

https://dspace.org/

Techno-economic optimisation of level and raise spacing range in planning a Bushveld Complex platinum reef conventional breast mining layout

Musingwini, Cuthbert

2010-07-15T08:03:38Z
http://hdl.handle.net/10539/8291

Downloaded from a DSpace repository. https://dspace.org/

TECHNO-ECONOMIC OPTIMISATION OF LEVEL AND RAISE SPACING RANGE IN PLANNING A BUSHVELD COMPLEX PLATINUM REEF CONVENTIONAL BREAST MINING LAYOUT

Cuthbert Musingwini

A thesis submitted to the Faculty of Engineering and the Built Environment, University of the Witwatersrand, Johannesburg, in fulfilment of the requirements for the degree of Doctor of Philosophy.

Johannesburg, 2009

DECLARATION

I declare that this thesis is my own, unaided work. Where use has been made of the work of others, it has been duly acknowledged. It is being submitted for the Degree of Doctor of Philosophy in the University of the Witwatersrand, Johannesburg. It has not been submitted before in any form for any degree or examination in any other University.

Signed:

_____

Cuthbert Musingwini

This _____ day of _____ 2009

"It is not the answer that enlightens, but the question"

(Eugène Ionesco, French-Romanian playwright and dramatist, 1909-1994, in *Découvertes, 1969*)

ABSTRACT

The Bushveld Complex in South Africa is a geological formation that hosts approximately 87% of all the known world platinum group metal (PGM) resources and reserves. It produces about 77% of the world's primary platinum production. However, the sheer size of the resources and reserves obscures the fact that the PGM mineral resources are a wasting asset, and should therefore be extracted optimally in order to ensure sustainable production. In addition, a 2006 survey of research and development (R&D) needs of the South African platinum mining companies by the CSIR-Miningtek, noted that out of 19 possible R&D areas, layout optimisation is one of the top four priority R&D focus areas. Section 51 of the Mineral and Petroleum Resources Development Act (MPRDA) of 2002, of South Africa, also emphasises that owners of mining rights should optimally extract mineral resources. About 70% of the platinum production from the Bushveld Complex is extracted using conventional mining methods, while the remainder comes from hybrid and mechanised mining methods. It was therefore prudent to focus on optimising conventional mining layouts.

Optimisation in mining broadly requires extracting the maximum amount of ore by excavating and moving the minimum amount of waste in the shortest possible time and in the safest and most environmentally acceptable manner. In open-pit mine planning, this broadly requires minimising the waste stripping ratio, while in underground mine planning it principally requires minimising the metres of waste development. A literature review revealed that minimising waste development in conventional breast mining is predominantly achieved by increasing level and raise spacing. However, when level and raise spacing are increased, other factors such as productivity are negatively affected, thus requiring a delicate trade-off of contradicting factors. This characterises the problem as a multi-criteria optimisation process that should be solved using multi-criteria decision analysis (MCDA) techniques.

The Analytic Hierarchy Process (AHP) was the most appropriate MCDA methodology for solving the problem of optimising level and raise spacing. By using real geological data on the orebody code-named OB1 that was typical of Bushveld Complex platinum reef deposits, the optimal range of vertical level spacing derived was 30m-50m, while the optimal range of raise spacing was 180m-220m. The layout designs and schedules were done in Mine2-4D® and EPS® software suite, which is one of the mine design and planning software currently used by the South African platinum mining industry for long-term mine planning. The research methodology used in this thesis and the results obtained were received positively by the South African platinum mining industry because for the first time in several decades, a holistic methodology and practically acceptable solution had been obtained for the controversial debate of optimising level and raise spacing for conventional mining layouts.

HIGHLIGHTS FROM RESEARCH

Some of the highlights of this research study include the following notable achievements:

- Four papers have been published out of this research study and these are listed in the next section and the associated paper abstracts are included in Appendix 10.1.
- A request was made to from Canada, through the South African Institute of Mining and Metallurgy (SAIMM), to have part of the paper Musingwini, Minnitt and Woodhall (2006), which was published out of this research, to be included in a forthcoming book on Real Options. A copy of the e-mail communication in this regard is included in Appendix 10.1.
- Some personnel in AngloGold Ashanti's Great Noligwa Mine expressed intention to adapt the concept of a Flexibility Index (FI) developed in this research study in order to link operating flexibility with safety because there is a general sentiment on the mine that some of the accidents occur due to inadequate operating flexibility making it difficult to move workers to safer panels. A copy of the e-mail communication in this regard is included in Appendix 10.1.
- After the final research findings were initially presented to the two largest platinum mining companies in South Africa, Impala Platinum and Anglo Platinum, on the 10th and 13th July 2009 respectively, an invitation was subsequently made to have the findings to be presented to the Association of Mine Managers of South Africa (AMMSA) meeting on the 06th August 2009. A copy of the presentation will soon be made available on the AMMSA website: http://www.ammsa.org.za. Feedback comments from industry are included in Appendix 10.3 and the AMMSA programme for the 06th August 2009 is included in Appendix 10.4.
- Subsequent to the presentation being made, AMMSA requested to have the presentation to be compiled as a technical paper for inclusion into their annual publication, *Papers and Discussions*. Work is currently under way to compile the paper.

LIST OF PUBLICATIONS

The publications listed below have emanated from this research work so far:

1. **Musingwini, C**, Minnitt, R C A and Woodhall, M. (2006), Technical operating flexibility in the analysis of mine layouts and schedules, in *Proceedings of the Second International Platinum Conference: 'Platinum Surges Ahead'*, Sun City, South Africa, 8th-12th October 2006, The Southern African Institute of Mining and Metallurgy, pp159-164. An electronic copy of the paper is available on INTERNET: http://www.platinum.org.za/Pt2006/Papers/159-164_Musingwini.pdf.
2. **Musingwini, C**, Minnitt, R C A and Woodhall, M. (2007), Technical operating flexibility in the analysis of mine layouts and schedules, in *Journal of The Southern African Institute of Mining and Metallurgy*, Vol. 107, No. 2, pp129-136. An electronic copy of the paper is available on subscription from the SAIMM website at INTERNET: http://www.saimm.co.za/Publications/downloads/v107n02p129.pdf.
3. **Musingwini, C** and Minnitt, R C A. (2008), Ranking the efficiency of selected platinum mining methods using the analytic hierarchy process (AHP), in *Proceedings of the Third International Platinum Conference: 'Platinum in Transformation'*, Sun City, South Africa, 6th - 9th October 2008, The Southern African Institute of Mining and Metallurgy, pp319-326.
4. **Musingwini, C**, Montaz, A and Dikgale, T. (2009), A linear programming and stochastic analysis of mining replacement rate for typical Bushveld Complex platinum reef conventional mining under variable geological losses, a paper accepted to the *Eighteenth International Symposium on Mine Planning & Equipment Selection (MPES 2009) Eleventh International Symposium on Environmental Issues and Waste Management in Energy and Mineral Production (SWEMP 2009)*, 16th -19th November 2009, Banff, Alberta, Canada.

