below, without any reference to the spatial distribution of the orebody. This task will lead to the optimal right size of the design. There are essentially only three elements to be considered in this high-level optimisation:

- Cost tonnage curve;
- Orebody signature; and
- BME (cash flow model).

This methodology requires two basic skills.

- "**Cost Tonnage Curve**" - a good understanding of economics and accounting of the operation;
- "**Orebody signature**" - a good understanding of the orebody structure, morphology, sedimentology, deposition, facies and evaluation and
- "**Cash flow model**" - to measure impact over the life-of-mine plan.

The very different skills required means that the exercise of optimisation cannot either be considered a financial exercise or an earth science exercise. The analysis requires a combination of the skills from the two disciplines.

### **3.5.1 Methodology**

The methodology used for optimisation is a combination of the cost tonnage curve, the grade tonnage signature of the orebody, the BME, and NPV calculation.

The logic works on the premise that an increase or decrease in tonnages mined entails either a movement up or down the unit cost line of the cost tonnage curve. The higher the tonnage, the lower the cost as a result of benefits of scale of operation attributable to the fixed cost component in the cost structure. The effect of diminishing returns and an increase in risk with higher tonnages also needs to be considered. (See Figure 3.3)

The purpose of the exercise is to calculate the unit cost, which in turn impacts on the cut-off grade and pay-limit. The unit costs obtained from the cost tonnage curves are then used in the BME to determine the profitability. The profitability is then adjusted to the required margin by changing the achieved grade. These changes are made changing the required grade manually on an iterative basis, or using the 'goal seek' function in Excel.

The required grade is then transferred to the grade tonnage curve in order to determine the required cut-off as well as the tonnage available for the scenario. The tonnage and grade are then used to create a life-of-mine cash flow model to determine the resulting NPV. (See Figure 3.7) This process is repeated and graphs are generated to test the impact of tonnage and margin changes. The optimal position is then determined from the graph. (See Figure 3.10) The detail is discussed in the following subsections.

### **3.5.2 Cost Tonnage Relationship**

The easiest change you can implement in terms of optimisation is probably a change in tonnages. This is easily understood, easy to engineer, and probably has a guaranteed outcome. A change in the tonnage is most likely the easiest optimisation method to quantify as well as to monitor. Thus, the first leg of the macro high-level optimisation considers the cost tonnage curve.

The cost curve needs to reflect the operation and should be free of abnormalities and inefficiencies. These abnormalities will be discussed in greater detail later on.

For more detail see Chapter 2, Section 2.3.9 around cost tonnage curves.

The sizing of an operation is usually addressed in the feasibility stage of an operation. However, the sizing must be reconsidered from time to time as circumstances change.

- Neighbouring operations may have spare plant capacity.
- Spare development waste capacity may be filled with reef,
- In addition more labour may become available.

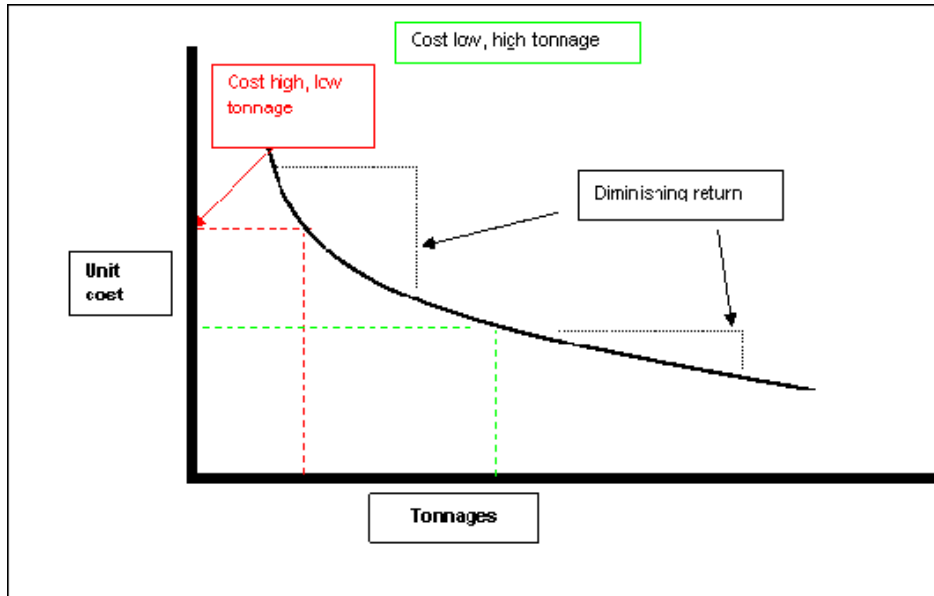
There are essentially two elements to be considered in the optimisation process, as reflected in the following two questions:

- What is the optimal tonnage for the operation?
- Are all the constraints in the system set to the same capacity, i.e. are there bottlenecks in the system that could be removed?

It was found during the audits of some mature operations, that constraints of the past are no longer constraints, e.g. more plant capacity is made available if some of the other operations are closed. Another example of that is often overlooked is that more ore could be extracted if the development reduces as the mine matures. That which are often considered bottlenecks could be eliminated through

some capital injection and that could have a very positive long-term effect. An example of this would be increasing skip sizes using lightweight material, which would improve the hoisting potential.

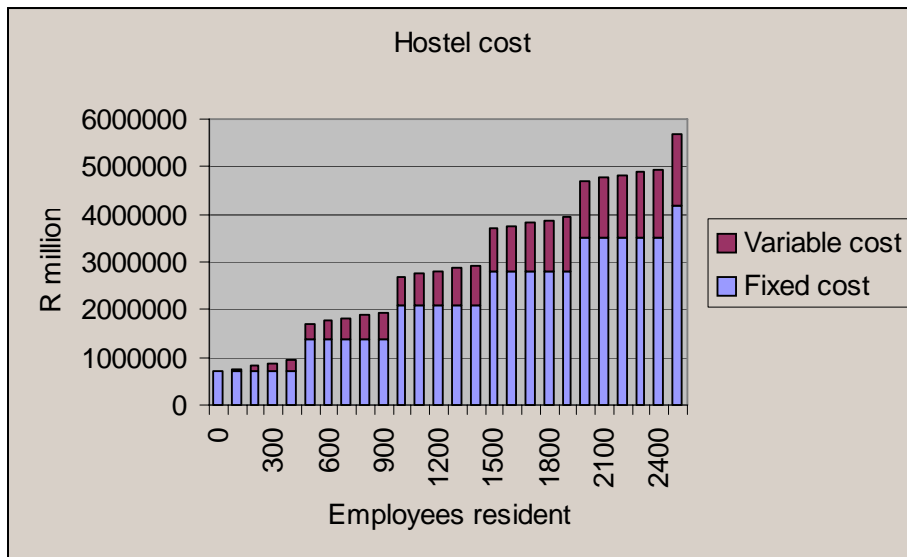
### 3.5.3 Cost-tonnage curve



**Figure 3-3: Cost-tonnage Curve**

The above graph (Figure 3-3) is probably an over simplification of the cost tonnage curve and is used for illustrative purposes only. In reality the curve is far more uneven as a result of the relevant ranges of fixed cost (also referred to as semi fixed cost). In order to explain the concept of relevant ranges, a cost element for example, hostel cost may be considered. The fixed cost consists of the hostel manager and his direct assistants, in other words, the employees one can find in most hostels regardless of size.

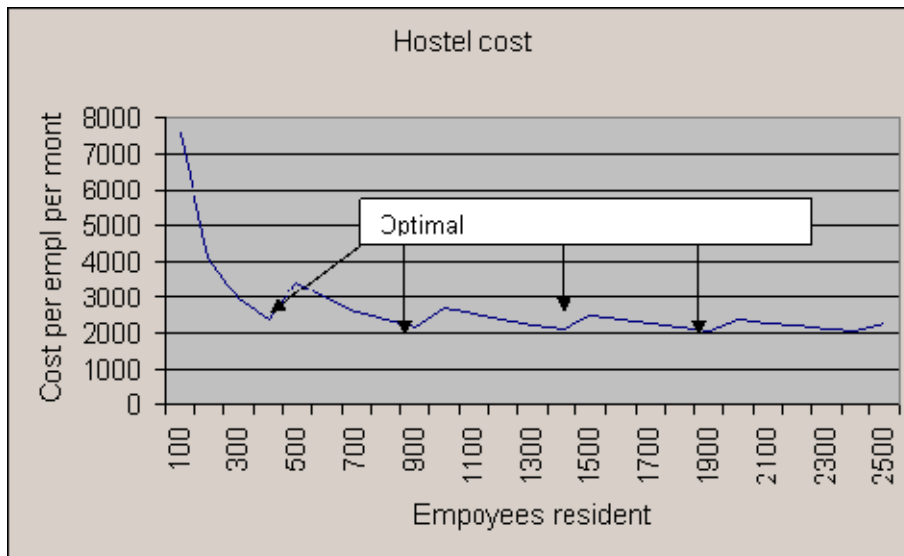
The next series of fixed cost may be associated with hostel clusters or sections containing accommodation for, say, 500 people. If one cluster is full and an additional employee enters the system, a new cluster is activated. The fixed number of personnel in the cluster is considered fixed for the next 500 inhabitants. The variable cost is associated with items that change as each unit is added. This is sometimes referred to as the incremental cost per unit. In the above example, the incremental cost consists mainly of consumables like food and cleaning material. The following graph (Figure 3.4) illustrates this concept.



**Figure 3-4: Hostel cost structure**

If the costs are converted to unit cost, it is clear that the optimal unit cost could be intersected at the point just before the next semi-variable or relevant range cost is introduced. This is demonstrated in Figure 3.5 below.





**Figure 3-5: Hostel unit cost**

### **3.5.4 Tonnage**

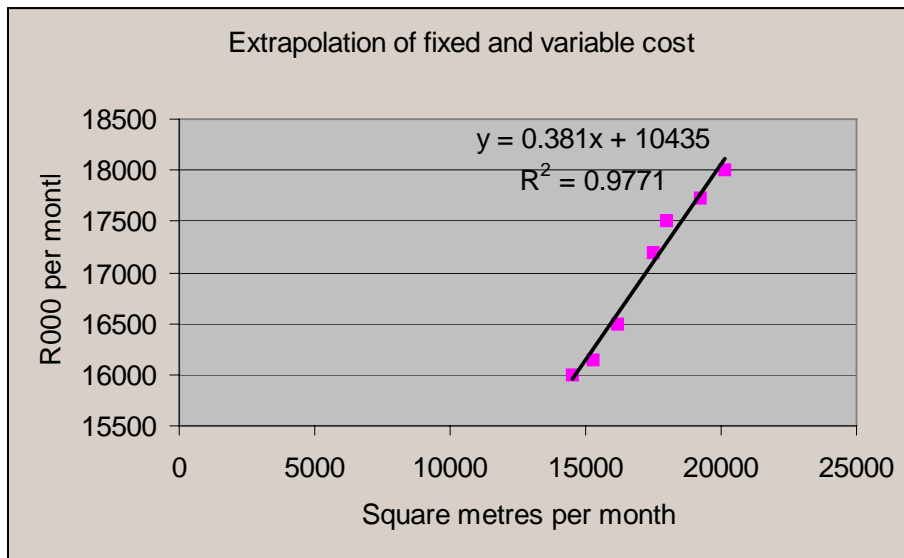
The above graph (Figure 3-5) is probably a closer illustration of the cost tonnage curve for a single item. The total cost for an operation is a combination of many of these lines.

It is evident that the cost jumps at the introduction of the new semi-variable cost element. Optimal achievement is often attributed to 'design small and overachieve' resulting in the lowest unit cost.

There are several ways to determine the cost tonnage curve and the most popular is through a process of good sense analysis and expert opinions. The person responsible for each cost centre is probably the best expert in the behaviour of his cost. This could be backed up by a statistical analysis of the cost, including applying BestFit curve to the cost and the driver. This process is very handy, as the

statistical analysis of each cost element could be used in predicting the future and the risk profile. This is discussed in detail in Chapter 3, the stochastic analysis section. Methods like Monte Carlo simulations would be suitable for combining the statistical parameters to the proposed cost profile. Microsoft's @ Risk and BestFit programs were used extensively in this research.

Another point that is critical in the cost exercise is the understanding of the cost driver and the relationship between the cost and the cost driver. It is important that costs are deflated or inflated to the same timeframe in order to facilitate a good comparison.



**Figure 3-6: Extrapolation of cost**

The previous chart (Figure 3-6) is an example of a statistical analysis of a cost item.

The correlation between the BestFit factor is very good, judging by the  $R^2 = 0.9$  factor, and the intercept suggests a fixed cost of R10 435 million per month and a variable cost of R381 per square metre. This correlation of the BestFit line is probably as accurate as needed and will probably be worse in most cases, as over or under expenditure is usually not recognised. However, armed with the statistical analysis and the expert opinion the analysis should deliver some concrete facts.

It is observed that the integrity of cost centre detail is sometimes suspect, due to inappropriate costing or a lack of controls at operational levels. Items are sometimes purchased where there is still some budget available. This makes further analysis difficult, considering the GIGO (garbage in garbage out) factor. Although the contamination of cost numbers may result in incorrect cost estimates, these estimates may not necessarily totally skew the outcomes. An example of this in an underground mine would be where stoping stores are purchased on the development budget. However, It is better to work with clean and reliable information. If such analyses are conducted and form part of the management systems, they will form part of operational key performance indicators (KPI's) and are likely to improve.

Abnormal items may appear on an irregular basis and may include major overhauls or equipment replacement, which in turn skews the costs. The impact of abnormal expenditure could be overcome by

the use of amortisation methods or withholding accounts. This practice is often referred to as 'normalising' costs.

Benchmarking, by comparing to other operations, could also be a useful task to confirm findings.

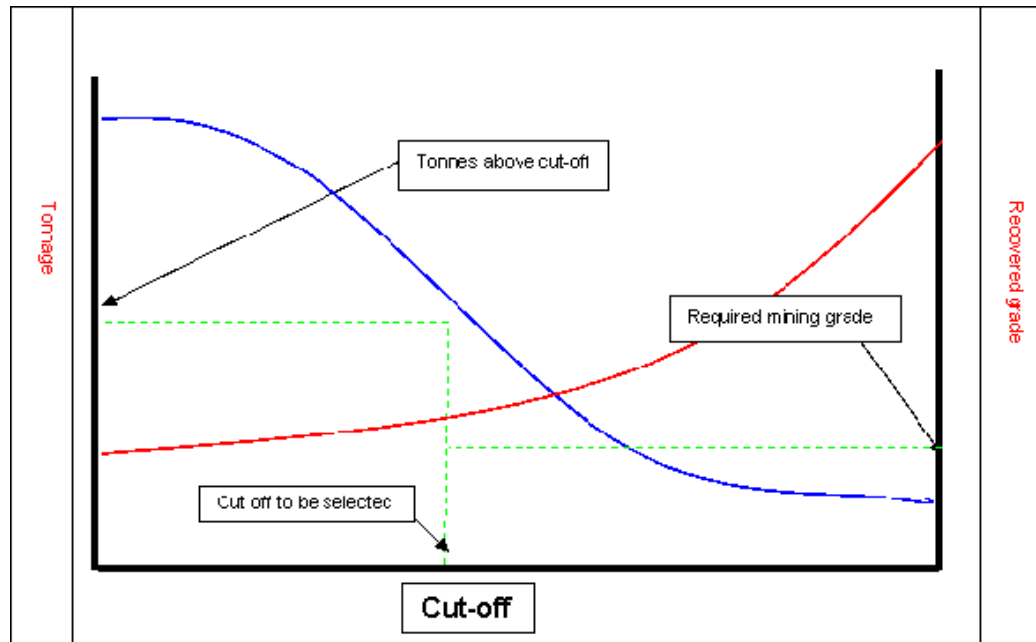
Several mines have conducted fairly extensive cost behaviour exercises and it is surprising what a large fixed component exists at most operations. Most underground conventional operations fall in the 60% to 75% range in terms of fixed components. Trackless underground operations, as a rule, have approximately a 50% fixed cost component, whereas open pit operations have a fixed cost component of less than 20%. It is also remarkable how many operations are functioning on the steep part of their cost tonnage curve, resulting in major variations in cost, considering relatively small changes in tonnage. The high fixed component also favours large operations.

A secondary and perhaps more important spin-off of a good understanding of the cost tonnage curve, for each activity, is that budgeting could become a far more controlled and scientific process. For that matter, it results in improved financial risk management.

### **3.5.5 Grade tonnage curve**

The second section of this model is based on decisions considering the orebody signature.

The graph (Figure 3-7) below depicts how the cut-off grade can be determined graphically from the grade-tonnage curve.



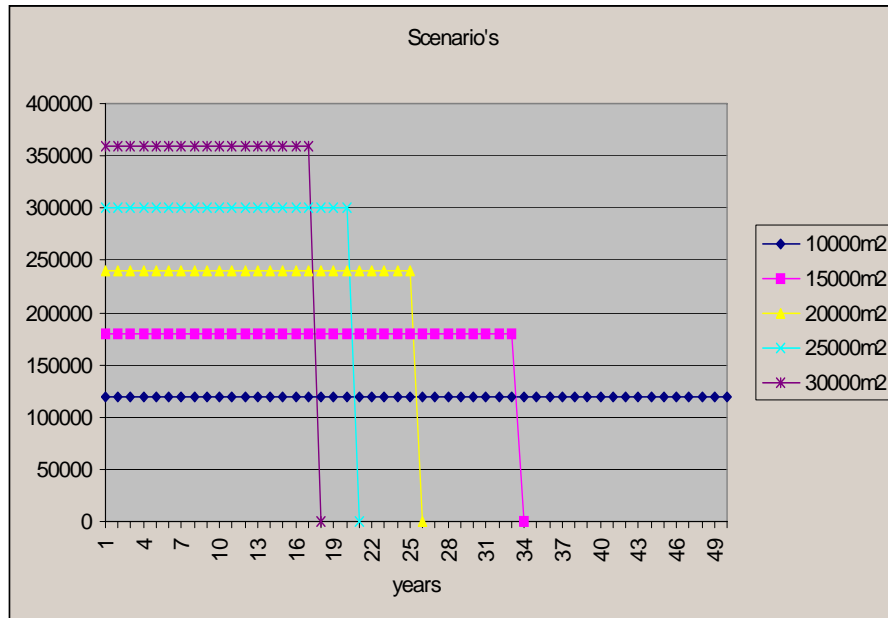
**Figure 3-7: Grade tonnage curve**

The grade tonnage curve shows that as the cut-off is decreased, the tonnage increases, while the grade achieved reduces. In other words, the life could be increased but at a reducing grade. Conversely, if the grade is increased through selected mining, the tonnage available will decrease. The mining grade is determined by the selected profit margin.