ACKNOWLEDGEMENTS

This thesis is a culmination of many ideas, guidance and encouragement from several people and organisations that I am grateful to but cannot possibly acknowledge all of them. I would like to particularly acknowledge the following individuals and companies for their specific contributions:

- My supervisor, Professor R.C.A Minnitt (JCI Chair of Mineral Resources and Reserves, University of Witwatersrand), for his invaluable academic guidance throughout the research and painstakingly reading through draft chapters.
- My mentor, Mr. M. H. Rogers (Head, Mine Technical Services, Anglo Platinum), Mr. A. Vermeulen (ex-Mine Planning Engineer, Anglo Platinum), Mr. D. Gray (ex-Exploration Manager, Anglo Platinum) for providing geological models, insights into the half-level optimisation model and constructive criticism of the entire research.
- Prof. H. Phillips, (Head, School of Mining Engineering, University of Witwatersrand), for providing School support in funding the study programme, mine visits and negotiating renewal of Mine2-4D® and EPS® software licences.
- Prof. F. Cawood, (Chair of Mine Surveying, School of Mining Engineering, University of Witwatersrand), for the advice and kind words of encouragement during difficult times when I felt like giving up.
- Dr. H. Mtegha, (Senior Lecturer, School of Mining Engineering, University of Witwatersrand), for help with proof-reading some of the drafts.
- Mr. M. Woodhall (Business Manager, Mining Operations Technologies, GijimaAST), Mr B. Macbeth and Mr M. O'Callaghan both of GijimaAST, for providing the Mine2-4D® and EPS® software suite, and associated educational licensing and training.
- Mr. L. Zindi (Project Director: Greefields, Impala Platinum) and Mr L. Mohloki (ex-Projects Manager, Impala Platinum) for providing assistance with design inputs and reviewing the Mine2-4D® and EPS® designs and schedules.
- Mr A. Clay (Managing Director, Venmyn) and Mr G. Njowa (Mineral Project Analyst, Venmyn) for assistance with setting up and reviewing the financial models.
- My wife Grace, son Shingirai and daughter Gamuchirai for their love, patience and understanding during times when the research took me away from them, even the times when I was physically present but not really there for them.

Although the opportunity and permission to use some of the material contained in this thesis is gratefully acknowledged, the opinions expressed are those of the author and may not necessarily represent the policies of the companies mentioned. Of course, any errors and ambiguities remaining in this thesis are entirely my own responsibility.

DEDICATION

To my late mother (Joyce), brothers (John and Osward) and sisters (Alice and Phillipa) with whom I would have liked to share exciting moments such as this one.
May the Lord bless us all!

CONTENTS **Page**

DECLARATION ... ii

ABSTRACT ... iv

HIGHLIGHTS FROM RESEARCH ... v

LIST OF PUBLICATIONS ... vi

ACKNOWLEDGEMENTS ... vii

DEDICATION ... viii

LIST OF FIGURES ... xiii

LIST OF TABLES ... xviii

NOMENCLATURE AND TERMINOLOGY ... xx

1 INTRODUCTION ... 1

1.1 CHAPTER OVERVIEW ... 1
1.2 BUSHVELD COMPLEX: LOCATION AND HISTORY OF PLATINUM DISCOVERY AND MINING ... 1
1.3 STRATEGIC IMPORTANCE OF PLATINUM MINING ON THE BUSHVELD COMPLEX ... 5
1.4 OBLIGATION OR RESPONSIBILITY FOR OPTIMAL EXTRACTION ... 8
1.5 A GENERAL MINING GEOLOGY OF THE BUSHVELD COMPLEX ... 9
1.5.1 Potholes ... 12
1.5.2 Rolling reef ... 13
1.5.3 Dykes and faults ... 13
1.6 A BRIEF HISTORY OF MINING METHODS ON THE BUSHVELD COMPLEX ... 13
1.7 CONVENTIONAL MINING ... 17
1.7.1 Up-dip and down-dip mining ... 19
1.7.2 Crosscut access breast mining ... 21
1.7.3 Laybye access conventional breast mining ... 24
1.8 LEVEL AND RAISE SPACING ON CONVENTIONAL MINING LAYOUTS ... 27
1.8.1 Brief history on level spacing on inclined tabular reefs ... 27
1.9 CURRENT LEVEL AND RAISE SPACING PRACTICES NARROW REEF TABULAR MINES ... 28
1.10 OBJECTIVES IN PLANNING LEVEL AND RAISE SPACING ... 30
1.11 EFFECT OF INCREASING LEVEL AND RAISE SPACING ... 31
1.12 RESEARCH QUESTION AND RELEVANCE ... 34
1.13 STRUCTURE OF THESIS ... 35

2 REVIEW OF LEVEL AND RAISE SPACING OPTIMISATION AND MULTI-CRITERIA DECISION ANALYSIS (MCDA) TECHNIQUES ... 37

2.1 INTRODUCTION ... 37
2.2 REVIEW OF PREVIOUS AND CURRENT WORK ... 37
2.2.1 Level spacing optimisation by Eaton (1934) ... 37
2.2.2 Optimisation of level spacing by Lewis (1941) ... 39
2.2.3 Scraper winch productivity and raise spacing by Brassell (1964) ... 41
2.2.4 Optimisation of level and raise spacing by Zambó (1968) ... 43
2.2.5 Optimisation of raise spacing by Lawrence (1984) ... 48
2.2.6 Anglo Platinum MTS (2005) Half-Level Optimisation Model ... 52
2.2.7 Lonmin's Six-Sigma Ideal Ore Reserve Replacement Rate Project ... 54

2.3 EXPLANATION OF POWER RELATIONSHIP BETWEEN DEVELOPMENT COSTS AND SPACING 55
2.4 OTHER RELATED STUDIES ON PLANNING THE LOCATION AND TIMING OF DEVELOPMENT 56
2.5 OVERVIEW OF MULTI-CRITERIA DECISION ANALYSIS (MCDA) METHODOLOGY 58
2.5.1 Structure of the MCDA decision problem..... 59
2.5.2 Decisions within MCDA framework..... 60
2.6 THE GENERIC MCDA PROCESS..... 62
2.6.1 Mapping objectives into criteria 62
2.6.2 Assigning weights to criteria 63
2.6.3 Aggregate the weights and O_{ij} values..... 64
2.6.4 Execute decision 65
2.7 MAIN CATEGORIES OF MCDA METHODOLOGIES 65
2.7.1 ELECTRE methodology 65
2.7.2 PROMETHEE methodology 66
2.7.3 MAUT methodology..... 67
2.7.4 AHP methodology 67
2.8 STRUCTURE COMPARISON OF THE FOUR CATEGORIES OF MCDA METHODOLOGIES 71
2.9 CHOICE OF AHP METHODOLOGY..... 71
2.10 SUMMARY 73

3 TECHNICAL OPERATING FLEXIBILITY IN MINE PLANNING 75

3.1 INTRODUCTION 75
3.2 DEFINING OPERATING FLEXIBILITY 75
3.3 CREATING TECHNICAL OPERATING FLEXIBILITY IN NARROW TABULAR REEFS 77
3.4 IMPORTANCE OF OPERATING FLEXIBILITY IN MINE DESIGN AND PLANNING 79
3.5 A METRIC FOR MEASURING TECHNICAL OPERATING FLEXIBILITY..... 81
3.6 BEHAVIOUR OF FLEXIBILITY INDEX UNDER VARIABLE TIMING OF DEVELOPMENT 82
3.7 CONCLUDING SUMMARY..... 84

4 GEOLOGICAL OREBODY MODEL (OB1) DESCRIPTION 85

4.1 INTRODUCTION 85
4.2 OVERVIEW OF EXPLORATION WORK DONE TO CONFIGURE AND DELINEATE OB1 85
4.3 OB1 STRUCTURAL GEOLOGY 86
4.4 OB1 FLAT TOPOGRAPHY ASSUMPTION 87
4.5 PRILL SPLIT 88
4.6 OB1 GEOLOGICAL BLOCK MODEL..... 89
4.7 OB1 GRADE-TONNAGE CURVE 95
4.8 SUMMARY OF OB1 WIREFRAME PROPERTIES..... 95
4.9 OB1 MEASURED RESOURCE (EXCLUSIVE OF GEOLOGICAL LOSSES AND OXIDISED ZONE) 97
4.10 SUMMARY 97

5 DESIGN AND SCHEDULING TECHNO-ECONOMIC ASSUMPTIONS 98

5.1 INTRODUCTION 98
5.2 CHOICE OF MINE2-4D® AND EPS® SOFTWARE AND OVERALL DESIGN PROCESS 98
5.2.1 Choice of Mine2-4D® and EPS® software 98
5.2.2 Overview of design and scheduling process and files 99
5.3 PRODUCTION RATE AND RAMP-UP PROFILES..... 102
5.4 ACCESS DEVELOPMENT 103
5.4.1 Capital development..... 104
5.4.2 Off-reef development..... 110
5.4.3 On-reef secondary development 112
5.5 STOPING LAYOUT 113
5.5.1 Regional and boundary pillars 113
5.5.2 In-stope (in-panel) pillars..... 115

5.5.3 Optimal stope shape (Lagrange multiplier method) ... 115
5.5.4 Panel length ... 118
5.5.5 Take-over scraper winch concept ... 118
5.6 BASIC GRADE EQUATION (BGE) ... 118
5.6.1 On-reef development dilution ... 119
5.6.2 Optimal resource mining cut ... 119
5.6.3 Over-break and under-break ... 120
5.6.4 Area extraction percentage per panel ... 121
5.6.5 Waste handling ... 121
5.7 BASIC MINING EQUATION (BME) ... 121
5.8 VENTILATION PLANNING ... 123
5.9 MANPOWER PLANNING ... 123
5.10 TRAMMING EQUIPMENT PLANNING ... 127
5.11 METALLURGICAL ASSUMPTIONS ... 127
5.12 FINANCIAL MODELLING ASSUMPTIONS ... 128
5.12.1 Forecasts of commodity prices, inflation and exchange rates ... 129
5.12.2 Royalty rate ... 129
5.12.3 Discount rate ... 130
5.12.4 Corporate Tax Rate ... 132
5.12.5 Capital costs ... 132
5.12.6 Operating costs ... 133
5.12.7 Environmental rehabilitation costs ... 134
5.13 SUMMARY ... 135

6 TECHNO-ECONOMIC RESULTS AND VALIDATION CHECKS ... 136

6.1 INTRODUCTION ... 136
6.2 REASONABLENESS CHECKS ... 136
6.2.1 Centares discrepancy ... 137
6.2.2 Tonnage discrepancy ... 137
6.2.3 Valuation check ... 138
6.3 GENERAL RESULTS ... 140
6.3.1 Backlengths at fixed vertical level spacing ... 140
6.3.2 Total development metres in relation to level and raise spacing ... 141
6.3.3 Number of levels ... 142
6.3.4 Number of stopes ... 143
6.3.5 Redundant criteria ... 145
6.4 TRENDS IN OPTIMISATION CRITERIA ... 146
6.4.1 PV of development costs per centare ... 147
6.4.2 Project NPV ... 148
6.4.3 Payback period ... 149
6.4.4 Replacement Factor (RF) ... 150
6.4.5 Shaft head grade ... 151
6.4.6 Productivity ... 152
6.4.7 Production rate ... 154
6.4.8 Flexibility index (FI) ... 155
6.4.9 Life of raiseline ... 156
6.4.10 Life of mine (LOM) ... 157
6.4.11 Build-up period ... 158
6.5 INTERPOLATION AND CROSS-INTERPOLATION OF OPTIMISATION CRITERIA RESULTS ... 159
6.6 SUMMARY ... 160

7 OPTIMISATION BY ANALYTIC HIERARCHY PROCESS (AHP) ... 161

7.1 INTRODUCTION ... 161
7.2 SURVEY OF RELATIVE WEIGHTING OF DECISION CRITERIA ... 161
7.3 MS EXCEL PROCEDURE FOR THE AHP ... 161
7.3.1 Enter pair-wise weighting of criteria from each respondent ... 162

7.3.2 Aggregate the pair-wise responses 162
7.3.3 Normalise the pair-wise comparisons 162
7.3.4 Calculate the average aggregate weight of each criterion 162
7.3.5 Estimate the consistency of judgements 162
7.3.6 Apply weights to layout scores to obtain AHP priority scores 163
7.4 OPTIMAL RANGE OF LEVEL AND RAISE SPACING 163
7.5 SENSITIVITY ANALYSIS 164
7.6 SUMMARY 167
PREVIOUS STUDIES USED THE AHP AS A SELECTION TOOL. THIS STUDY USED THE AHP AS AID TO OPTIMISATION UNDER MULTI-CRITERIA CONDITIONS 167

8 OBSERVATIONS, CONCLUSIONS AND RECOMMENDATIONS 168

8.1 INTRODUCTION 168
8.2 OBSERVATIONS 168
8.3 RESEARCH CONTRIBUTIONS 169
8.4 RESEARCH LIMITATIONS 170
8.5 RECOMMENDATIONS FOR FUTURE RESEARCH WORK 171