### **3.5.6 Profit/annum**

It is suggested that the above two tools (cost tonnage curve and the grade tonnage curve) is used to create a cash flow for several scenarios in order to determine the NPV for each scenario. The

following graph (Figure 3.8) shows how the profiles may look. The graph shows that the life of the operation decreases as the mining tonnages are increased.



**Figure 3-8: Cash flow for different scenarios**

There are often benefits of scale in higher tonnages. The lower cut-off may make more of the ore body viable. The time value of money is now considered. A shorter life with higher profits may generate a better NPV than a longer life with lower profit.

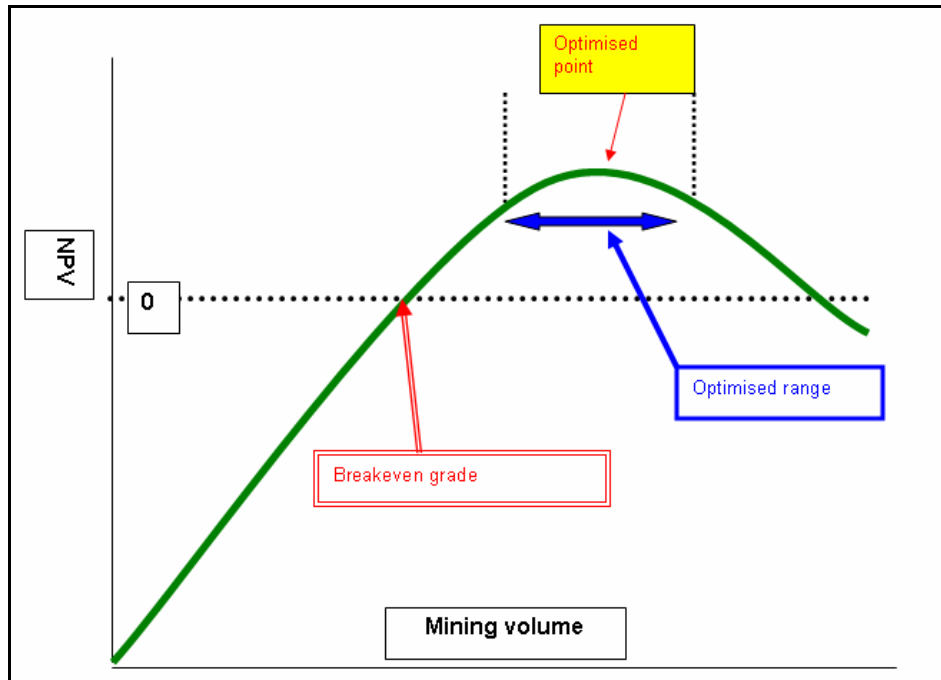
There are essentially two sets of variations of strategy that can be tested namely:

- Increasing and decreasing production tonnages at a required margin; and
- Increasing and decreasing margins at the optimal tonnage.

The optimum strategy will be where the NPV is maximised. The following graph (Figure 3.9) is a typical representation of the outcomes. The biggest drawback of this CBO method is that it ignores the dynamic changes from year to year, as well as the physical spatial positioning of the orebody. This CBO method merely shows the maximum potential for the orebody as expected from a high-level tool. It may be prudent to generate these curves at smaller logical mining areas, as implementation may be more viable. This process can also be linked to linear programming, which does not form part of this research.

The optimum tonnage design is where the NPV is maximised, but it is important that risk is considered in this equation, risk increases as the tonnages are increased. There is no clear and simple method to determine these risks and it is likely that 'expert opinion' based on past experience is one tool to consider. The main variables to be considered when determining the risk in achieving the production will be around mining, capacity and marketing constraints.

A critical component of selective mining is the ability to predict the grade accurately enough. This will be discussed later in this chapter.



**Figure 3-9: Optimal NPV Chart**

Owing to inaccuracies of this methodology, it is probably better to consider a range of solutions rather than a specific point.

Through the above process one can determine the right size of the operation as well as the best profit margin.

### **3.5.7 Cut-off grade**

There are many books, papers and notes written on cut-off grades. These include my notes from the University of Witwatersrand,<sup>5</sup>

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<sup>5</sup> University of Witwatersrand, Johannesburg (2002) Decision making for mining investments MINN 570. Department of Mining Engineering.



Johannesburg (2002), which are very useful. These notes were used for the course " Decision making for mining investments MINN 570". There are generally many views on how such a grade is calculated. The book that Kenneth F. Lane (1997)<sup>6</sup> compiled on cut-off is probably the best collection of ideas around cut-offs and is highly recommended. The issues around cut-offs, pay-limits and economic cut-offs are often misunderstood and abused. Several mines are running at a cut-off grade calculated several years ago, in spite of the many changes that have taken place in economic parameters. At a specific operation the reserves almost doubled when the cut-off was revisited for the first time in 10 years. Many an opportunity was sterilised during the last 10 years and significant value was destroyed for the shareholders.

There may also be more than one cut-off and three will be considered for this research, namely:

- Minimum economic cut-off;
- Pay limit; and
- Economic cut-off.

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<sup>6</sup> Lane, Kenneth F. (1997). *The Economic definition of Ore: Cut-off grades in theory and practice.*

The use of a cut-off is also dependant on one's ability to predict the grade. It was found at some of the nickel operations, that they were using a cut-off over many years, but their ability to predict grades was poor. A regression analysis showed a very wide 'rugby ball' of results if predicted grades were compared with achieved grades. The mining section has subsequently lowered their cut-off to allow for a margin of error and is focusing more on a geological cut-off that considers physical natural features.

### **3.5.8 Minimum economic cut-off**

This minimum cut-off is calculated on the basis of the incremental cost of mining one additional tonne. This is essentially the variable cost of an operation. If one mines ore at a grade lower than the economic minimum, it is actually costing the mine money for this additional tonne. Mining ore below this cut-off should not happen under normal circumstances and should only be considered for safety reasons, for example (i.e. pillars), and only if other alternatives are more expensive.

### **3.5.9 Pay-limit**

This is the grade where neither a profit nor a loss is made and is essentially a breakeven grade. Areas with grades below the pay-limit should be seriously considered before they are mined, as they may make a contribution to some of the fixed costs. The problem with the pay-limit is that it does not consider scale of operations. Discarding

all the 'unpay' may also lead to destruction of value. In summary, the ore between the minimum economic cut-off and the pay-limit should be seriously analysed, as it may keep good ore out of the mill and destroy value.

### **3.5.10 Economic cut-off**

This is the most critical part of the cut-off as it will be the target for the mining operation. This part is determined from the grade tonnage curve. The decision that needs to be made is what margin the operation should deliver and the required grade then needs to be calculated using the BME to deliver such a margin. The margin will be directed by the management team who should consider the optimal margin, as explained in this chapter, as well as the promise made to the shareholders by the project team in terms of the return on their investment.

This required grade, as determined by the optimisation and promised to the shareholders by the project team is then applied to the grade tonnage curve and the accompanying cut-off is then determined as part of this analysis. This method assures that the cut-off is a function of the mining plan, the cost tonnage curve, and the orebody signature.

It is important that the tonnage mined above the cut-off is mined to reflect the grade tonnage curve. This means that the grade histogram of the mined areas should closely resemble the histogram

of the reserves. Distortion to this curve may cause long-term problems as it distorts the grade tonnage curve.

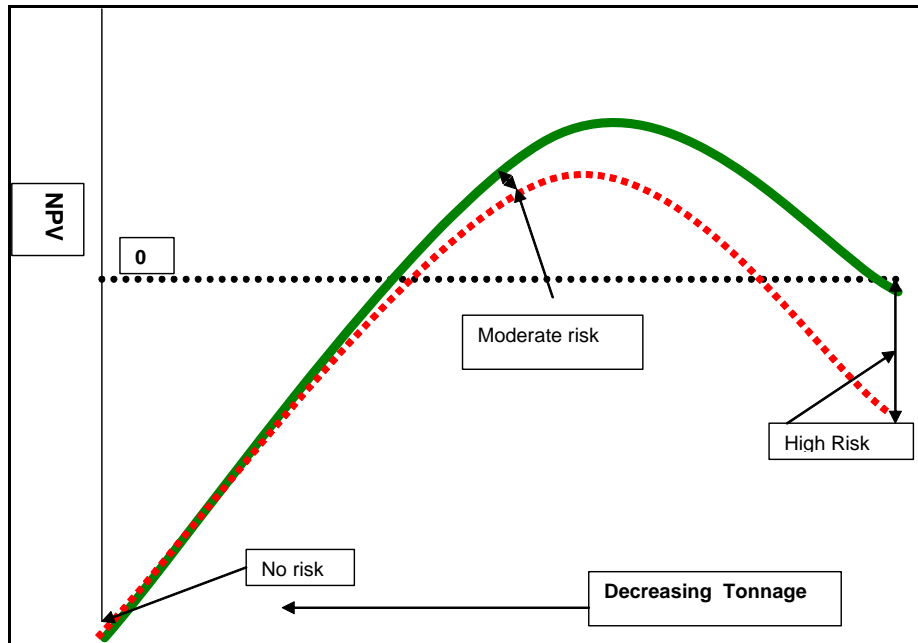
### **3.5.11 Risk analysis within the cost tonnage curve**

There are two factors of influence in this tonnage exercise. Simply speaking, if you increase the tonnage, you decrease the unit cost as a result of the scale of operations benefits; secondly, as you increase tonnage, you increase your risk. If a mine which has 5 000 employees who go on strike, the impact is far worse than it would be on a small mine with 500 people.

The following sketch reflects the risk-adjusted profile of the NPVs. It is based on the law of probabilities, which can be expressed as follows:

$$\text{Risk adjusted outcome} = \text{outcome} \times \text{probability}$$

If the law of probabilities is applied to the graph in figure 3.9, it is suggested that the higher tonnage scenario is likely to be more risky than lower tonnage scenario. This means that higher tonnage outcomes will have a bigger discount than the lower tonnage outcomes. This is reflected in the graph below (Figure 3.10)



**Figure 3-10: Risk adjusted NPV**

The determination of the NPV is usually fairly easy and can be done to reasonable levels of accuracy and reliability. Determining the probability of success and measuring risk is far more difficult to determine, as it encompasses several variables<sup>7</sup>. These risks are often difficult to measure and may be subjective. These variables include, inter alia, some of the following:

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<sup>7</sup> Smith, L.D. (1994) Kilborn Inc, Toronto, Ontario. CIM Bulletin September 1994 pp 34 to 43. *Discount Rates and Risk adjustment in Mineral Project Evaluation*. Mineral Economic Society of CIM

### **3.5.12 Capacities**

Capacities are often exceeded for short periods of time or are underestimated. Determining this risk will require the expert opinion of the shaft mine overseer in terms of tramming and hoisting capacities. Calculations regarding hoisting shifts versus tonnage and skips potential should be done to facilitate the determination of this risk.

### **3.5.13 Efficiencies associated with employees**

The chances of achieving low targets are far greater than of achieving high targets. There are essentially two ways of achieving higher production tonnages: increasing efficiencies or increasing the number of people. The favourite and least costly is to increase efficiencies. The risk, however, is higher as a person can only perform to certain levels for sustained periods. Most mines keep records of the efficiencies achieved by production teams. This will be an excellent tool to determine the risk associated with achieving the required production.

If, however, the decision is made to increase the number of employees, the cost will be higher but the risk will be less. The policy not to replace people when they leave, implemented by many operations, may be cheap in the short term but expensive from a long-term economic perspective.

The labour efficiencies are often planned very high for new operations. This does not materialise as training and flexibility need to be established. Some mines require up to seven years to achieve the desired efficiencies

#### **3.5.14 Equipment efficiencies**

Again, the chances of achieving low targets are far greater than of achieving high targets. There are essentially two ways of attaining higher tonnages; increasing efficiencies or increasing the units of equipment (locomotives LHDs etc). As per the employee variable, the favourite and least costly is to increase efficiencies. Again, the risk is higher as a piece of equipment can only perform to certain levels for sustained periods. Most mines keep records of the utilisation and availability of equipment. This analysis of records is an excellent tool to determine the risk associated in achieving the required production. This analysis has not been done here as it falls outside the scope of this research.

If, however, the decision is made to increase the units of equipment, the cost will be higher but the risk will be lower. A full cash flow leading to IRRs and NPVs should be estimated to ensure the investment would beat the required hurdle rates.

The combined impact of the above risk analysis of the efficiencies and capacities will constitute the risk around the cost tonnage curve.

### **3.5.15 Determining risk (systems reliability)**

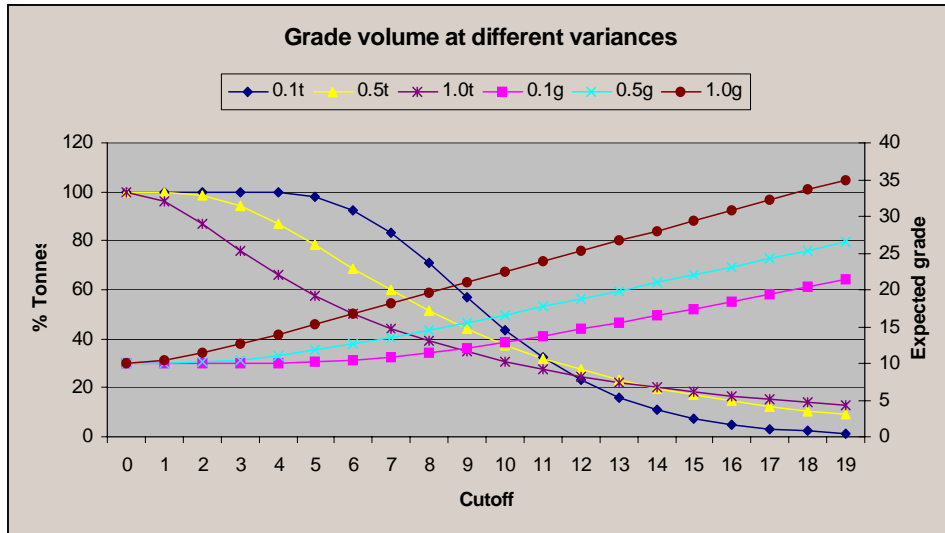
There are many techniques available to determine risk. These include historical analyses, as well as systems reliability analysis calculations. The systems, for example, hoisting or conveyor belt network, has its own system reliability. These are reliabilities that can be classified as either a series and/or a parallel system and are quite simple and easy to determine. In short, series reliability suggests a cumulative effect in that if, for example, there are two conveyor belts in tandem, each with an 80% reliability the system's reliability is  $80\% \times 80\% = 64\%$ . In terms of a parallel system the impact is compensatory in that, if you have two belts running side by side the risk in each belt ( $100\% - 80\% = 20\%$ ) compensates for the other and the overall system's risk is calculated as follows.  $\{100\% - (20\% \times 20\%)\} = 96\%$ .

### **3.5.16 Orebody signature parameters**

The grade tonnage curves are created in different ways for the different ore bodies, and the geostatistical programs used in recent years have increased the use, understanding, and accuracy of gradtonnage curves. The geostatistical methods are also different at different operations. Most operations use the standard Kriging methods. Some gold mines are moving to macro co-Kriging techniques. The spreadsheet used to generate the curves for this research is based on the standard methods. The lognormal



probability curves are generated from a mean grade value; log variance and size of the orebody. The understanding of the variability of the orebody is critical for this exercise. The following graph (Figure 3.11) shows the impact of different variabilities.



**Figure 3-11: Differences in variability**

From the above Figure 3-11, the differences between low and high variabilities are clearly visible. The higher variability has a smoothly changing tonnage curve grade, with higher tonnage at higher cut-offs. Hence the curve results higher grades at these high cut-offs. Thus, for a high cut-off, a higher variability facilitates selective mining. Conversely, a low variability probably does not facilitate selective mining. This low variability is typical of the Free State marginal mines and most of the platinum operations. Shrinking margins are difficult to handle and relatively small changes in the pay

limit may bring in or remove a great deal of the reserves. The Harmony mining group has been capitalising on this principal.

The second factor that needs to be considered is the continuity of the variability. If the variability is spread haphazardly, optimisation is unlikely to be successful. However, a high level of continuity will give relatively large areas of high grade and improve the level of success. The selected minimum mining unit will be the minimum size of selected mining. If the mining size exceeds the continuity, contamination will dilute success. This issue will become clearer in the Macro Mining Grid (MGO) section in Chapter 4.

### **3.5.17 Prerequisites for optimisation from grade tonnage perspective**

#### **3.5.17.1 The ability to predict**

The following sketch (Figure 3-12) represents the regression between the predicted grades and the grades achieved. If the distribution of points is widely spread in the shape of a rugby ball, the ability to predict the orebody using the current techniques is suspect: the wider the ball the more incorrectly one will predict the grades. This wider distribution will lead to uneconomic ore being mined as it was considered payable through the prediction method. Moreover, some payable ore will be left behind because it was considered uneconomical in the estimation process.