9 REFERENCES 173

10 APPENDICES 10-1

10.1 SUMMARY OF PAPER ABSTRACTS AND E-MAIL COMMUNICATIONS ON THE RESEARCH 10-1
10.2 COPIES OF AHP SURVEY LETTER, QUESTIONNAIRE AND RESPONSES 10-7
10.3 INDUSTRY FEEDBACK COMMENTS 10-18
10.4 AMMSA PROGRAMME FOR THE 06TH AUGUST 2009 10-22

LIST OF FIGURES

Figure **Page**

Figure 1.1: Platinum operations and projects on the Bushveld Complex 2

Figure 1.2: Relative contributions to total South African platinum production by mining companies in 2005..... 4

Figure 1.3: Relative contributions to total South African platinum production by mining companies in 2006..... 4

Figure 1.4: Relative proportion of global platinum resources and reserves by major countries in 2005..... 5

Figure 1.5: Relative proportions of global primary platinum production by major countries in 2005 6

Figure 1.6: South African gold and platinum production..... 6

Figure 1.7: Gold and platinum labour as proportion of total formal South African mining labour 7

Figure 1.8: Relative contributions to South Africa's GDP by gold and platinum sectors..... 7

Figure 1.9: Typical section across a Bushveld Complex mine showing the Merensky overlying the UG2 reef..... 10

Figure 1.10: General UG2 stratigraphy showing optimal location of footwall development .. 11

Figure 1.11: Typical UG2 stratigraphy showing the actual rock type appearance..... 11

Figure 1.12: Section across a typical UG2 pothole 13

Figure 1.13: Topology of South African platinum underground mining methods 14

Figure 1.14: Distribution of PGM production output by mining method in 2005..... 15

Figure 1.15: Forecast 2010 distribution of PGM production output by mining method 15

Figure 1.16: A schematic illustration of crosscut access conventional mining in plan view .. 18

Figure 1.17: Typical scraper winch arrangement in a conventional mining stope panel layout 19

Figure 1.18: Schematic layout of a typical half-level for conventional up-dip mining 20

Figure 1.19: Schematic illustration of position of SPD relative to production panel 20

Figure 1.20: A typical crosscut access conventional mining layout 21

Figure 1.21: A 3-D perspective of a crosscut access layout at a raise connection in Mine 2-4D® 22

Figure 1.22: Longitudinal plan of a typical conventional breast layout 24

Figure 1.23: Plan view of a laybye access layout at a raise connection 25

Figure 1.24: Side view across a laybye access raise connection 25

Figure 1.25: A 3-D perspective of a laybye access raise connection in Mine2-4D® 26

Figure 2.1: Variation of cost per tonne ore mined with increasing level interval or spacing .. 40

Figure 2.2: Variation of scraper winch productivity with increasing strike gulley length on a panel 42

Figure 2.3: Typical plot of K_A and q values for mines operating under similar conditions 44

Figure 2.4: Variation of the specific cost function, K, with level interval, h 46

Figure 2.5: Variation of specific cost function, K, with crosscut panel length, S 47

Figure 2.6: Convex quadratic relationship between panel productivity with raiseline spacing 49

Figure 2.7: Extrapolation of the Lawrence (1984) model at longer raiseline spacing 50

Figure 2.8: Variation of PV of cost savings with increasing raise spacing 50

Figure 2.9: Typical Half-Level Model Output 53

Figure 2.10: Half-Level Model output for different shafts 53

Figure 2.11: Analogy of line and strike distance being repeatedly into equal parts 56

Figure 2.12: Range of MCDA decisions 61

Figure 2.13: Relationship between Sorting and Choice decisions for a large set of alternatives 61

Figure 3.1: Sequence of typical mining phases in the build-up to technical operating flexibility 78

Figure 3.2: OB1 production profiles at variable ore availability 82

Figure 3.3: Variation of Flexibility Index with ore availability 83

Figure 3.4: Relationship between NPV and flexibility Index 84

Figure 4.1: A Mine 2-4D® representation of OB1 structural geology 86

Figure 4.2: A Mine 2-4D® illustration of the oxidised zone (in green) extending for about 30m below surface 87

Figure 4.3: A Mine2-4D® illustration of OB1 (in blue) overlain by a flat topography (in brown) 87

Figure 4.4: Typical average 4E prill splits for UG2 and Merensky reefs 88

Figure 4.5: Typical average 4E prill splits for UG2 and Merensky reefs 89

Figure 4.6: A Mine 2-4D® illustration of the OB1 geological block model in 2-D 92

Figure 4.7: A Mine 2-4D® illustration of overlay of OB1 block and wireframe models in 2-D 92

Figure 4.8: A Mine 2-4D® illustration of overlay of OB1 block and wireframe models in 3-D 93

Figure 4.9: A Mine 2-4D® illustration of OB1 block model stretched to intersect the mine design on the wireframe model 94

Figure 4.10: Mine 2-4D® project setup showing *workmod* as the default geological block model to be used interrogated when evaluating designs 94

Figure 4.11: Grade-tonnage curve for OB1 95

Figure 4.12: Summary of OB1 wireframe properties as evaluated in Mine 2-4D® 96

Figure 4.13: Default values assumed for OB1 97

Figure 5.1: The engineering circle or wheel of design 100

Figure 5.2: Relative positions of the three-cluster decline system for OB1 in Mine2-4D®... 105

Figure 5.3: Photos taken underground illustrating a chairlift decline 106

Figure 5.4: Location of Merensky reef decline cluster at Boschfontein shaft illustrated in Mine2-4D® 107

Figure 5.5: Location of OB1 decline cluster relative to centre of gravity in Mine2-4D® 108

Figure 5.6: Mine2-4D® illustration of water reticulation infrastructure at bottom of OB1 108

Figure 5.7: Mine2-4D illustration of intake and exhaust shaft splits on declines near surface 109

Figure 5.8: Mine2-4D® illustration of design definitions of fixed cross-sectionals assumed for OB1 110

Figure 5.9: 3-D illustration of off-reef development breaking away from declines 111

Figure 5.10: A Mine2-4D® illustration of the 67_400 layout showing stoping areas (in white), regular pillars and geological structures left as irregular pillars (in blue). 114

Figure 5.11: Stope blocked out by two consecutive raiselines and two consecutive levels 116

Figure 5.12: Cross-sectional area of a raise or winze showing the ore and waste portions 119

Figure 5.13: Typical PGE value distribution over reef thickness in UG2 reef 120

Figure 5.14: Mine2-4D® illustration of design definitions of outlines assumed for OB1 122

Figure 5.15: An EPS® illustration of a maximum of two teams for secondary on-reef development 125

Figure 5.16: An EPS® illustration of a secondary on-reef development team using the *SecondaryOnReefDev* look-up table..... 126

Figure 5.17: An EPS® illustration of a production look-up table showing variable mining rates for different development ends 126

Figure 5.18: Individual contribution to project risk by 19 project development risk components 131

Figure 5.19: Average annual wages published by the Chamber of Mines..... 134

Figure 6.1: Relative positions of the 15 layouts on the platinum valuation curve (Courtesy of Venmyn) 139

Figure 6.2: Variation of total development metres with raise spacing 142

Figure 6.3: Variation of total development metres with level spacing..... 142

Figure 6.4: Variation of number of levels with increasing vertical level spacing 143

Figure 6.5: A Mine2-4D® illustration of OB1 stopes divided into blocks by dykes..... 144

Figure 6.6: Variation of number of stopes with increasing raise spacing..... 144

Figure 6.7: Variation of PV of development cost per centare mined in relation to level spacing 147

Figure 6.8: Variation of PV of development cost per centare mined in relation to raise spacing 147

Figure 6.9: Variation of project NPV with increasing level spacing..... 148

Figure 6.10: Variation of project NPV with increasing raise spacing 149

Figure 6.11: Variation of payback period with level spacing..... 150

Figure 6.12: Variation of payback period with increasing raise spacing 150

Figure 6.13: Variation of RF with level spacing 151

Figure 6.14: Variation of RF with increasing raise spacing 151

Figure 6.15: Variation of shaft head grade with increasing raise spacing..... 152

Figure 6.16: Variation of productivity with increasing level spacing..... 153

Figure 6.17: Variation of productivity with increasing raise spacing 153

Figure 6.18: Variation of production rate with increasing level spacing 154

Figure 6.19: Variation of production rate with increasing raise spacing..... 154

Figure 6.20: Variation of flexibility index with level spacing..... 155

Figure 6.21: Variation of flexibility index with raise spacing 156

Figure 6.22: Variation of life of raiseline with increasing level spacing..... 156

Figure 6.23: Variation of life of raiseline with increasing raise spacing 157

Figure 6.24: Variation of LOM with increasing level spacing 157

Figure 6.25: Variation of LOM with increasing raise spacing 158

Figure 6.26: Variation of build-up period with level spacing 158

Figure 6.27: Variation of build-up period with increasing raise spacing 159

Figure 6.28: Cross-interpolation of PV values to estiamte PV values for layouts that were not designed 160

Figure 7.1: A 3-D Excel surface contour plot of the AHP priority score for the base case .. 163

Figure 7.2: A 3-D Excel surface plot of the AHP priority score when criteria carry equal weighting 165

Figure 7.3: A 3-D Excel surface plot of the AHP priority score when importance of NPV increases by 10% 166

Figure 7.4: A 3-D Excel surface plot of the AHP priority score when importance of NPV decreases by 10% 166

LIST OF TABLES

Table **Page**

Table 1.1: Level and raise spacing ranges for Witwatersrand gold mines using scattered or longwall mining 29

Table 2.1: The structure of a generic MCDA problem 59

Table 2.2: Random Index (RI) for n-ordered matrix 70

Table 2.3: Relationship between number of criteria and comparisons 71

Table 2.4: Comparison of the four MCDA methodology categories 71

Table 2.5: Examples of minerals industry problems solved using MCDA techniques 73

Table 4.1: UG2 prill split assumed for OB1 89

Table 4.2: Meaning of codes for OB1 key variables 90

Table 4.3: Summary statistics for OB1 91

Table 4.4: Summary of the geological characteristics of OB1 97

Table 5.1: Geological data files obtained from industry 101

Table 5.2: Application of Taylor's Law to estimate production rate for OB1 103

Table 5.3: Mining rates of development ends by category 112

Table 5.4: Labour complement setting for OB1 124

Table 5.5: Beneficiation Assumptions 128

Table 5.6: 4E Commodity price forecasts in US$/oz using 2009 as the base year 129

Table 5.7: Project discount rate estimation for OB1 using CAPM 131

Table 5.8: R153 and R157 bond rates as of 13th June 2009 131

Table 5.9: Summary calculation of ventilation capital and unit operating costs 133

Table 5.10: Calculation of environmental rehabilitation costs for OB1 135

Table 5.11: Weighting factors applied to 'flat rate' 135

Table 6.1: Summary centares discrepancy for the 15 layouts 137

Table 6.2: Summary tonnage discrepancy for the 15 layouts 138

Table 6.3: Summary of ranges of backlengths obtained for the 15 layouts 141

Table 6.4: Deviation of actual backlengths from calculated backlengths at constant reef dip of 9.6º 141

Table 6.5: Area extraction percent for all 15 layouts 145

Table 6.6: Estimates of build-up period in years for OB1 layouts 146

Table 6.7: Estimates of production tail-off period in years for OB1 layouts 146

Table 6.8: Shaft head grade constant with respect to increasing level spacing for all 15 layouts 152

Table 6.9: PV of development cost per centare mined (ZAR mil) 159

NOMENCLATURE AND TERMINOLOGY

Term or Symbol	**Explanation**
Backlength (stope back)	The length of a raise between two consecutive levels, measured along reef dip, following the reef horizon.
Centare (ca)	One square metre (1 m^2) of stoped-out reef area measured on the reef plane.
Dip	The angle at which a stratum or other planar feature is inclined below the horizontal datum.
Drift	Alternative term for a drive or haulage or tunnel.
Footwall	Mass of rock beneath a geologically identifiable discontinuity surface. In tabular mining, it is the rock mass below the reef plane.
Hangingwall	Mass of rock above a geologically identifiable discontinuity surface. In tabular mining it is the rock mass above the reef plane.
Muckpile	The fragmented rock mass obtained after a blast.
Reef / orebody / vein	A mineral deposit, other than a surface alluvial mineral deposit, that contains economically exploitable minerals.
Replacement Factor (RF) or Replacement Ratio (RR)	The ratio of stoping centares to development metres which measures how many stoping centares are made available by each metre of development mined.
Rock mass	Rock occurring in its *in-situ* state, including all the discontinuities in it.
Abbreviation, Symbol or Unit	
λ	Lagrange multiplier.
AHP	Analytic Hierarchy Process.
ASG	Advanced strike gulley.
ktpm	kilo tonnes per month.
masl	metres above sea level.
/mo	per month.
psi	an imperial unit to measure pressure or stress in pound per square inch.
SPD	Stope preparation drive.
tpa	tonnes per year.
tpd	tonnes per day.
tpm	tonnes per month.
w/h	width to height ratio of support pillars.
Conversions	hp x 0.7457 = kW; 1.341 x kW =hp. These conversions make the following scraper winch sizes equivalent: 75kW = 100hp; 56kW = 75hp; 37kW = 50hp; 22kW = 30hp.