The narrower the distribution, the prediction of the grade is likely to be accurate and thus, the chance of optimising the extraction of the orebody is improved.

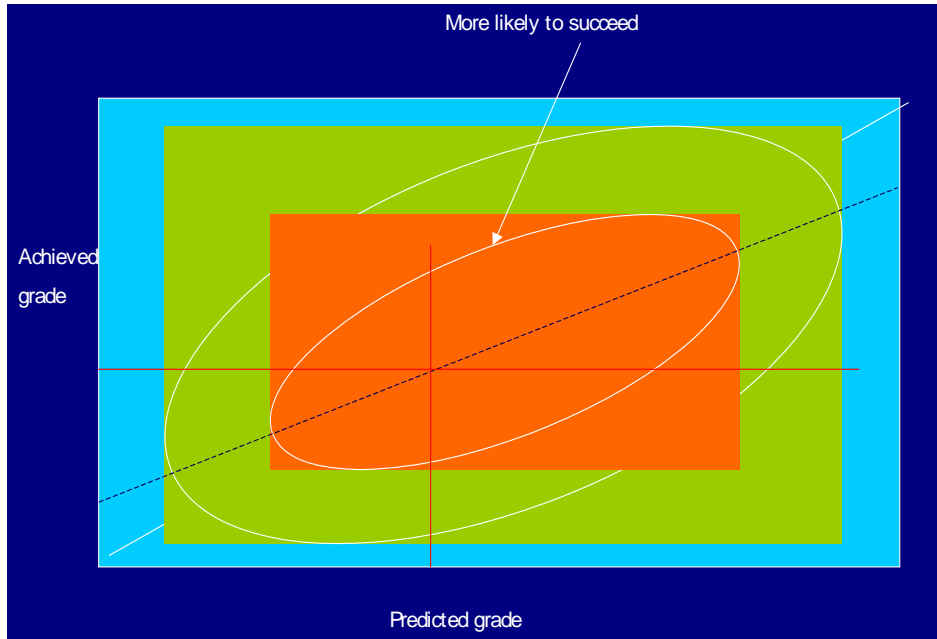


Figure 3-12: Variability in the orebody

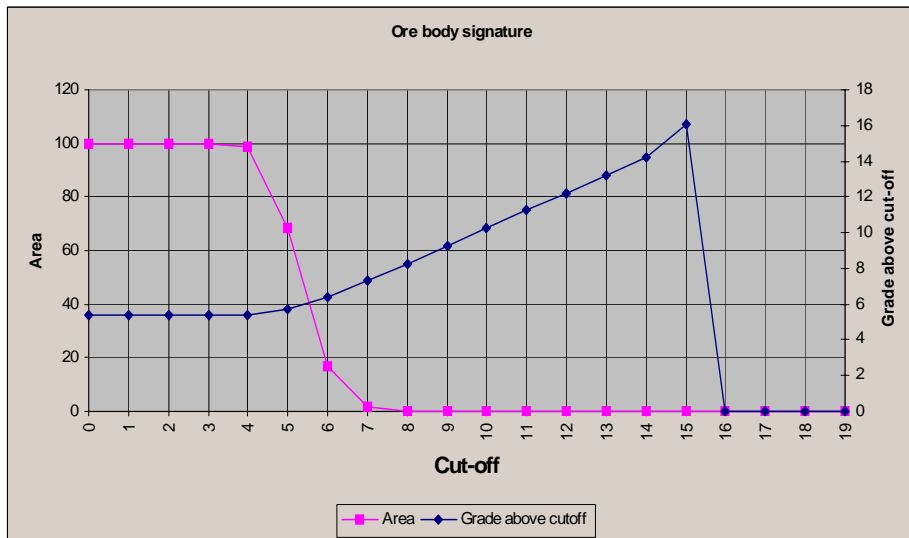


Figure 3-13: Platinum orebody signature

The above graph (Figure 3-13) shows an orebody signature of a typical Merensky Reef platinum orebody. The variability is low, which means that all the ore is above 4g/t and 90% is below 7g/t. It is unlikely that selective mining will take place, especially if the distribution is well spread, without concentrations of low or high grades. This low variability to some extent justifies the platinum definition of a pay limit as: "If you can see it you can mine it!"

### 3.5.18 Continuity of grade

The following sketch indicates that even though the amount of yellow, which reflects the high-grade is the same (the same variability), the distribution is significantly different. For example, compare the spots on a Dalmatian dog with the spots on a Friesland cow. It would be better to attempt optimisation on a Friesland than a Dalmatian!

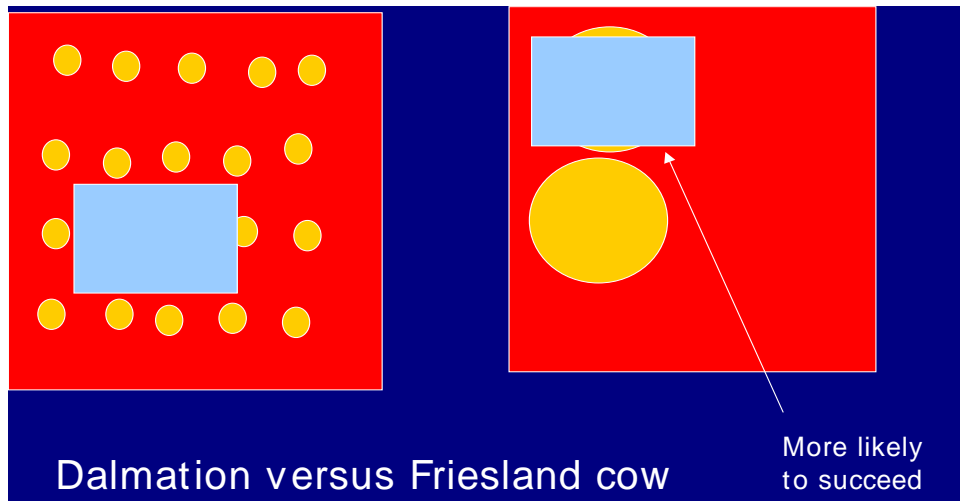


Figure 3-14: Continuity diagram

The following graph (Figure 3-15) shows a further view on continuity, namely that of relative continuity. Considering the sixty samples, the cut-off has an impact on the continuity. At a 300cmg/t cut-off, the orebody will have 100% continuity. At 800cmg/t there is less than an estimated 40% continuity. This also ignores the spatial impact of the continuity, which is another field of specialised research.

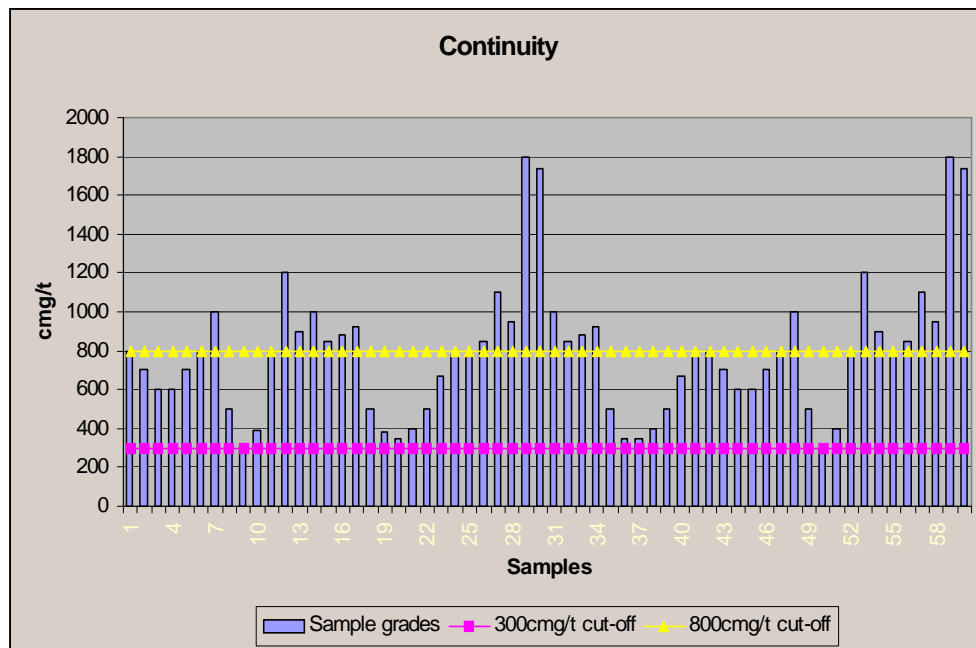


Figure 3-15: Continuity graph

### 3.6 Overall conclusion of orebody optimisation requirements

It is suggested that some formulation be developed to quantify the orebody's capacity for optimisation considering the regression, variability, continuity and economic assumptions. The outcome should be a factor that will indicate to planners what level of optimisation could be achieved.

In summary, the following could be said regarding optimisation of ore bodies:

- The lower the variability the less likely the planner is to succeed in optimisation (more of the same);
- The higher the continuity the more likely the planner is to succeed in optimisation; and
- The better the predictability the more likely the planner is to succeed in optimisation.

### **3.7 Case Study: -Optimisation Using Cost Tonnage Curve And Orebody Signature**

#### **3.7.1 Overview**

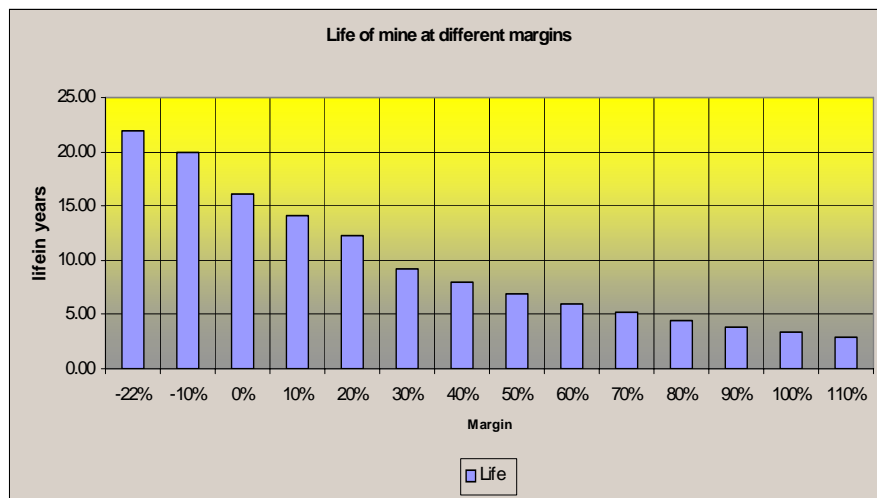
A simple model was developed to test various scenarios and to find out which orebodies are more suited to optimisation than others. The different variables were tested to determine what assumptions around the variables have a greater impact on the profitability of the mine. For example, does a low-grade mine have a greater chance to be improved, through optimisation than a high-grade mine?

A Free State gold mine at a depth of 1000m was modelled with 5 million square metres at 10 g/t with log variance of 0,5. The mining operation's Mineral Resource Manager supplied these variables. The operation mines 20 000 m<sup>2</sup> a month at a cost of R2000 m<sup>2</sup>. The efficiencies include 5% off-reef mining, 90% MCF, and 95% recovery rate. The price assumed for the test is R70 000 per kg.

The above operation has a break-even grade of 1275 cmg/t. Using the model it is determined that this orebody will not produce a profit if it is mined to the average grade of the orebody or, in other words, at zero cut-off. In fact, it will result in a 22% loss. The resource suggests that 21,9 years of mining would be possible provided some kind sponsor could be found. These results demonstrate one potential optimisation scenario, namely optimising life. This solution is obviously not feasible.

The next scenario tested is to maximise life but to eliminate the losses, i.e. mine at breakeven. This suggests that mining should take place through a selective mining process, producing an average grade of 1275 g/t (12,75 g/t). In order for this grade to be achieved, the grade tonnage curve is used to determine the cut-off. Moreover, the grade tonnage curve must reflect the block size relevant to the decision. For example, if you make your selection to mine at the micro level, e.g. per panel, the support of 30 x 30 m blocks could be used, but if you select by raise line, the support of eg 750 x 150 or 180 x 180, should be used.

The following chart (Figure 3-16) shows that if the necessary selective mining can be done, the life will decrease as uneconomical ore is cut out. This specific orebody shows that the longest 'economic life' is 16,1 years, where the mine covers its costs but makes no profit for the shareholders.



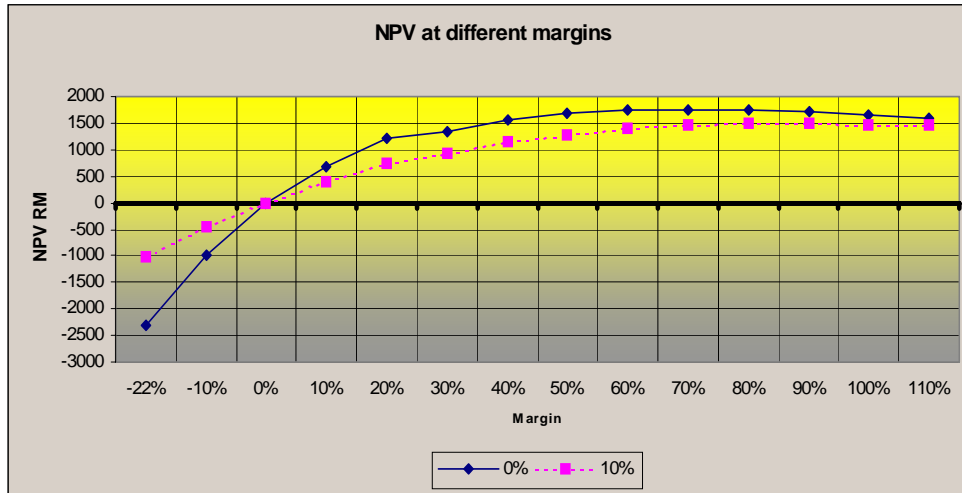
**Figure 3-16: Life of mining analysis**



These 16 years will be the maximum life of the mine, but the mine will have no value for the shareholders. This scenario is the best for the employees, as they will have employment for 16 years. However, if the margin is increased, the shareholders will be advantaged by the cost of a shorter life. At a 100% margin, the mine is likely to produce only for 3,3 years. It must be stressed that this is only valid if this level of selective mining is achievable and sustainable.

### **3.7.2 Margin**

However, in terms of value, there is a link between profit and life and it is important that one determines where the optimum position is. The following chart (Figure 3-17) shows how the NPV increases at higher profit margins. However, due to the shorter life as a result of selective mining, the NPV reaches optimality at 90% margin and then starts to reduce. It is also evident that the incremental value reduces substantially after 30% profit margin. An additional factor to be considered for this exercise is the risk associated with selective mining. The higher the selectivity opted for, the higher the inherent risk. If the impact of risk is considered, a mining plan with a margin of 30% to 40% is probably preferable as it has significantly less risk than a mining plan with a 90% margin.



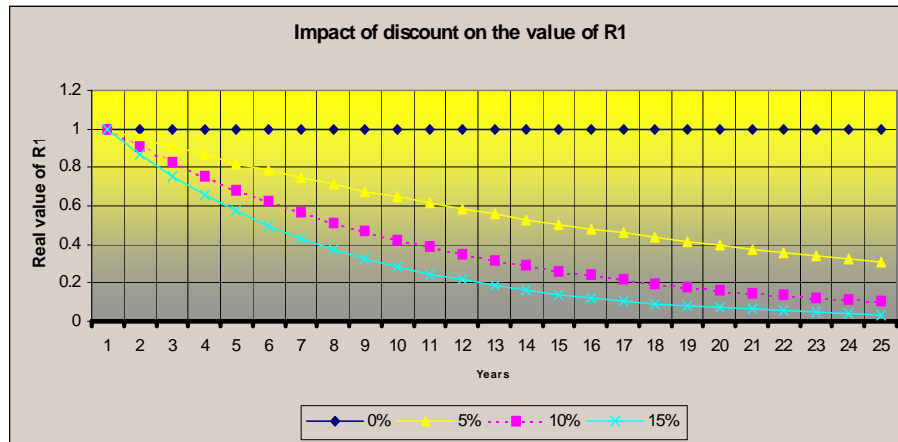
**Figure 3-17: Value at different margins**

It is recommended that a further discount for risk be made before the final selectivity is decided upon. The above graph (Figure 3-17) also shows the impact of difficult discounts on the final outcome. (See section on risk discounts).

In optimising a project, the return promised to the investors when they originally supplied the capital for the operations should be revisited. An internal rate of return (IRR) calculation should be carried out, including sunk capital and earnings, to determine if the new plan matches or improves on the original promised return.

### **3.7.3 Impact of Discounts**

The following graph (Figure 3-18) shows the value of R1 in Rand terms at various discounts. It is clear that at a 15% discount, the R1 profit will only equate to 25 cents in 10 years.



**Figure 3-18: Impact of discount rate**

A rule of thumb for this time value of the discounts of R1 is that the value reduces by half every 15 years at 5% (8 years for 10% and 6 years for 15%).

The discount used is usually a function for the cost of capital and an allowance for risk. Thus, the higher the risk is, the higher the discount rate should be. Therefore, if you have a high-risk project it is probably better to earn the profits as early as possible, which is mostly the case in South Africa.

This exercise to determine optimality as described in this case study is usually completed in the feasibility stage of the operation, as the impact of various tonnages could also impact on the profitability and, as such, the design should match the optimisation.

Once this exercise to determine the optimal cut-off and volume is completed, the spatial distribution needs to be considered, which will be discussed in Chapter 4. This exercise, considering spatial

distribution, is geared to the medium- to long-term plan, aimed mainly at the probable reserves and is conducted on the macro block plan. The purpose is to determine strategy.

The third leg of the optimisation exercise is to optimise the short-term plan, this is not covered in this research, as the planning teams normally do it. A process of iterations is the favoured tool. This includes the reallocation of resources to different working places and getting the required target from the production staff.