1 INTRODUCTION

1.1 Chapter overview

This chapter presents an overview of the relevant historical, techno-economic and strategic mine planning contexts which justify the decision to particularly want to optimise level and raise spacing in conventional platinum mining layouts of the Bushveld Complex. Firstly, the chapter describes a brief history of the discovery and mining of platinum on the Bushveld Complex. It then addresses the economic and strategic importance of the Bushveld Complex to South Africa, and the associated responsibility for optimal extraction of the platinum resources. The discussion leads to a description of the Bushveld Complex mining geology and its relation to the mining methods used for ore extraction. It emerges from the discussion that conventional mining is the most prevalent mining method on the Bushveld Complex, and its mining layout on a macro-scale is dictated by level and raise spacing. Practices on level and raise spacing selection are subsequently discussed together with the objectives surrounding the selection process. The objectives indicate that optimising level and raise spacing is a multi-criteria decision analysis (MCDA) problem which should be solved by MCDA techniques, something that has not been done before. This then leads into the research question and its relevance. The research question is, “Is there an optimal range of level and raise spacing for a given Bushveld Complex platinum reef mine using conventional breast mining, considering that current operations using conventional breast mining are planned on different combinations of level and raise spacing?”. Lastly, a preview of the structure of the thesis is given at the end of the chapter to show how the various components of the research are related.

1.2 Bushveld Complex: location and history of platinum discovery and mining

The Bushveld Complex, previously known as the Bushveld Igneous Complex, is currently the only known source of economically mineable platinum group metal (PGM) or platinum group element (PGE) resources in South Africa. The terms PGM and PGE are used synonymously throughout this thesis. The Bushveld Complex is a geological formation located in the north-eastern part of the country covering an area spanning three provinces namely; the Mpumalanga, Limpopo and North-West provinces (Figure 1.1). The Bushveld Complex comprises two main limbs, the eastern and western limbs, and a much smaller northern limb (Figure 1.1). The northern limb hosts the Mogalakwena Section of Rustenburg Platinum Mines Limited (RPM). RPM is a subsidiary of Anglo Platinum Limited (Anglo Platinum). The Mogalakwena Section was previously known as the Potgietersrust Platinums Limited (PPL or

PPRust) and existed as a stand-alone mining business unit within Anglo Platinum before it was transferred internally within Anglo Platinum, to become part of the RPM portfolio.

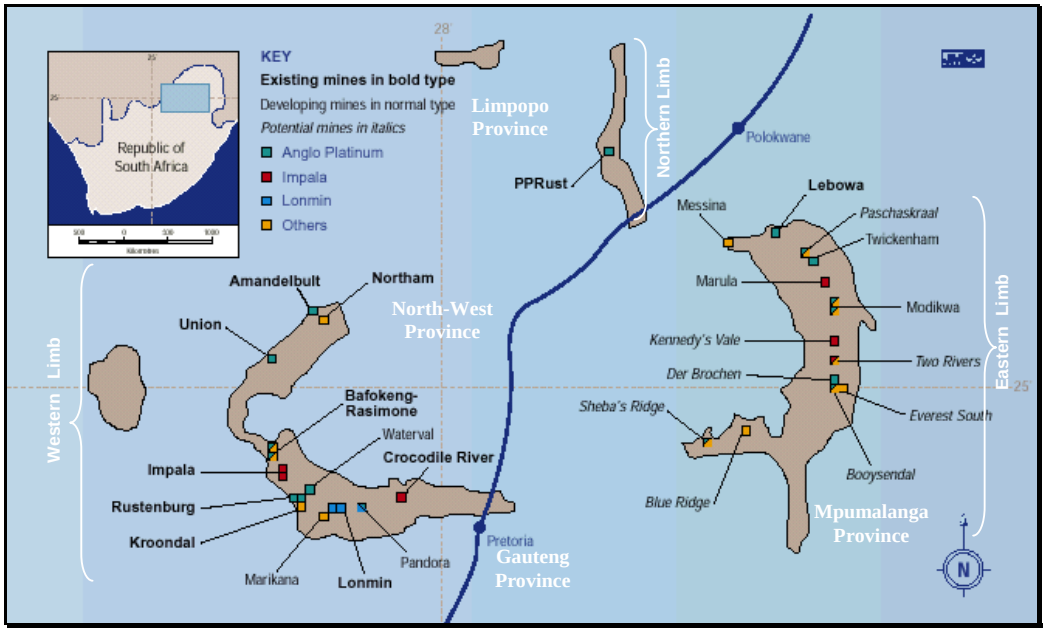


Figure 1.1: Platinum operations and projects on the Bushveld Complex
(Adapted from Department of Minerals and Energy, 2006a)

Commercial mining of platinum on the Bushveld Complex can be traced back to the 1920s. However, there appears to be two different dates relating to the discovery of platinum on the Bushveld Complex. Cawthorn (2006) and Cawthorn (2007) reported that the earliest authenticated scientific report on the discovery of platinum in rocks from the Bushveld Complex was made by William Bettel on 10th November, 1906. An alternative discovery date, which is much more documented, is based on four consecutive platinum reef discoveries in the early 1920s. The first of the four platinum discoveries was in 1923 near Naboomspruit, just north of Warmbaths in the then Transvaal province (Beath, Cousins and Westwood, 1961; Collender, 1987; Ralph Morris and Associates, 1994; Ackerman and Jameson, 2001). This discovery gave impetus to the search for more platinum in that province. In 1924, prospectors working under the direction of Dr. Hans Merensky, a notable geologist of that time, made a second discovery of platinum reef on the eastern limb of the Bushveld Complex in the then Lydenburg district. A year later in 1925, Dr. Merensky's prospectors made two more discoveries in the Potgietersrus area on the northern limb, and in the Rustenburg area on the western limb.

A natural sequel to the platinum discoveries was the formation of a number of mining companies to exploit the platinum reefs. Bye (2003) noted that by the end of 1925 over 50 platinum mining companies had been incorporated in South Africa. Mining of the platinum

reef began in 1924 but, the mining activities were short-lived as they were stopped in 1926 (Beath, Cousins and Westwood, 1961). The early closure of the mine was caused by among other things, extraction difficulties, erratic distribution and small size of the rich PGM mineralised lenses. Emphasis therefore shifted from the Lydenburg and Potgietersrus districts to the Rustenburg area where the PGM values were comparatively good and the geology was much simpler, leading to a boom in platinum mining operations (Ralph Morris and Associates, 1994).

The boom in platinum operations was also short-lived as the industry suffered a market slump in platinum demand compounded by the global economic recession of the early 1930s, which is also known in history as the Great Depression. Only two mining companies, the Potgietersrust Platinums Limited and Waterval Rustenburg Platinum Mining Company, survived the Great Depression but, were later merged in 1932 to form the Rustenburg Platinum Mines Limited, alternatively known as RPM (Beath, Cousins and Westwood, 1961; Ralph Morris and Associates, 1994). Since that time, new platinum mining companies have been formed, new acquisitions made, unbundling and mergers executed, to result in the current ownership structure of platinum mining interests on the Bushveld Complex.

Based on the Research Channel Africa (2009) report, there are at least 16 companies with PGM interests, and over 30 mining operations and projects on the Bushveld Complex. Three major platinum mining companies currently control much of the platinum mining on the Bushveld Complex and dominate South Africa's PGM production. These companies are in order of size, Anglo Platinum, Impala Platinum and Lonmin. Figure 1.2 and Figure 1.3 indicate the relative annual contributions of these major producers to total South African platinum production in 2005 and 2006, respectively. Although the relative contributions vary from year, Anglo Platinum, Impala Platinum and Lonmin still remain as the dominant platinum producers in South Africa. In addition some mining projects have been or are currently being undertaken as joint ventures between the large producers and emerging junior mining companies which are mainly Black Economic Empowerment (BEE) companies. The current control structure is therefore likely to change with time as more junior platinum mining companies, particularly BEE companies, increase their participation in the platinum mining industry or existing companies participate in possible mergers and takeovers. For example, in 2008 and 2009 the media reported talks of possible consolidation and takeovers within the South African platinum mining industry such as the Impala Platinum-Mvelaphanda Resources consolidation, Xstrata-Lonmin takeover, BHP Billiton-Impala Platinum takeover, Aquarius considering options for participation in a possible consolidation with an undisclosed company and Mvelaphanda Resources winding up business to comply with the Johannesburg Stock Exchange (JSE) regulations that prohibit pyramid holding structures for listed companies. The bullish platinum prices experienced in 2008 with a record peak price

of US$2,276/oz in March 2008 (Research Channel Africa, 2009), partly contributed to the decisions surrounding possible mergers, takeovers or consolidations. However, these possible transactions seem to have waned off under the effects of the current global financial crisis that has been experienced since 2008 and forced platinum prices to plummet down to trade around a monthly average of US$851/oz in December 2008.

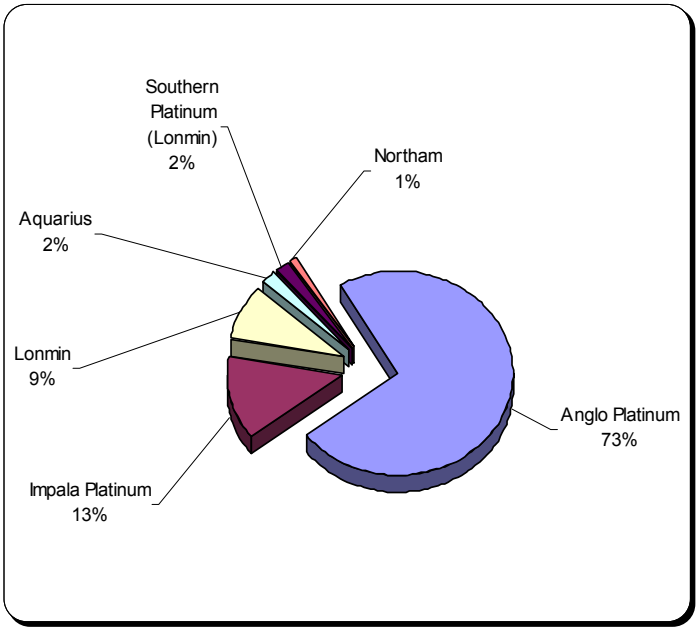


Figure 1.2: Relative contributions to total South African platinum production by mining companies in 2005
(Adapted from Chunnett, 2006)

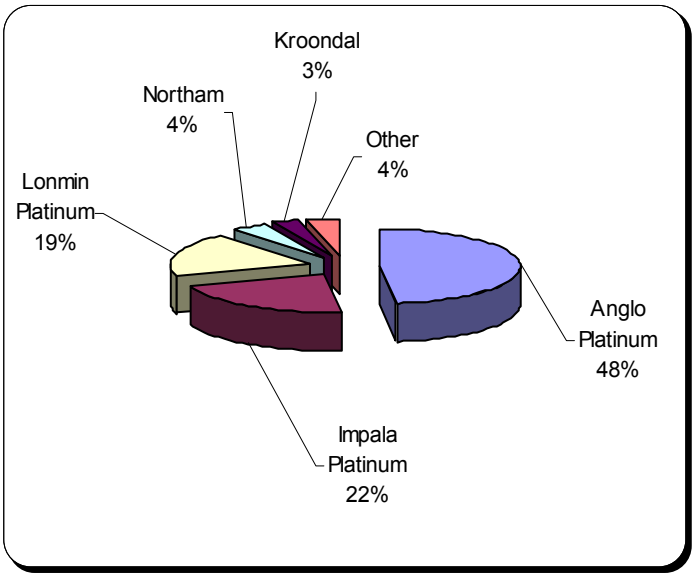


Figure 1.3: Relative contributions to total South African platinum production by mining companies in 2006
(Adapted from Pickering, 2007a)

1.3 Strategic importance of platinum mining on the Bushveld Complex

In the past decade, platinum mining on the Bushveld Complex has grown in terms of economic significance and strategic importance to South Africa due to a number of reasons. Firstly, when platinum is considered alone, the Bushveld Complex hosts approximately 63% of all the known world platinum resources and reserves (Figure 1.4). However, when platinum is considered together with other PGMs, the Bushveld Complex is host to an estimated 87% of global PGM resources and reserves (Chamber of Mines, 2005; Chamber of Mines, 2006). In terms of production, South Africa produced about 77% of annual global platinum production in 2005 (Figure 1.5), a figure that closely agrees with a most recent estimate of slightly over 75% of global output for 2008 (Research Channel Africa, 2009). This South African production level equates to slightly over 5 million ounces of platinum annually (Chunnett, 2006; Pickering, 2007a; Research Channel Africa, 2009). However, as reported by Research Channel Africa (2009), there are current platinum exploration activities in the Ural Mountains in Russia, Brazil, Canada, Botswana, Mozambique, Greenland, Madagascar and Zimbabwe by the mining companies Norilsk Nickel, Anglo Platinum and Impala Platinum. These exploration activities can possibly lead to some changes in the proportion of South Africa's share of global PGM resources and reserves.