## 4 MACRO GRID OPTIMISER (MGO)

- *BME*
- *Grade grid*
- *Development grid*
- *Infrastructure*
- *Extraction*
- *Services*
- *Overheads*
- *Risk allowance*
- *Present value*
- *Timing*
- *Contribution*
- *Multi-product approach and future applications*
- *Conclusion*

### 4.1 Overview

Chapter 3 covered optimisation from a macro perspective ignoring the spatial distribution of the orebody and focused on the orebody signature as well as the cost volume curve. However, the spatial distribution of the orebody does have an effect on optimisation and hence this chapter, which covers the Macro Grid Optimiser (MGO).

This third method of optimisation is a macro tool considering the spatial positioning of reserves and resources and in reality is an optimisation tool that can be used to optimise the extraction strategy. It is

run as a spreadsheet model for this research but it essentially belongs in the 3D-graphics environment as a planning tool. This method holds major potential as it fundamentally combines the three-dimensional ore body signature with the economic assumptions, the science of the cost of extractions, as well as the impact of the time value of money.

This tool has been adapted as a tool to identify exploration targets for base metal operations. It is also used to overcome complexities with multiple products with different economic implications, as is the case with platinum.

The current focus in the industry is still on a compartmentalised approach, where the geologist focuses on the resource, and the mine planner on the mine design and schedule that ultimately leads to the reserve statement. The mine plans are often devoid of optimisation as the planner does not focus on the cost of the plan, and the grade often plays a minor role. Moreover, the cost accountants do not understand the mining process.

This optimal planning process can only be embarked upon when the optimal operation size, optimal margin and optimal designs have been completed, using the cigarette box optimiser and the grade tonnage curves. In existing operations, it is normally assumed that the macro optimisation utilising the grade tonnage curves were conducted during the feasibility study. However, changing price and cost scenarios, as well as operational constraints, may change the

optimal solution. It is suggested that the optimisation process is revisited every year to ensure continued optimisation. To quote from the 70's best seller 'Future shock': " The future is not what it used to be!"

The process starts once more with the Basic Mining Equation (BME).

The following BME was used with the macro grid optimiser.

BME			BUDGET MONTHLY	BUDGET MONTHLY
	FACE LENGTH	m	2,500	2,500
x	FACE ADVANCE	m	9.00	108.00
=	TOTAL m2	m2	22,500	270,000
x	ON REEF PERCENTAGE	%	95.54%	95.54%
=	REEF m2	m2	21,497	257,958
x	ON REEF cmg/t	cmg/t	1,556	1,556
x RD =	kg GOLD EX STOPES	kg	930	11,158
+	VAMPING kg	kg	40	480
+	REEF DEVELOPMENT kg	kg	20	240
=	TOTAL kg BROKEN	kg	990	11,878
x	MINE CALL FACTOR	%	98.00%	98.00%
x	RECOVERY FACTOR	%	97.20%	97.20%
=	GOLD RECOVERED	kg	<b>943</b>	<b>11,315</b>
x	GOLD PRICE	R/kg NOMINAL	R83,592	R83,592
=	REVENUE	R ('000) NOMINAL	R78,820	R945,835
-	PRODUCTION COST	R ('000) NOMINAL	R58,000	R696,000
=	CONTRIBUTION	R ('000) NOMINAL	R20,820	R249,835
RD Factor = 0.0000278			2.780	
INFORMATION				
	STOPPING WIDTH	cm	128.0	128.0
	EXCHANGE RATE	R/\$ NOMINAL	R6.50	R7.80
	GOLD PRICE	US-\$/OZ NOMINAL	\$400.00	\$333.33
	SPOT GOLD PRICE	US-\$/OZ NOMINAL	\$400.00	\$270.00
MEMO				
	BREAK EVEN PRICE	R/kg NOMINAL	R61,512	R61,512
	BREAK EVEN PRICE	\$/oz NOMINAL	\$294	\$245
	MARGIN (CONTR/COST)	%	35.90%	35.90%
	BREAK EVEN GRADE	cmg/t	1,118	1,118
	BREAK EVEN GRADE	g/t	8.74	8.74
	Cost/ square metre for BME		2,578	2,578

Figure 4-1: Basic mining equation

The final section of the BME checks the macro grid and tests the correctness of the inputs. As can be seen at the bottom of the BME (Figure 4-1), the overall costs of the BME are estimated at R2578 per square metre, which compares well with the Macro Grid Optimiser (MGO) cost of R2571 per square metre, as seen in Figure 4.2.

Test of cost input	Ave cost in MGO model	R/m2
Should reflect BME costs within reason	Extraction	R781
	Services	R241
	Development	R149
	Overheads	R1,400
	<b>Total cost</b>	<b>R2,571</b>

**Figure 4-2: Cost summary table of MGO**

## **4.2 Grade grid**

The MGO process starts with the capturing of the grade of each block. This is obtained from the macro kriging model of the orebody. These blocks are colour coded displaying all blocks below the cut-off (as determined in the previous exercise) in red, between the cut-off and the pay-limit (also as determined in the previous exercise) in yellow. The green represent the profitable ore above the pay-limit. The following diagram (Figure 4-3) reflects the grade as captured.



	A	B	C	D	E	F	G	H	I	J	
1	1360	1800	1900	2000	2100	2100	2300	2400	2500	2600	1
2	1000	1100	1200	1300	1400	1400	1600	1700	1800	1900	2
3	1000	1100	1200	1300	1400	1400	1600	1700	1800	1900	3
4	1000	1100	1200	1300	1400	1400	1600	1700	1800	1900	4
5	1000	1100	1200	1300	1400	1400	1350	1450	1550	1650	5
6	1000	1100	1200	1300	1400	1500	1600	1700	1800	1900	6
7	1000	1100	1200	1300	1400	1500	1600	1700	1800	1900	7
8	900	1000	1100	1200	1300	1400	1500	1600	1700	1800	8
9	800	400	500	600	700	800	900	1000	1100	1200	9
10	400	500	600	700	800	900	1000	1100	1200	1300	10
	A	B	C	D	E	F	G	H	I	J	

**Figure 4-3: MGO grade grid**

It is clear that the top right hand corner holds the best potential, and knowledge of an NPV calculation (time value of money) suggests the best value should be mined out as quickly as possible.

The average grade for the orebody is estimated at 1284 cmg/t. The blocks are 300 metres by 300 metres as this size had been determined as the optimal block size from a macro Kriging perspective and based on the statistical signature of the orebody. This process could be applied to a three-dimensional orebody as well, if the logic was converted to a graphics package like Datamine.

In poly-metallic ore bodies, the grade could be converted to equivalent grades as discussed in this chapter, or the grade could be converted to revenue per tonne. This equivalent grade is discussed in Section 15 of this chapter.

### 4.3 Development grid

However, some blocks are developed where others are not, and they cannot be treated in the same way. In order to facilitate an equitable decision, the cost of development for each block is determined and converted to a paylimit that is required to pay for the development. The development cost grade (paylimit) is then subtracted from each block, resulting in the remnant grade after the cost of the development has been accounted for. The following diagram (Figure 4.4) shows all the blocks at an equivalent reserve and more compatible to each other.

	A	B	C	D	E	F	G	H	I	J	
1	1281	1721	1821	1921	2021	2021	2221	2321	2421	2521	1
2	921	1021	1121	1221	1321	1400	1600	1621	1721	1821	2
3	921	1021	1121	1221	1321	1361	1568	1621	1721	1821	3
4	921	1021	1176	1221	1321	1361	1553	1621	1721	1821	4
5	921	1021	1181	1221	1321	1376	1334	1371	1471	1571	5
6	921	1021	1121	1221	1400	1500	1600	1621	1721	1821	6
7	921	1021	1121	1221	1321	1421	1576	1621	1721	1821	7
8	821	921	1021	1121	1221	1321	1421	1521	1621	1721	8
9	721	321	421	521	621	721	821	921	1021	1121	9
10	321	421	521	621	721	821	921	1021	1121	1221	10
	A	B	C	D	E	F	G	H	I	J	

**Figure 4-4: MGO development adjusted grid**

The development cost is determined using either graphical methodologies or factorising, using square metre per metre ratios.

The dip of the reef, the mining methods and geological complexities are all factors that could influence the quantities and cost of development. The mine could be broken down into development zones. The process would require a reasonable understanding of the ratios, cost, risks, and efficiencies that are associated with the development design.

It is important that the impact of locking money up in development that will only be mined well into the future is measured against the risk that is mitigated in the process. This issue of development is often poorly addressed at numerous operations. It is also observed that lean and mean mine designs can reduce the stoping efficiencies. This is tantamount to being 'penny-wise and pound-foolish'. Needless to say, the development programme and its implications is one of the highest key performance indicators for the Business Manager and the Minerals Resource Manager.

The average grade is now estimated at 1215 cmg/t for the orebody. This means the grade is reduced by 70cmg/t to account for the development. All the blocks are now essentially adjusted to the 'measured' category from a development perspective.

The cut-off and pay-limit are also adjusted to account for the cost of the development.

The process of 'reducing the grade' to account for cost, is repeated in a similar way for all the cost categories. The final outcome is a profit grade. The items to be addressed are:

- Major infrastructure (split back to the blocks serviced);
- Development;
- Services;
- Mining method;
- Balance of overheads; and
- Risk.

#### **4.4 Major infrastructure**

The following item to be considered is the major infrastructure required in each area. Major infrastructure typically comprises of items associated with the capital programme, like decline extensions, transfer systems, access haulages and associated equipment. The capital estimate is often subject to significant effort and detail. The return on investment is then calculated and finally presented to the board for approval. Finally, the expenses are usually well controlled. However, when it comes to payback time, the cost of the capital does not appear in the pay-limit or cut-off calculations. It is suggested that the capital is considered either through the inclusion of ongoing capex or that amortisation of these assets be included in the calculation.

The following diagram (Figure 4-5) shows the grade grid, inclusive of the cost of infrastructure. This model includes some R400 million to create access to a block, out of reach of the existing infrastructure.

	A	B	C	D	E	F	G	H	I	J	
1	10000000	0	0	0	0	0	0	0	0	0	1
2	0	0	0	0	0	0	0	0	0	0	2
3	0	0	0	0	0	0	0	0	0	0	3
4	0	0	0	0	0	0	0	0	0	0	4
5	0	0	0	0	0	0	0	0	0	0	5
6	0	0	0	0	0	0	0	0	0	0	6
7	0	0	0	0	0	20000000	5000000	0	0	0	7
8	0	0	0	0	0	5000000	1000000	25000000	100000000	0	8
9	0	0	0	0	0	0	0	250000000	250000000	0	9
10	0	0	0	0	0	0	0	0	0	0	10
	A	B	C	D	E	F	G	H	I	J	

**Figure 4-5: MGO Major Infrastructure costs grid.**

This infrastructure cost is converted to cost per square metre or tonne, as required. This cost in turn is converted to grade required to pay for this infrastructure and subtracted from the orebody.

The following diagram (Figure 4-6) reflects the grades available after considering the development and infrastructure. It is now clear that the remnant grade may not be viable in certain blocks, as the cost of infrastructure is prohibitive.

This process is often not present in the planning procedure, as it is often assumed that some mysterious capital programme run by head office pays the infrastructure.

The infrastructure section is revisited at the end of the exercises and is removed from the spread sheet if not required, or redistributed if partially mined.

	A	B	C	D	E	F	G	H	I	J	
1	1229	1721	1821	1921	2021	2021	2221	2321	2421	2521	1
2	921	1021	1121	1221	1321	1400	1600	1621	1721	1821	2
3	921	1021	1121	1221	1321	1361	1568	1621	1721	1821	3
4	921	1021	1176	1221	1321	1361	1553	1621	1721	1821	4
5	921	1021	1181	1221	1321	1376	1334	1371	1471	1571	5
6	921	1021	1121	1221	1400	1500	1600	1621	1721	1821	6
7	921	1021	1121	1221	1321	1316	1550	1621	1721	1821	7
8	821	921	1021	1121	1221	1295	1416	1390	1096	1721	8
9	721	321	421	521	621	721	821	-392	-292	1121	9
10	321	421	521	621	721	821	921	1021	1121	1221	10
	A	B	C	D	E	F	G	H	I	J	

**Figure 4-6: MGO Major infrastructure costs grid**

The average grade is now estimated at 1177 cmg/t for the orebody. This means the grade is reduced by 40cmg/t to account for the additional infrastructure. All the blocks are now essentially adjusted to the 'measured' category from a development and infrastructure perspective.

#### 4.5 Extraction grid

The extraction costs are often different in different areas because of differences in the mining method or in the orebody.

The factors that may influence the extraction costs include:

- Additional refrigeration at depth;
- Backfill;
- Low efficiencies;
- Down-dip or up-dip mining;
- Additional support;
- Hydropower;
- Additional shifts;
- Trackless mining;
- Lower or higher widths;
- Secondary mining;
- Special areas;
- Throw blasting; and
- Density or reef difference.

Different extraction cost rates can be used for the different areas.

The biggest difference is usually associated with labour efficiencies, as labour cost constitutes between 40% and 70% of the costs.

The following diagram (Figure 4-7) shows the grade, inclusive of accounting for the extraction costs.

	A	B	C	D	E	F	G	H	I	J	
1	756	1248	1348	1448	1548	1548	1843	1943	2043	2048	1
2	448	548	648	748	848	927	1222	1243	1343	1348	2
3	448	548	648	748	848	888	1190	1243	1343	1348	3
4	448	548	704	748	848	888	1174	1243	1343	1348	4
5	496	643	897	937	1037	1093	1051	1087	1093	1098	5
6	543	643	837	937	1116	1216	1316	1337	1343	1348	6
7	543	643	837	937	1037	1032	1266	1337	1343	1348	7
8	443	543	737	837	937	1011	1132	1106	718	1248	8
9	343	-57	137	237	337	437	537	-676	-671	648	9
10	-57	43	237	337	437	537	637	737	743	748	10
	A	B	C	D	E	F	G	H	I	J	

Figure 4-7: MGO grade grid post the extraction costs

The average grade is now estimated at 807 cmg/t for the orebody. This means the grade is reduced by 377cmg/t to account for the extraction cost.

#### 4.6 Service grid

The next stage of the exercise is to account for the services of the area. Some areas are close to the shaft, whilst others are far away, which results in lower efficiencies and more expensive services. Access to some areas is more complex and requires more services in getting the ore to the station.



Items to be considered include, inter alia, the following:

- Distance from shaft;
- Declines or inclines;
- Transfer systems;
- Refrigeration;
- Workshops;
- Additional surface fans;
- Sub-shafts;
- Trammings systems; and
- Age of infrastructure (old areas needs more maintenance and efficiencies are sacrificed).

The following table (Figure 4-8) reflects the previous grid (Figure 4-7) but adjusted for the cost of infrastructure.

	A	B	C	D	E	F	G	H	I	J	
1	661	1154	1254	1354	1454	1454	1748	1848	1948	1954	1
2	354	454	554	654	754	833	1127	1148	1248	1254	2
3	307	407	507	607	707	746	1048	1101	1201	1207	3
4	307	407	609	654	754	793	1080	1148	1201	1207	4
5	354	501	803	843	990	1045	1003	993	951	957	5
6	401	501	743	843	1069	1169	1269	1243	1201	1207	6
7	401	501	743	843	990	985	1219	1243	1201	1207	7
8	301	401	643	743	843	917	1038	1012	576	1107	8
9	201	-199	-4	96	196	296	396	-818	-812	507	9
10	-199	-99	96	196	296	396	496	596	601	607	10
	A	B	C	D	E	F	G	H	I	J	

Figure 4-8: MGO grade grid post the service costs

The average grade for the orebody is now estimated at 691cmg/t. This means the grade is reduced by 116 cmg/t to account for the services.

#### 4.7 Final overheads

The final adjustment is for the overheads not accounted for to this stage. These include the following:

- Shaft services;
- Surface infrastructure;
- Surface services;
- Treatment;
- Mine overheads;

- Rehabilitation;
- Off-mine cost;
- Processing;
- Smelting and refinery; and
- Additional ongoing capital.

These are unlikely to vary over the different blocks unless there is a specific reason for varying them. Reasons may include adjustments for stoping width, if the exercise is conducted in square metre units, or density adjustments in the case of platinum.

	A	B	C	D	E	F	G	H	I	J	
1	-1	492	592	692	792	792	1086	1186	1286	1292	1
2	-308	-208	-108	-8	92	171	465	486	586	592	2
3	-355	-255	-155	-55	45	84	386	439	539	545	3
4	-355	-255	-53	-8	92	131	418	486	539	545	4
5	-308	-161	141	181	328	383	341	331	289	295	5
6	-261	-161	81	181	407	507	607	581	539	545	6
7	-261	-161	81	181	328	323	557	581	539	545	7
8	-361	-261	-19	81	181	255	376	350	-86	445	8
9	-461	-861	-666	-566	-466	-366	-266	-1480	-1474	-155	9
10	-861	-761	-566	-466	-366	-266	-166	-66	-61	-55	10

**Figure 4-9: MGO profit grid post the overhead costs**

The above diagram (Figure 4-9) shows the profit grade (expressed in cmg/t) accounting for all costs.

Note that the colours are now different, as they no longer differentiate on the basis of cut-off and pay-limit but now reflect the profit margin. The white areas are uneconomical, with the pink areas representing the ore that is economical but yields less than 10% margin. The red areas are expected to yield between 10% and 30% margins, with the blue areas expected to yield in excess of 30%.

#### **4.8 Risk adjustment**

This step facilitates some adjustment to be made for differential risk in different areas. Not all areas have an equal chance of success, as there may be more complications in certain areas, and these complications may or may not be predictable and quantifiable.

These risks may include the following:

- Adverse ground conditions;
- Geological variations;
- Safety risks;
- Water risks;
- Joint ventures or tributes; and
- Depth risks.

The concept behind this step is based on applying the law of probability to account for the risk differential. Determining the risk is subjective and is discussed in some detail in Chapter 3. The purpose of this adjustment is account for risk on a relative basis.

The following diagram (Figure 4-10) shows the margin in each block adjusted for risk.

	A	B	C	D	E	F	G	H	I	J	
1	-1	492	592	692	792	792	1086	1186	1286	1292	1
2	-308	-208	-54	-4	46	171	465	486	586	592	2
3	-355	-255	-78	-28	22	84	386	439	539	545	3
4	-355	-255	-53	-8	92	131	418	486	539	545	4
5	-308	-161	141	181	328	383	341	331	289	295	5
6	-261	-161	81	181	407	507	607	581	539	545	6
7	-261	-161	81	181	328	323	418	436	270	272	7
8	-361	-261	-19	81	181	255	282	262	-43	222	8
9	-461	-861	-666	-566	-466	-366	-200	-1110	-737	-78	9
10	-861	-761	-566	-466	-366	-266	-166	-66	-61	-55	10
	A	B	C	D	E	F	G	H	I	J	

Figure 4-10: MGO profit grid post the risk discount

#### 4.9 Present value per block

As the sizes of the blocks are known, the potential profit that can be unlocked for each block can now be determined. Each block is 300 X 300m at a known stoping width and density, and the cost profit per square metre or tonne is now calculated for each block.

The following diagram (Figure 4-11) now shows the present value (in real terms) of each block as if they are all mined immediately. It is expressed in millions of Rand profit before tax, for each of the 300m X 300m blocks.

The colouring is different, as the blocks are now shaded according to the profit ranking reflected in quartiles. The dark-blue blocks are above the 3<sup>rd</sup> quartile and thus represent the immediate targets. The medium-blue blocks are the next best, above the second quartile, where the light-blue blocks are below average (first and second quartile blocks). The uncoloured blocks are not economically viable.

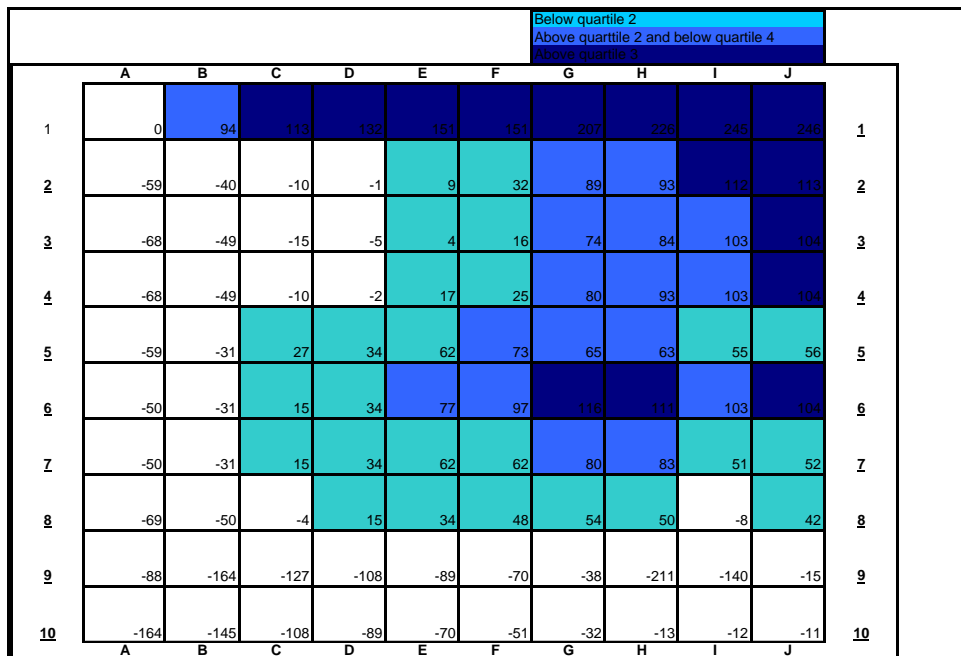


Figure 4-11: MGO NPV grid

Profits can now be visualised in two dimensions, and this is an extraordinarily useful tool for any planner.

#### 4.10 Timing

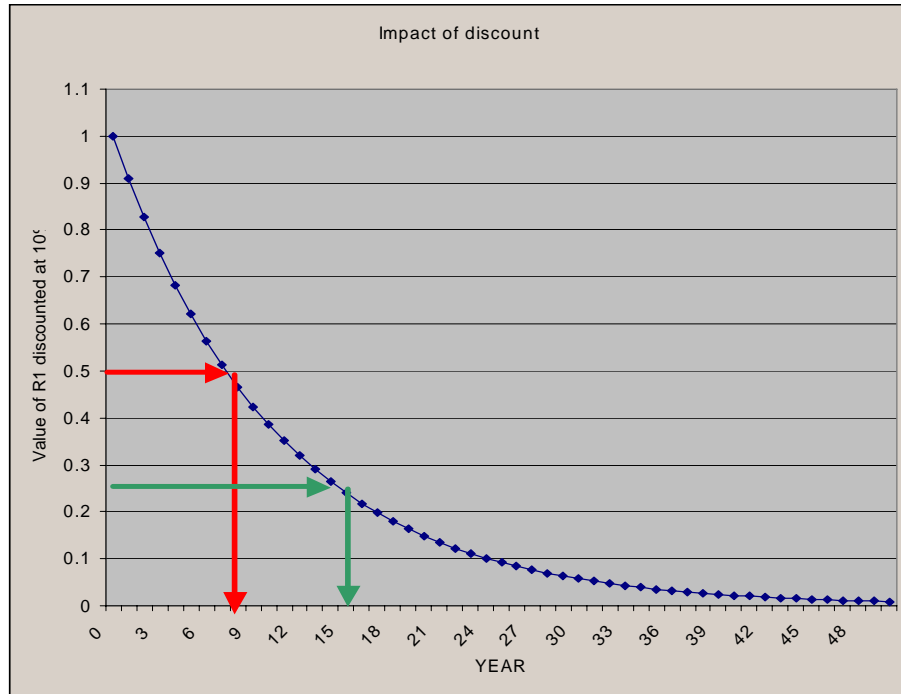
The way we measure the value of any project is by the net present value of the future real free cash flows. This is an internationally acceptable practice. The supporters of option pricing and real

options are currently challenging this logic. Nonetheless, it is difficult to fault the use of discounted cash flows, and the logic associated with the NPV calculation (time value of money) lies at the heart of this step.

The use of NPVs suggests that investing money in a project has a 'cost of capital' associated with it. The opportunity cost of money is brought into the equation in the form of the discount rate that reflects the company's weighted average cost of capital. This discount rate is in principle a function of the risk-free return on cash adjusted for technical and political risk associated with the projects.

Because of this discount, every year that profits are delayed causes further destruction of value. To optimise the NPV, everything should be mined in one day. This is obviously beyond reality and a plan needs to be developed to expedite the mining of the highest profit areas as quickly as possible.

The following graph (Figure 4-12) shows the impact of the discount rate on a Rand of profit discounted over time. From the graph it is clear that every Rand made in year 9 has only 50 cents' impact on the NPV. This is halved again by year 16, and any value generated beyond year 30 has almost no impact. For this reason, the best reserves should not be mined late in the life of the operation.



**Figure 4-12: Discounted Rand**

In order to determine the net present value for each block, all that needs to be done is to discount each block for timing and estimate what percentage of each block will be extracted. The summation of the discounted values of all the mined blocks in the lease will approximate the NPV of the mine before taxes and cost of finance.

The MGO model, based on the above process, can be created in a relatively short time and is used to determine the best mining strategy, which in turn determines the appropriate development programme. This plan or macro schedule is then handed over to the planner, who can turn it into a more accurate reality.

The following diagram (Figure 4.13) shows which blocks should be mined and what the sequence is. Note that the 300m X 300m block



equates to 90,000 square metres and will probably be extracted over three years. The 'middle' year's discount is allocated to this block. It is also likely that there will be more than one attacking point and that sequence could be modelled in the MGO. The year in which this block is mined is reflected in each square.

	A	B	C	D	E	F	G	H	I	J	
1	22	21	19	18	12	10	9	11	13	19	1
2	0	23	20	18	13	8	7	5	6	18	2
3	0	0	21	15	11	4	3	3	5	15	3
4	0	0	21	17	7	2	1	2	4	6	4
5	0	0	22	15	8	7	3	4	5	14	5
6	0	0	20	16	14	9	6	8	12	17	6
7	0	0	23	20	14	12	9	10	13	16	7
8	0	0	0	0	22	19	10	11	16	17	8
9	0	0	0	0	0	0	0	0	0	0	9
10	0	0	0	0	0	0	0	0	0	0	10
	A	B	C	D	E	F	G	H	I	J	

**Figure 4-13: Schedule grid**

Moreover, there are logical constraints as well as rock engineering constraints that have to be honoured in the planning process. These constraints must be kept in mind when the macro schedule is completed.

In a new operation, this process can also be used in conjunction with other methods to determine the optimal economic position of the shaft. Existing operations have less flexibility but definitely hold potential for grade optimisation.

The optimal NPV is determined by changing the schedule iteratively until the optimal NPV is achieved. The next section discusses other methods that could be used to assist in the decision making process.

The following diagram, Figure 4.14, shows which blocks are mined and what the NPV is expected to be. The best solution for this area can generate an NPV of R1.809 billion if the sequence proposed in the diagram is achieved. The green blocks represent the blocks that will not be extracted. The red blocks are the uneconomical blocks that will be extracted as a result of the required mining patterns, as well as to maintain volumes.

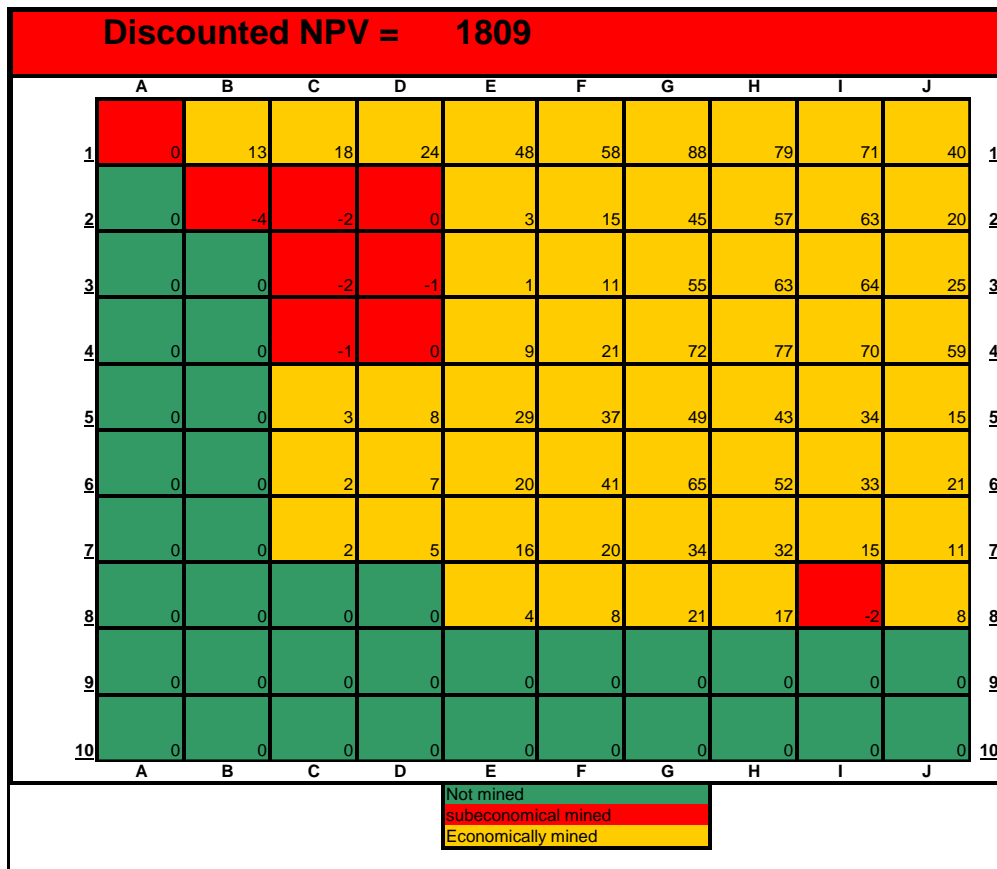


Figure 4-14: Final NPV grid

## 4.11 Additional methodologies

### 4.11.1 Moving averages

There are several schools of thought on methodologies used to determine an optimal approach (for example moving averages and linear programming). The best is probably still good logic and the diligent use of the 'eyetometer' (the process of visual inspection). Good logic tells one to choose the large areas of high quality tonnage concentration, and a moving average can be used to simulate a "floating cone" as used in some software. This exercise uses a 9X9 block moving average to determine where the 'hotspots' are. The outcome is visible in the next figure. The primary targets are blocks I and H, 1 and 2, as well as G and J, 2.

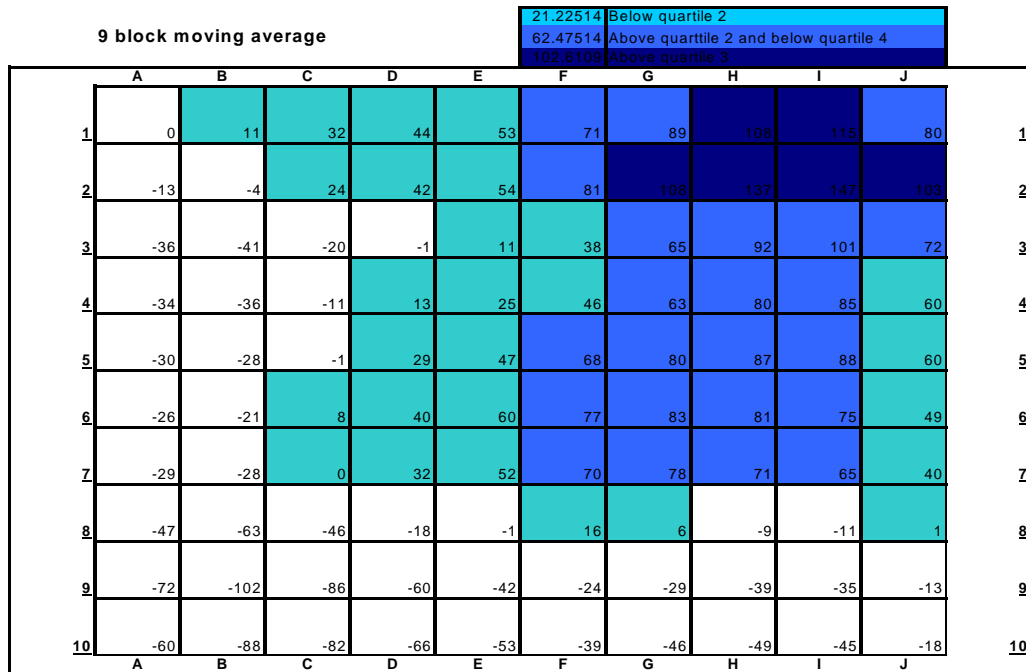


Figure 4-15: 9 x 9 moving average grid

#### **4.11.2 Linear programming**

The use of linear programming, if combined with the macro grid optimising process (MGO) holds significant potential. The use of linear programming has not been addressed for this research. It will, to all intents and purposes, marry the optimisation of the infrastructure with the optimisation of the orebody.

#### **4.11.3 Stochastic methods**

The current thinking is also moving away from the single outcome (single point) methodologies and is moving to stochastic methods as described in Chapter 2 of this research. The input variables are converted to input distributions and the models are run multiple times to determine the risk profile and most likely outcomes. There is a significant research focus on such methods in progress at the WH Bryan Mining Geology Research Centre (BRC), The University of Queensland, Brisbane, Australia.

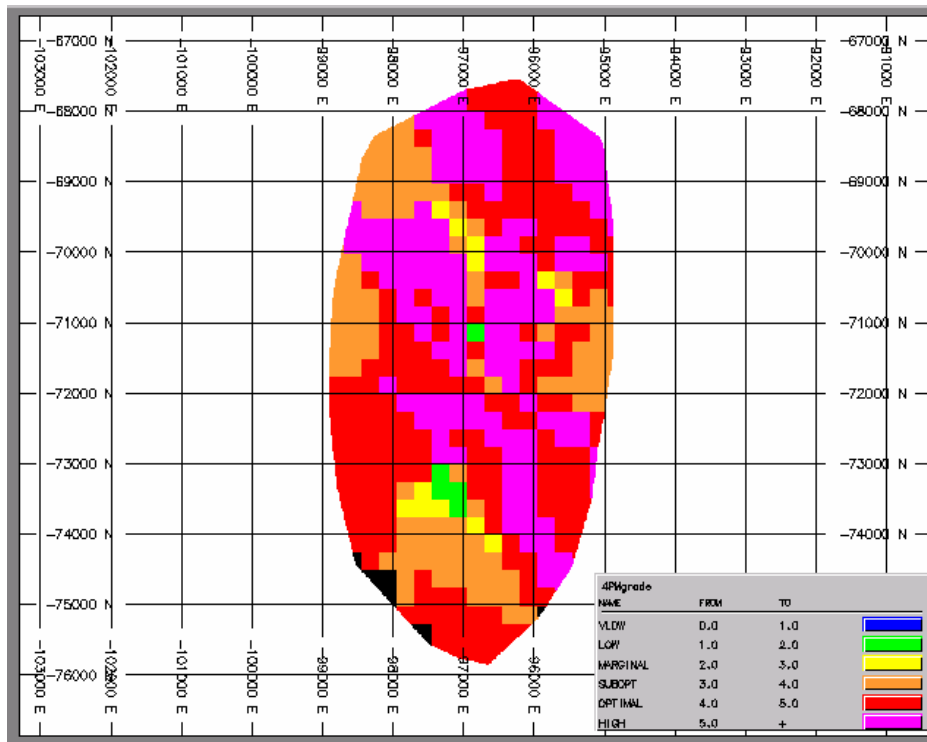
The methods described in this research are likely to fit well with the Queensland University's thinking and significant synergies may be possible if efforts could be joined. The Queensland focus is on determining uncertainty and optimisation in ore reserves and mine planning, using Stochastic Integer Programming (SIP) methods.

#### **4.12 Practical application**

Some of the processes as described above have already been used in high-level decision making using Datamine software. One problem

associated with a multi-product environment is the impact of the other products on the primary product.

The following diagram (Figure 4-16) reflects the grades of all the products combined. However, product "A" is far more valuable than product " B" and thus the combined grade is actually meaningless.



**Figure 4-16: Combined grade of all products**

The above diagram is the product usually supplied by the geologist and geostatistician. It may satisfy their needs but is totally useless to the mine planner. He needs to understand the profitability (locked up in the resource).

The following diagrams (Figure 4-17) show the grade associated with product "A", "B" and "C".

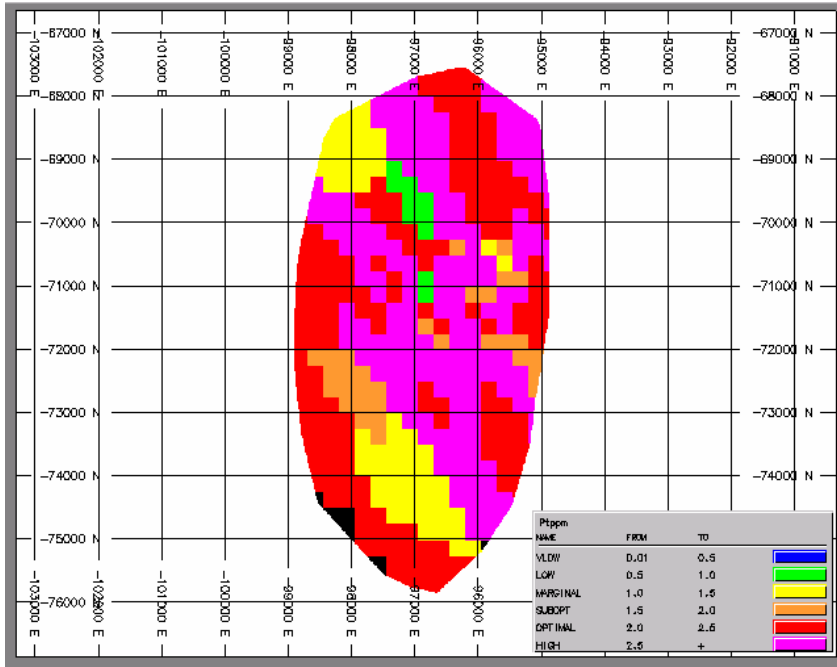


Figure 4-17: Product A's grade

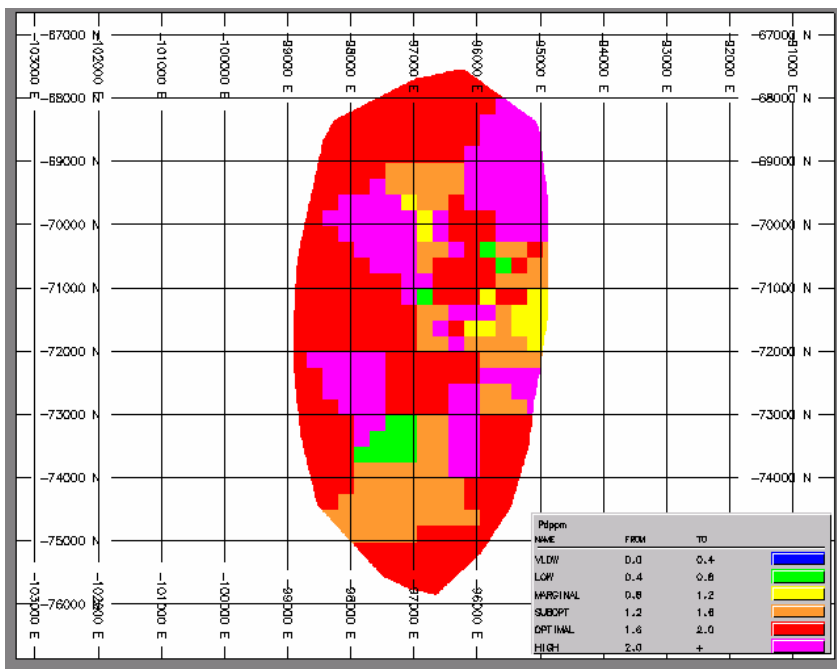


Figure 4-18: Product C's grade

There are two ways to overcome the problem. The first method is to convert the grades of the secondary products to the primary product using the equivalent methodology as described in Chapter 2.

The second method is to convert all the grades to revenue per tonne, taking cognisance of the metal prices, exchange rate, recoveries and transfer agreements. The mining and process cost per tonne is then subtracted from the revenue per tonne and the result is the profit per tonne for all the products are combined in a single picture.

On the contrary, you could have different profiles for different metal prices or different exchange rates.

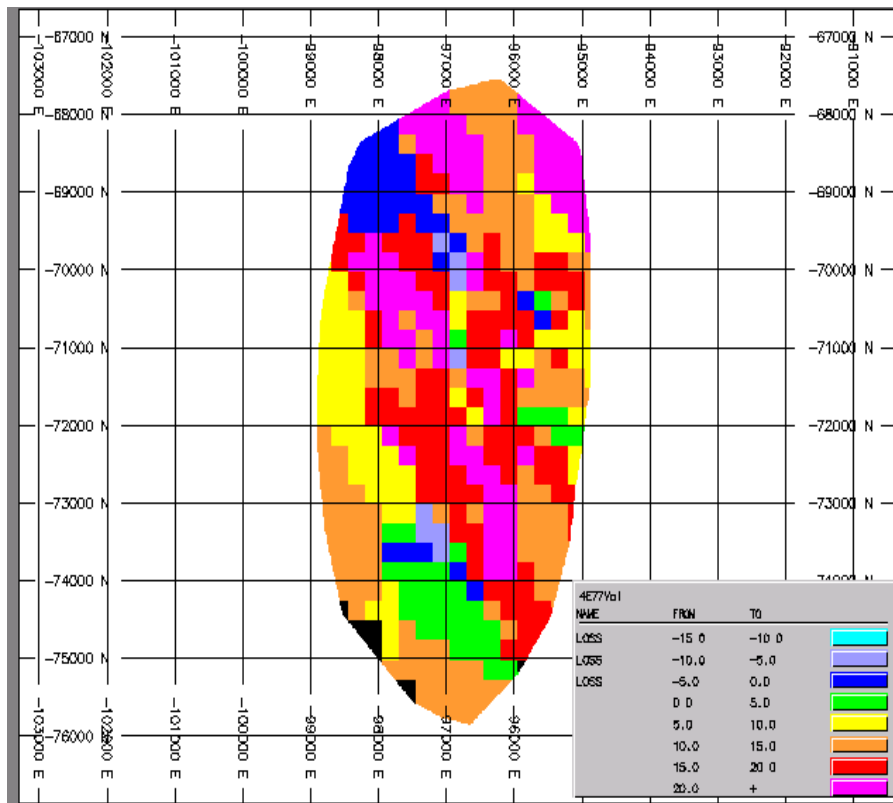


Figure 4-19: Profit grid

This macro grid optimiser process' profit grid is far more useful for a planner than multiple grade grids. Moreover, the geostatitcian and geologist will also have a better understanding of the orebody from a business perspective, which would be beneficial to the company.

#### **4.13 Prospecting tool**

As recent application, this macro grid optimiser methodology was used to determine target areas for the prospecting of the base metals. Several grids were developed to account for:

- Trimming distances to the plant;
- cost of overburden;
- distance from infrastructure;
- cost of treatment; and
- cost of transport.

These grids were tied to the topography of the area where the prospective orebodies could be found. The costs were calculated for each variable and the grids were summised to give an overall cost per tonne for each block. These blocks each reflected an area of one kilometre by one kilometre. This gave a very clear indication of where to look for potential economically viable orebodies. It is useless finding a small orebody, under a large amount of overburden, and/or a long distance away from the plant. It is not likely to be economically viable. So why look for uneconomic orebodies?



Using the macro grid optimiser, the prospecting areas were reduced to areas where the chances of economic extraction of the orebody is better. This methodology is currently being applied in Namibia in order to prospect for zinc orebodies.

#### **4.14 Conclusion on MGO**

The current planning process requires several days for an exercise to be prepared and adjustments are often not properly executed. The favourite method is to 'tune up' the Mine Call Factor until an acceptable profit is achieved. This is a dangerous practice as the planning credibility is jeopardised in the process.

The problem with the current planning systems is that the planner tries to stretch the life of the mine, and the long-term plan usually follows a sequential process, without any major interventions to ensure optimised outcomes.

The MGO process affords the ability to generate many plans in a relatively short timeframe. This exercise was conducted using EXCEL to demonstrate the principle but should in reality be adapted to the 3 D graphics environment as partially demonstrated in the first practical example that deals with multi-product orebody.

These MGO scenarios are not very accurate but do allow the best strategy to be selected for the detailed plan that will follow and hence unlock the maximum realistic value from the orebody.

## 5 MINING ECONOMICS RISKS

- *Overview*
- *Use of the 'S' curve to determine risk*
- *Trade-off studies in feasibilities including:*
  - *Depth of shaft*
  - *Optimum volumes*
  - *Number of levels*
  - *Optimal strike length*
- *Conclusion on mining economics*
- *Overall conclusions on the research*

### 5.1 Overview

There is an overlap in risk management processes between minerals resource management and mining economics. This research is essentially focused on the risks associated with the optimisation of orebodies, which is discussed in detail in Chapters 2 to 4. However, there are risk management processes that are very specific to the mining economics domain. A selection of these processes is discussed at varying levels of detail in this chapter.

Different tasks have different risk exposures and different processes to address these risks. The processes to address risk in mining economics could be categorised as follows:

Cash flow component analysis, is discussed in some detail, as it is a relatively new school of thought.

Mergers and acquisitions are usually based on high-level public-domain data. The risk associated with Mergers and Acquisitions (M&A) is usually mitigated by conducting due diligence exercises and is not discussed in this research.

Trade-off studies conducted in feasibilities to facilitate optimal design.

The following is discussed in some detail:

- Optimal depth of shaft;
- Optimum volumes;
- Optimal number of levels; and
- Optimal strike length.
- Benchmarking processes, which are only touched on.
- Technical and financial modelling, which is only touched on.

## **5.2 “S” curve optimisation**

The use of discounted cash flows is an internationally acceptable valuation methodology and there are four critical elements in the cash flow, namely:

The production profile, which is a function of converting resources to reserves;

- The cost associated with the extraction plan; and
- The market assumptions in terms of prices and exchange rates.

- The timing of the inputs.

The discounted cash flow methodology basically accounts for the time value of money and the opportunity cost of the investment. This Chapter focuses on the key drivers of the discounted cash flow and how they should be optimised to unlock maximum value from the orebody.

### **5.2.1 Cash flow type**

Unfortunately, a discounted cash flow is a singular outcome, based on the combined impact of several variables. The only guarantee one has with a singular outcome analysis is that it is true only for the assumptions adopted in the valuation. However, it is a best guess and thus may not be totally wrong. This method holds noteworthy merit, especially if the associated risks are understood and have been accounted for.

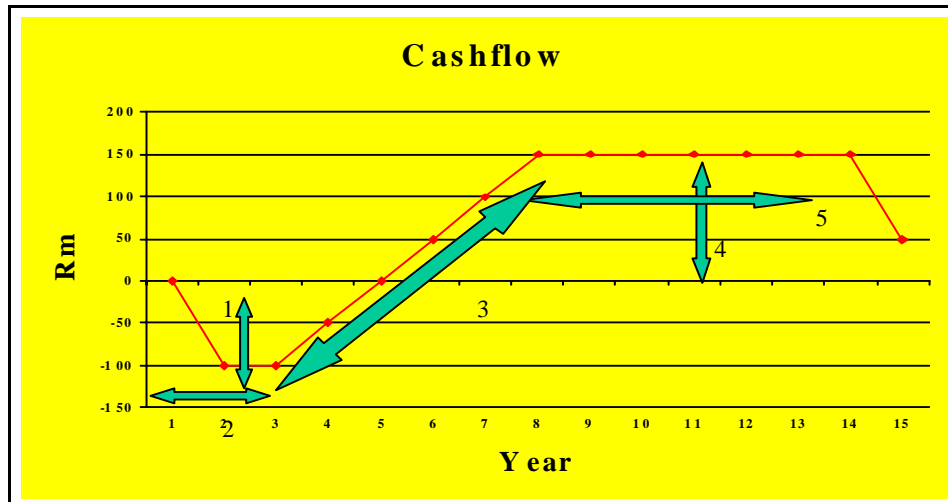
The issues related to single solution options could be overcome by using stochastic analyses (this is discussed extensively in Chapter 2). In the stochastic process, the key variables are identified and their sensitivities are tested. Not all variables have a major impact. In addition, some variables do not have a significant range of variations. It is important to understand the variations of the assumptions, as well as their relative importance. The utilisation of Monte Carlo simulations is beneficial in determining these relationships. The

principles discussed in this chapter are equally valid for Monte Carlo type cash flows.

A further derivative of the above stochastic method is to make allowances for managerial intervention, which essentially is an option analysis. The detail of this is not discussed in this research. The basic principles are the same as with a singular outcome analysis, the only difference being the introduction of multiple iterations.

A Mining Economist tends to get involved in projects on an ad hoc basis. The key issue for a Mining Economist is to understand the impact of any decision on the operation's profitability. The mining economist wants to know what the NPV and IRR of the investment are before any decision is made. In order to determine the risk encompassed in any variable, one needs to determine the impact that the specific risk has on the 'bottom line'. Simplistically, therefore, the impact on the 'bottom line' is probably the most important measure of the risk.

A study of a generic cash flow has identified five critical items related to cash flow, as can be seen from the following sketch in Figure 5.1:



**Figure 5-1: Generic cash flow**

1. Capital outflow;
2. Timing of outflow;
3. Build-up;
4. Annual profit at designed capacity; and
5. Life of the project.

This methodology of analysing a project is now an accepted standard procedure for AngloGold Ashanti and is referred to as "project DNA". This method of evaluating the value of an operation has also been used in countless reviews in Anglo Platinum to ensure the focus is on the items that add value. This process is also used as a tool in Anglo Technical Division (ATD) in order to review new projects or as a tool during consulting on feasibility studies.

### 5.2.2 Capital outflow

This element (item 1 of Figure 5.1) is a function of the capital expenditure, which in turn is categorised in the following areas of risk (confidence). This confidence in the capital out flow is usually measured and quantified in terms of the **class of the estimate**:

The risk variables related to capital out flow can be classified into two areas namely:

- Technical issues; and
- Financial issues.

The risk associated with this section of the cash flow is a function of the risk embedded in the capital programme. As a rule of thumb, 'The more you spend the more the exposure to risk'.

In order to quantify the risk associated with these exposures different categories of confidence in capital have been designed and the following reflects the typical classification used in Anglo.

- Class 0  $\pm$  30%
- Class 1  $\pm$  20%
- Class 2  $\pm$  10%
- Class 3  $\pm$  5%

It is unfortunate that there is no international classification system for capital expenditure. Moreover, several companies have identified the need for a system of classifying Capex. In order to mitigate this risk, it is common practice to allow for a contingency, based on the

class of estimate. In order to determine the correct contingency, the following is considered.

There are two uncertainties in terms of the capital estimates, namely the uncertainty of the estimate and the uncertainty of the technical design. The uncertainty of the estimate could be determined using Monte Carlo simulations and the overall estimation risk is determined from the compounded effect of the estimation risks embedded within each of individual variables.

The second contributor to risk, namely the technical uncertainty is more difficult to quantify. Certain elements may be proven technology, with tried and tested designs and with historical cost estimates. Other elements may be new "Pie in the Sky" technology with little or no previous benchmarks. These carry a significantly greater risk. There is no definitive methodology to describe a risk value to these uncertainties and any estimate will be likely to be subjective.

The following philosophy is generally applied in the Anglo projects in order to mitigate the technical risk: - **"Old technology for new projects and new technology for old projects"**

What is missing in most capital estimates is the sensitivity around elements in the estimate. It is suggested that in addition to the normal sensitivities an item specific - sensitivity be created. The table (Figure 5.2) that follows may facilitate such a risk analysis.



ITEM	RAND M	FOREX	MIN	MAX
Pumps	7,5	10%	7,2	7,8
Fans	15,2	80%	15,1	15,3
Cooling System	35,8	100%	20	45
Piping	120,0	5%	110	125
<b>TOTAL</b>	<b>178,5</b>			

**Table 5-1: Item specific risk analysis**

As can be seen in the table there are essentially two elements to the cost: namely, design criteria and foreign exchange components. It is possible to incorporate the use of, for example, triangular distributions with all risk to determine the overall risk profile of the capital expenditure. These distributions are then used in the Monte Carlo simulations. Distributions representing currency fluctuations can also be considered.

Table 5-1 suggests that there is a significant risk embedded in the cooling system. Special attention needs to be given to the accuracies of this estimate.

In terms of the forex component the impact of changes in the exchange rate is then determined. Derivates are often put in place to minimise the risk on major items with a large Forex exposure, in order to mitigate this risk.

If the investment is destined for a country with high political risk, insurance is usually put in place to mitigate this risk. The country risk determines this premium (usually between 2% and 5% of the Capex value). A case where a political insurance case was settled was where Delta Mining was paid out for a lost investment in Papua New Guinea. The company was fully compensated but had to sacrifice 50% of the mining rights to the insurance company.

### **5.2.3 The Timing of the Outflow**

This element (item 2) of the cash flow (Figure 5.1) is a function of the timing of the capital expenditure. The key considerations for this component are:

- Critical path; and
- Project management.

A good project management control system, with proper project flows and critical path analysis ensures that risks are kept to the minimum. There is usually very little in place in most projects to ensure that the timing of expenses is optimal in order to improve the NPV and the IRR. The constraints usually considered are the availability of capital funding and tax shields.

It is unfortunate that the higher the discount, the greater the impact on the NPV. It is unlikely that any deep gold mine (with up to 15 years to achieve full production) will ever deliver returns robust enough to withstand the erosion of high discounts.

Open-cast operations are often less sensitive to the impact of discounting due to the relative quick returns.<sup>8</sup> In underground mining, there are often attempts to generate income up front through pillar extraction programmes and pre-development exercises. It is important for this to be evaluated very carefully as value could be destroyed if the extraction of the pillar delays the primary income of the orebody.

It is important that macro economic exercises are conducted to optimise the shaft from an orebody perspective as well as a cost perspective. (See the section 5.4 on trade-off studies).

#### **5.2.4 Build-up (Time required to get to full production)**

This phase (item 3 of Figure 5.1) is essentially controlled by the mine design, which in turn is affected by the following categories of risk (confidence):

- Micro design within Macro design; and
- Micro schedule within the Macro schedule.

The **macro design** is also discussed in Chapters 3 and 4. The focus of the risk analysis is on the mine design, the layouts, and the

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<sup>8</sup> Toll, G.L. (1994). 4<sup>th</sup> Large Open Pit Mining Conference, Perth 5-9 September 1994. pp 281 to 283. *Mining Risk – Perceptions and Reality*.

efficiencies associated with the layouts. Several mines are paying a price in the long-term because the mine design was established 'lean and mean' and cannot afford the flexibility to ensure good efficiencies. The price is paid in low efficiencies, which are translated into high operating costs. Very little work is done on measuring efficiencies built into mine layouts.

On the other hand, some mines are totally over designed and there is often a mismatch in capacities, which is capital inefficient. In fact, the 'debottlenecking' exercise often contributes significant value with limited investments.

In terms of the **micro design**, the schedules associated with the build-up holds significant risks. These should be scrutinised very carefully as many of the new projects do not reach their targeted build-up. The impact on the return is usually significant. The use of stochastic modelling in projects leads to an improved understanding of risk associated with the schedules.

#### **5.2.5 Plateau (Full production level)**

This (item 4 in Figure 5.1) is often the most critical phase of the cash flow curve. This phase starts when full production is reached and is repeated year after year until the reserves are depleted. There may be some deviations as parts of the orebody may be different. The factors that influence this phase are a function of technical issues and financial issues:

### **Technical issues**

- Mine design;
- Tonnage planned;
- Mining method;
- Grade including dilution;
- Optimisation;
- Evaluation of the ore body;
- Factors (MCF, Recovery, BF);
- Legislation; and
- Infrastructure capacity, condition.

### **Financial issues**

- Price, escalations, marketing assumptions;
- Cost assumptions, benchmark, equipment, labour, efficiencies;  
and
- Cost volume curves (optimisation).

Some of the methodology to mitigate risks in this phase is covered in Chapter 2, where the concept of stochastic analysis is discussed.

#### **5.2.6 Life of mine including the tail-off**

This element (item 5 of Figure 5.1) of the 'S' curve determines the number of times production, as reflected in the plateau, can be

repeated. It is essentially the conversion of resource to reserves.

This element is affected by risks associated with issues including:

- The Life of Mine plan (the accuracy and reliability of the Cadamine, Datamine, paper and pencils plans)
- Optimisation as discussed in previous chapters;
- Blue sky potential that may or may not be considered.
- Closure / rehabilitation programmes and provisions;
- Rebuilds of equipment and re-establishing infrastructure; and
- Environmental impact.

The risk associated with the conversion process is usually well covered if sufficient attention is afforded to the SAMREC code and governmental legal requirements. Needless to say, Chapters 3 and 4 on optimisation using the CBO and MGO will add significant value to this section.

### **5.3 Models**

Mining Economics work is usually conducted at lower levels of detail and at higher levels of strategic thinking. In terms of modelling risk, it is often difficult to decide what level of detail should be included. A study of models was conducted during this research, which suggests that the models tend to be far too complex. In addition, it was found that approximately two thirds of spreadsheet models have flaws, with around one in ten having material flaws. The Mining Economist

needs to have an overall knowledge of the risks encompassed in the model and their impacts. He needs to adjust the level of modelling to suit the needs and the risks of the decision at hand

The main sources of errors in modelling are attributable to:

- Inappropriate models;
- Incorrect formulas;
- Time value of money issues;
- Hard-coded items in models;
- Links to other models not functioning;
- Inflexible mainframe systems;
- Expensive systems;
- Wrong logic;
- Modelling earning rather than free cash flows;
- Tax issues;
- Production profiles that often stay static at different price assumptions, which is probably wrong;
- Technical and financial items not linked;
- Outdated data;
- Inexperience and lack of understanding of economic concepts like time value of money;

- Double agendas of project leaders;
- Published reserves statements not matching profiles in models;
- Projects not modelled on incremental basis; and
- Multi currency complications

#### **5.4 Trade-off studies in feasibilities**

##### **5.4.1 Overview**

Trade-off studies were conducted on many operations but unfortunately their detail cannot be published. In order to demonstrate the principle a typical (but fictitious) platinum operation was modelled. These studies attempt to illustrate how these trade-offs are approached. They are essentially based on the time value of money and cover the following concepts

- Depth of shaft;
- Optimum volumes;
- Optimal number of levels; and
- Optimal strike distance.

The mineralisation under consideration is the platinum group of elements (PGE) associated with the tabular orebodies of the Merensky Reef and/or UG2 chromitite layer.



## **5.4.2 Overall Assumptions**

Generic exercises were conducted to determine the optimal depth and volumes for a vertical shaft, and the strike length for a surface decline. These exercises used the standard 'S' curve to distribute capital used in the industry. The opex is flexed according to volumes to account for losses and gains resulting from the scale of benefits.

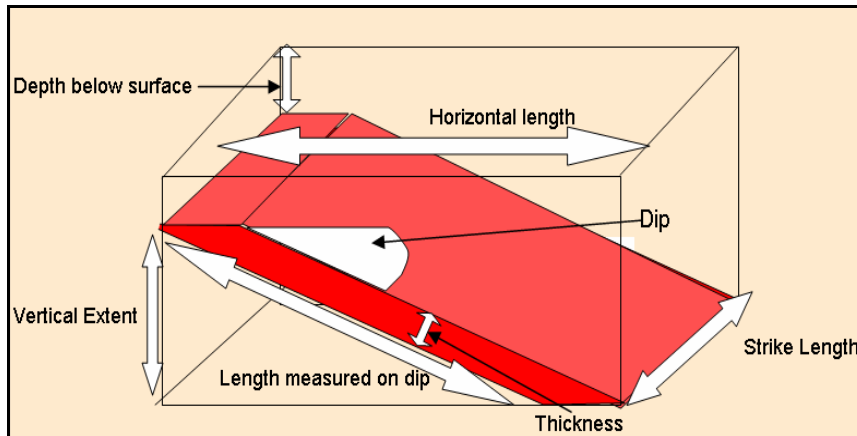
Although the exercises are theoretical, the inputs to the models are based on actual figures where available, in particular the project capex estimates, the duration and operating costs.

The assumptions were also based on actual figures as much as possible; hence the models can be used as a basis for more detailed work on actual operation or projects.

### **5.4.2.1 Orebody Assumptions**

The PGM resource is assumed to lie between at least 1000m below surface and surface. The length (measured along dip), vertical extent and strike length were some of the variables that were altered to give a different reserve for the various models (Figure 5.3).

The thickness of the orebody was kept constant at 1m, and the in situ grade was set at 8.5 grams per tonne of four PGE elements, namely platinum, palladium, rhodium and gold. The dip of the orebody was kept fixed at 18 degrees for all the exercises.



**Figure 5-2: Idealised orebody.**

#### **5.4.2.2 Capex and Opex Assumptions**

The capital expenditure figures used in the exercises cover the development of a generic new mine, in a green-field environment. It is assumed there would be access to power, water and road services nearby.

The mine consists of a vertical shaft with surface infrastructure including refrigeration but excludes a process plant. The cost estimate meets the Anglo American class 0 capital cost estimate classification (>25% and >-15%). The estimate is derived from recent projects.

The models were constructed using a start date for development of January 2005, in order to allow for a yearlong feasibility study.

### 5.4.3 Shaft Depth and Volume vs NPV

#### 5.4.3.1 Exercise assumptions

The aim of this exercise is to determine the best mining volumes for a Bushveld PGM mine. Six different shafts are modelled ranging from a 1350m deep, 100 kilotonne per month shaft to a 2050m deep, 350 kilotonne per month shaft. The shaft(s) were assumed to be centrally located over the orebody, the production levels are assumed to be 70m apart vertically, and it is assumed that each level produces 25 kilotonnes of ore per month.

The total reserves available were determined by the 6km strike length and dip length for each shaft; hence the depth and volumes determine the life of mine.

	Max Depth (m)	Number of Levels	Project Time (months)	Total Project Capex (Rand Billion)	Shaft Head Delivery Costs (Rand per tonne)
Shaft 1: 100ktpm	1350	4	64	2.45	288.00
Shaft 2: 150 ktpm	1490	6	71	3.02	216.00
Shaft 3: 200 ktpm	1630	8	78	3.60	180.00
Shaft 4: 250 ktpm	1770	10	84	4.18	158.40
Shaft 5: 300 ktpm	1910	12	91	4.75	144.00
Shaft 6: 350 ktpm	2050	14	98	5.33	133.71

Table 5-2: Attributes of Shafts Modelled

The shaft head delivery costs and indirect on-mine costs varied with the different production volumes and shaft depths (Table 5.4 and

Figure 5.5). Other operational costs such as refinery and smelting costs (in Rand per tonne) were the same for all the shafts.

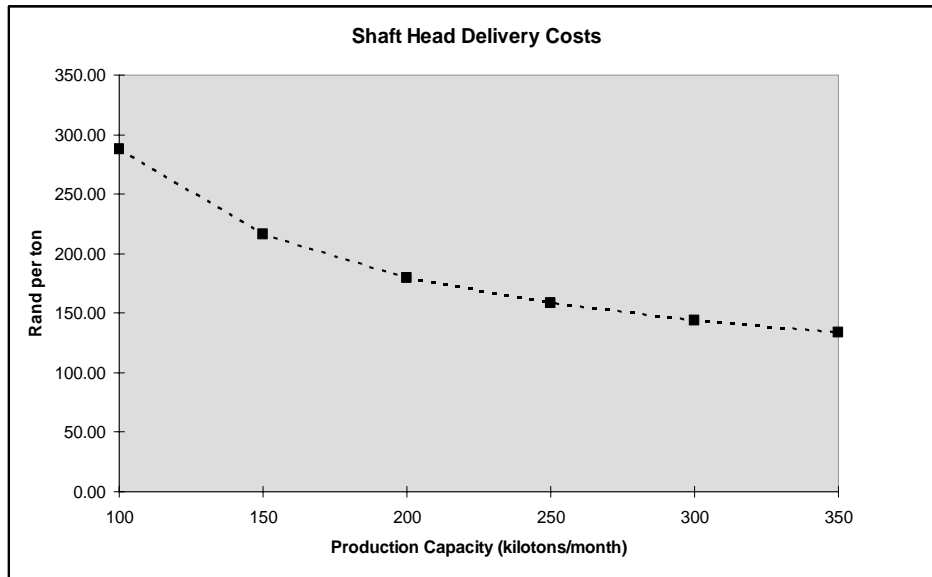
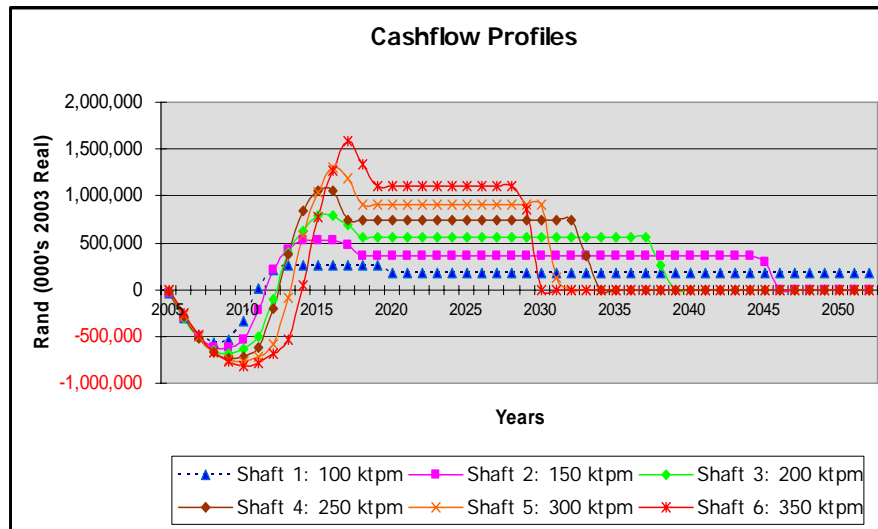


Figure 5-3: Shaft head delivery costs

#### 5.4.3.2 Results

The lower production shafts have a longer life of mine, and lower capex, but are not optimal in terms of NPV and IRR. See Figure 5.6 to compare the cash flows for the different options.



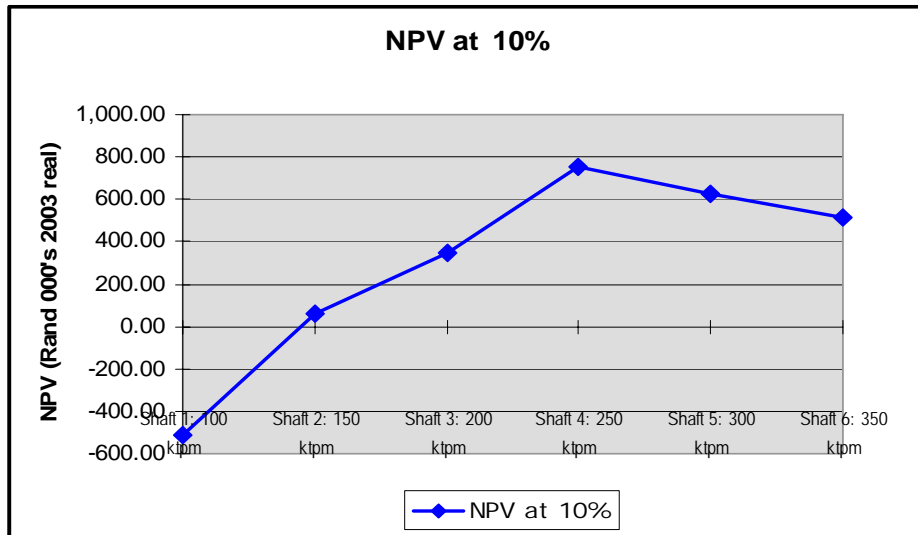
**Figure 5-4: Cash Flow Profiles for 6 different shafts.**

The optimal shaft is the 250-kilotonne per month option, which is 1.8km deep. It has the highest NPV and IRR and a life of mine of 30 years.

The NPV (Figure 5.7) of the project is very sensitive to changes in volume below 250 kilotonne per month and less sensitive above 250 kilotonne per month (250ktpm).

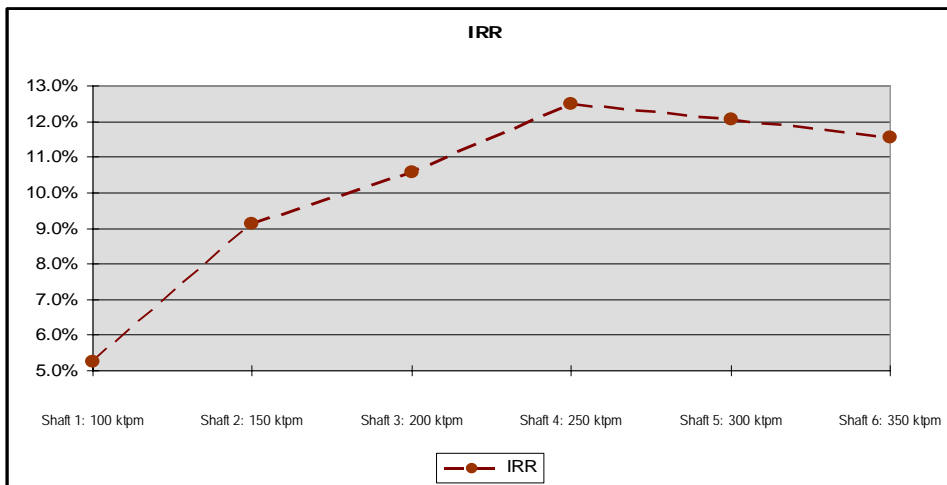
The NPV increases at a reduced rate above 250ktpm as a result of diminishing returns associated with economies of scale. In other words, the bulk of the benefits are realised at 250ktpm. Moreover, the risk increases as volumes increase and, as such, the probability of sustained success is less likely. Applying the law of probability to this equation, the 250ktpm return would in all likelihood be the best.

In summary, it appears that economies of scale are achieved at about 250 ktpm and the technical risks are acceptable.



**Figure 5-5: NPV at 10% of 6 different shafts.**

The IRR similarly peaks at the 250 kilotonne per month shaft at about 12% and starts to drop at the higher production levels. The change in gradient on both the NPV and IRR graphs at 150 ktpm is due to a change in the gradient of the cost curve (Figure 5.8).



**Figure 5-6: IRR of 6 different shafts.**

### 5.4.3.3 Conclusions

The optimal production level using the assumptions outlined was found to be 250ktpm from a 1770m deep shaft with a life of 29 years.

Production levels greater than 250ktpm require a larger capex, as the shaft has to be deeper and the production build-up is delayed, which adversely affects the value of the project.

### 5.4.4 Number of Levels/Depth versus NPV

#### 5.4.4.1 Exercise assumptions

In the second exercise the production level is kept fixed at 250ktpm and the number of levels was increased from 8 to 16 and, hence, the depth of the shafts and available ore reserves increased in line with the number of production levels. (See Table 5-3 below for the detail)

The aim of the exercise was to determine the optimal number of levels for this type of orebody.

Capacity (ktpm)	Number of Levels	Shaft Depth (m)	Project Duration (months)	Ore Reserves (000' tonnes)	Capex (Rand Billions)
250	8.00	1630.00	80	31,518	3.95
250	10.00	1770.00	83	40,520	4.10
250	12.00	1910.00	85	49,523	4.50
250	14.00	2050.00	88	58,525	4.75
250	16.00	2190.00	91	67,527	5.10

Table 5-3: Optimal levels for a 250 kiloton shaft

### 5.4.4.2 Results

At eight levels the capex is lower and the production build-up is much faster, but the mine has access to a smaller reserve, hence the life of the mine is shortened. However, due to the time value of money, the NPV (Figure 5-8) of the shorter life mine is greater, because the additional cash flows 20 to 30 years from now are discounted heavily and have a negligible effect on the NPV. (See Figure 5-7 for a comparison of the cash flows.)

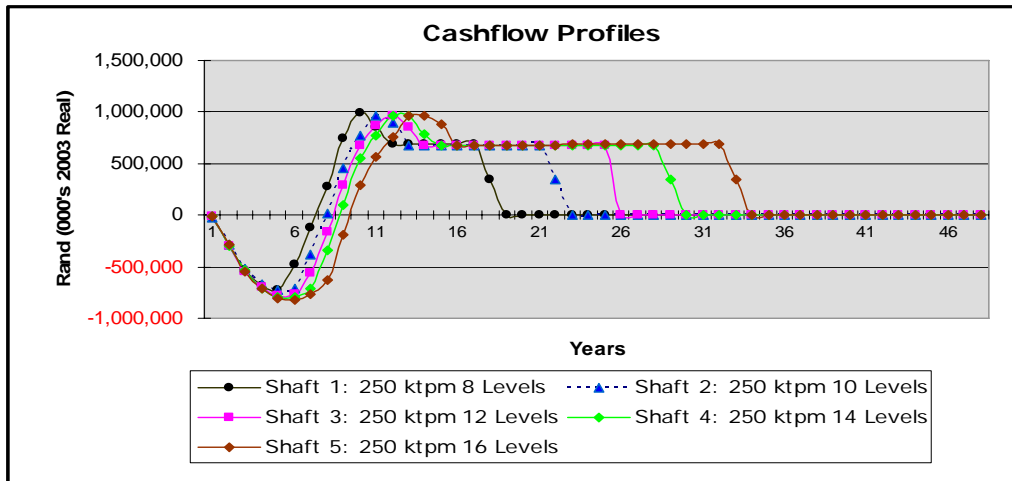
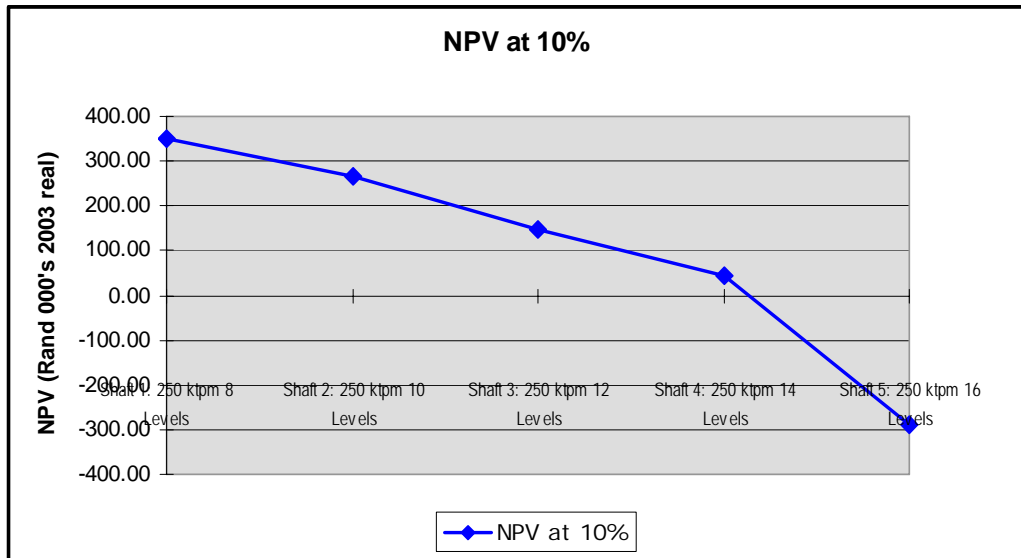
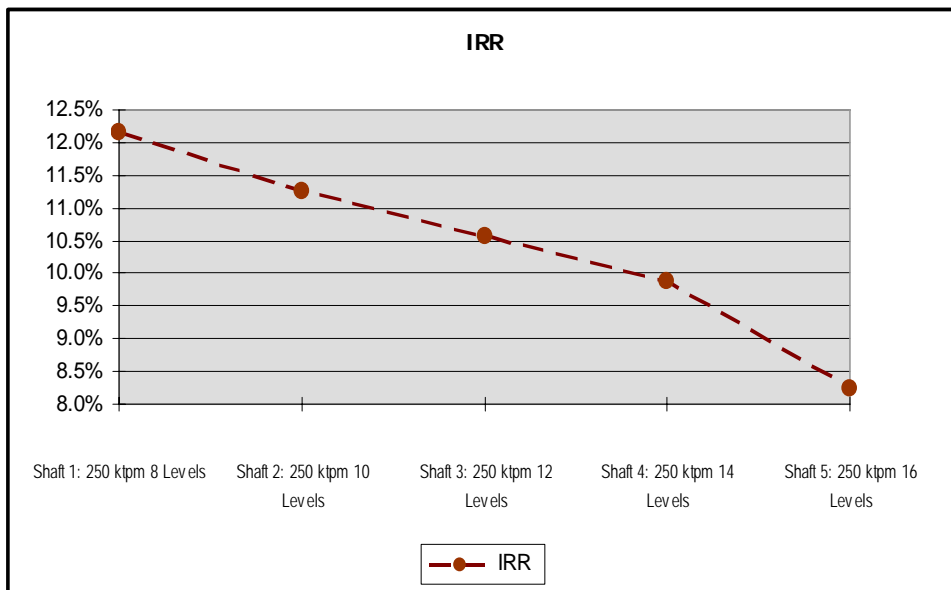


Figure 5-7: Cash Flow Profile





**Figure 5-8: NPV at 10%**



**Figure 5-9: IRR**

#### **5.4.4.3 Conclusions**

The standard requirement for a 250-ktpm shaft is 10 levels. If the shaft is sunk deeper than 10 levels, the operational risk reduces but a penalty is paid in terms of capital and timing.

It appears that roughly 2% of IRR (Figure 5-9) is sacrificed for every level added without the immediate benefit of additional production.

This analysis suggests that emphasis should be placed on designing a shaft that can produce more than the standard 25 ktpm per level per month. This may include mining UG2 and Merensky simultaneously.

These exercises may also include sinking the shaft in stages, utilising a deeper ventilation shaft as a secondary sinking base. This staged approach will allow the infrastructure to unlock the resources but the cost of unlocking such resources will also be delayed.

#### **5.4.5 Decline Mining Options**

##### **5.4.5.1 Exercise assumptions**

The aim of the third exercise is to determine the optimum strike length that can be mined using a decline shaft. In order to do this the NPV from a set of decline shafts was modelled. The decline shaft extends to a vertical depth of 500m below surface and costs R750 million over a project time of 35 months. The capex estimate is based on an existing detailed study. The capex covers the cost of

developing a twin decline system for belt and materials, a ventilation shaft and ore reserve development.

The production level of the decline is set at 150 ktpm, i.e. with six levels, each producing 25 kilotonnes of ore per month. The strike length of orebody available to be mined is the key variable in this exercise and varies from 1000m to 6000m, in 1000m increments. The decline shaft is assumed to be located over the centre of the orebody.

The decline shaft is assumed to dip at the same angle as the orebody, 18°, and would be equipped with a winder. The capital expenditure does not include the cost of a plant and other surface infrastructure such as roads, housing etc.

The operating costs are based on a cost of R180/tonne shaft head delivery cost for mining operations within 1km of the shaft. As the mining operations extend further out along strike, the operating costs will increase. The most accurate way to reflect this increase is to change the operating costs over the life of mine, according to the mine plan. However, as this is a purely theoretical model an aggregate cost was assigned to each of the declines, which represents an average shaft head delivery cost over the life of mine.

These costs are shown in the table 5-4.

Dist from shaft (km)	R/t Cost
0-1 km	180.00
1-2 km	183.74
2-3 km	188.26

**Table 5-4: Variation in Shaft head delivery cost with distance from the shaft**

#### **5.4.5.2 Results**

The results show that although the NPV of the different decline shafts (Figure 5-11) increases with the strike length, the NPV increases at a slower rate for the 4, 5 and 6km strike length.

This analysis of optimal strike lengths indicates that the additional value obtained by mining, at a constant rate, beyond 4 km from the decline shaft decreases and may need to be compensated for by increasing the rate of production.

The IRR (Figure 5-12) indicates the same point, i.e. that the benefits of mining at a distance of more than four kilometres from a decline shaft at a constant rate decrease and should, perhaps, be combined with increasing the mining rate.

Figure 5-10 shows the different cash flows for the different options.

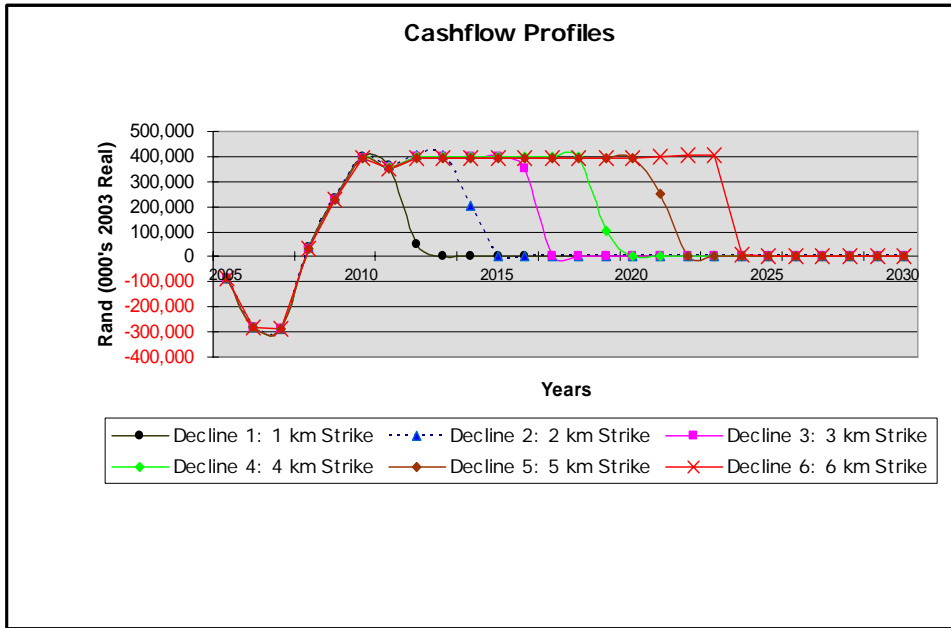


Figure 5-10: Project Cash Flow Profiles

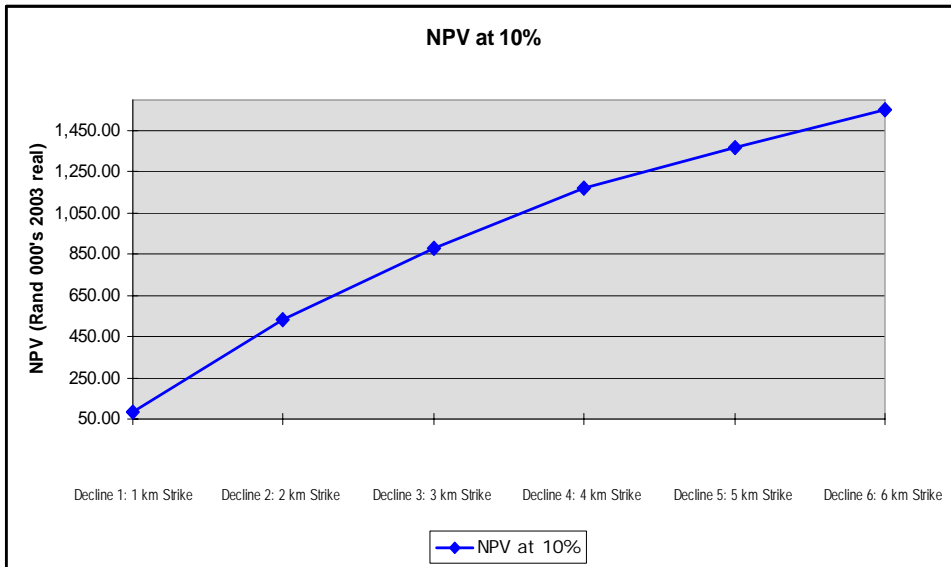
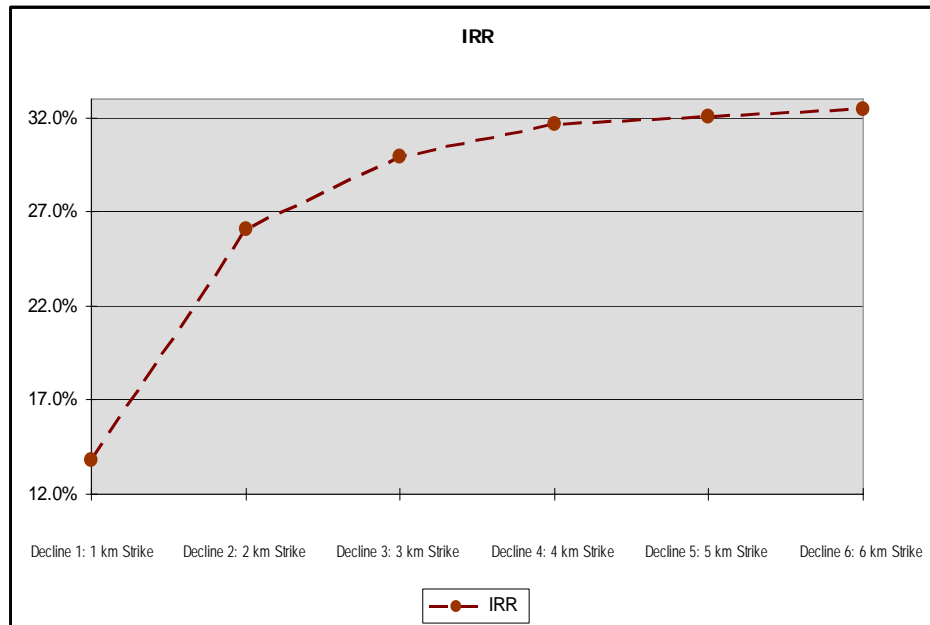


Figure 5-11: NPV at 10% of Decline Shafts



**Figure 5-12: IRR of Decline Shafts**

#### **5.4.6 Conclusion**

The NPV and IRR (Figure 5-12 and Figure 5-9) of the decline shaft are very sensitive to changes in strike length below 3km and less sensitive to those above 4 km strike length.

The NPV increases at a reduced rate above 4 km strike length, as a result of diminishing returns associated with economies of scale. In other words, the bulk of the benefits has been realised at about a 3 km strike distance allocation.

#### **5.5 Conclusion in terms of Mining Economics risks**

The biggest risk associated with any project is that it is designed sub optimally from day one and, hence, the focus is on optimisation. This sub optimality can be avoided relatively easily by conducting a series

of exercises as described in the previous sections. It is also critical that the key drivers are identified and optimised to facilitate maximum feasible returns with an acceptable risk exposure.

There is also potential to improve on existing infrastructure by reviewing the key drivers and determining if there is room to unlock further value by removing bottlenecks. This is often possible and includes opportunities like replacing regular skips with longer lightweight aluminium skips to increase the hoisting capacity.

Specialised Mining Economics input in the pre-feasibility phase of a project is likely to have a major impact on the success of the project.

## **6 CONCLUSION**

The tools as described in the preceding four chapters were used in all the exercises and have been tested in practice. The title of this research report is: 'Risk Management in Mining and Minerals Economics as well as Minerals Resource Management'. The first section of the report focuses on the definitions and risks encountered in the industry. There is also an analysis on what a Mining Economist and a Mineral Resource Manager will encounter in terms of risk.

The second chapter touches on the Basic Mining Equation (BME) and its uses. The chapter looks at using stochastic methods to improve optimisation and identify risk. The Palisade @Risk software was used to analyse five years of historical data and predict the future value of the operation with its associated risk.

The third chapter is based on the use of the cigarette box optimiser (CBO), where the cost volume curve and the orebody signature are used to determine optimality in returns. This chapter also looks at various forms of the BME and how it could be used to identify risk. Chapter 4 covers quantification of risk from a probability perspective, using systems reliability logic.

The fifth chapter focuses on the Macro Grid optimiser, which considers the spatial differentiation of the orebody and determines optimality through an iterative process.



As an overall conclusion, it was found that the biggest risk associated with mining could be to extract the orebody sub-economically. Some ore bodies could yield double the return than was originally intended. In order for that to happen, the extraction programme should undergo some form of optimisation. This will ensure that the optimal volume, cut-off, selectivity and efficiencies are met.

Our prime purpose as miners is to unlock value from the orebodies in our care. Mining is a destructive process and you only have one chance of extracting the orebody. It is obviously best to extract optimal value for the shareholders and other stakeholders in the process.

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