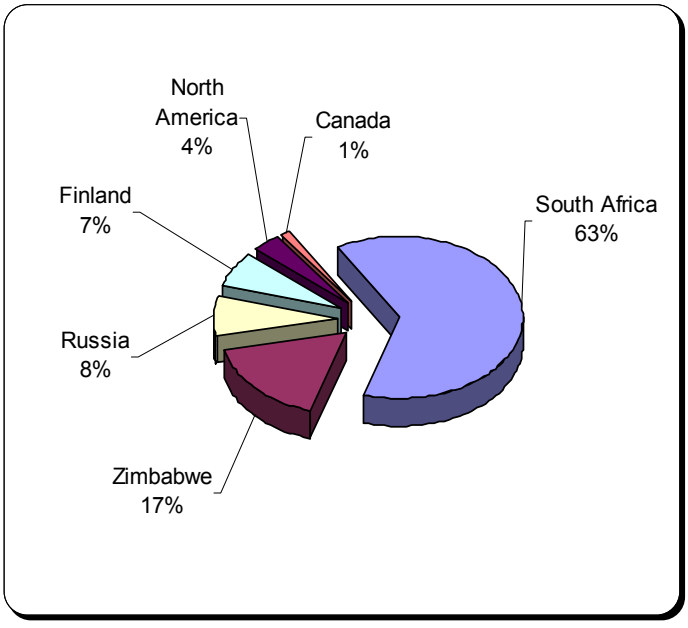


Figure 1.4: Relative proportion of global platinum resources and reserves by major countries in 2005
(Adapted from Chunnett, 2006)

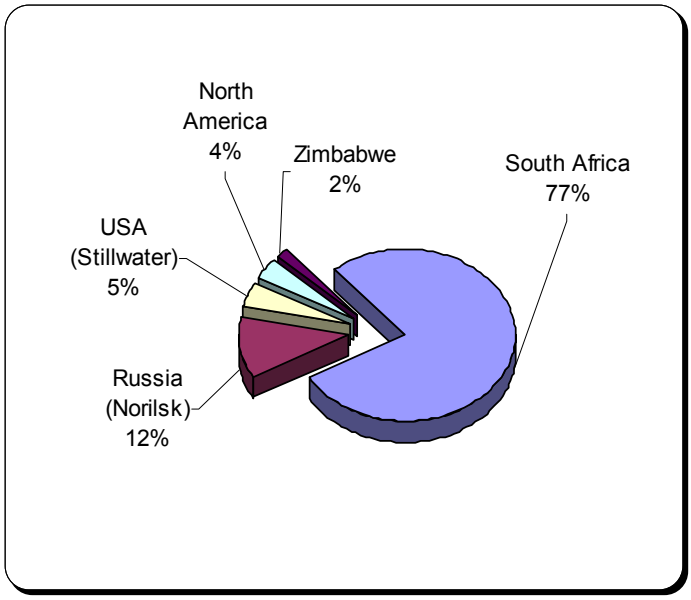


Figure 1.5: Relative proportions of global primary platinum production by major countries in 2005
(Adapted from Chunnett, 2006)

The rich PGM mineral endowment and high PGM production capacity attributable to South Africa give the country the enviable status of leading player in the international platinum industry. The impact of this status has been seen in the economic contribution of the platinum mining sector to the South African economy. For example, while gold production has been declining since 1980, platinum production has been increasing steadily (Figure 1.6). A similar trend is observed for labour employment by the two sub-sectors (Figure 1.7).

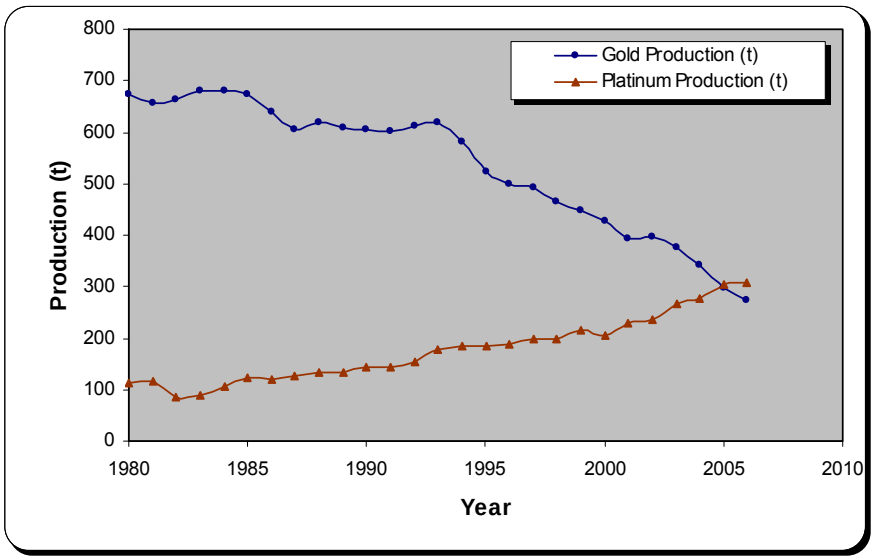


Figure 1.6: South African gold and platinum production
(Adapted from: Department of Minerals and Energy, 2005; Department of Minerals and Energy, 2006b; Department of Minerals and Energy, 2007; Chamber of Mines 2005; Chamber of Mines, 2006; Chamber of Mines, 2007)

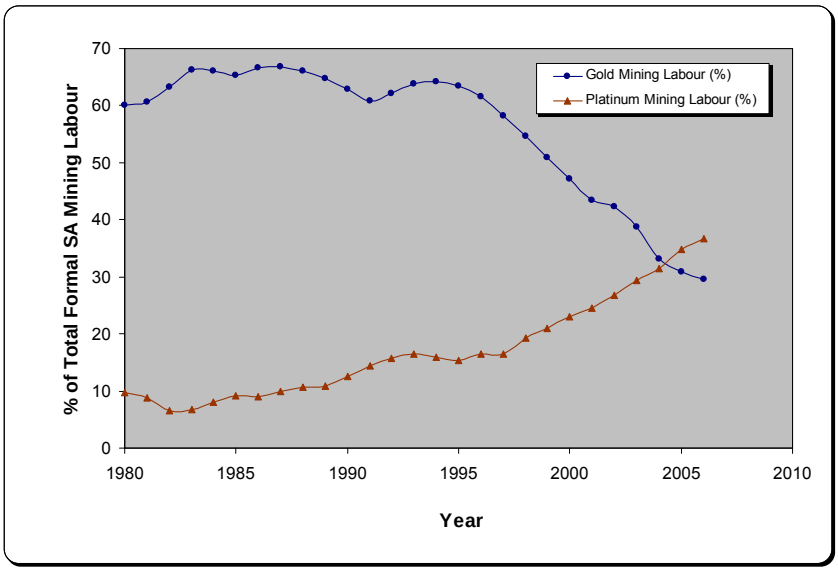


Figure 1.7: Gold and platinum labour as proportion of total formal South African mining labour
(Adapted from: Department of Minerals and Energy, 2007; Chamber of Mines, 2006; Chamber of Mines, 2007)

South Africa was famous for gold production for nearly a century but, in 2001 platinum surpassed gold in terms of Gross Domestic Product (GDP) contribution to the economy (Ruffini, 2005). Since then platinum has performed nearly as much as gold or out-performed it in terms of GDP contribution (Figure 1.8) and is now the mainstay of South Africa's mining industry. This fact is noted by the Chamber of Mines (2005:32) that, *"The pgm industry is the largest component of the SA mining sector"*. Ruffini (2005:80) also highlighted this fact when quoting Bernard Swanepoel, the then Chief Executive Officer of Harmony Gold, as having said, *"South Africa is no longer the gold mining industry and the gold mining industry is no longer the South African economy, and we are making the painful adjustment"*.

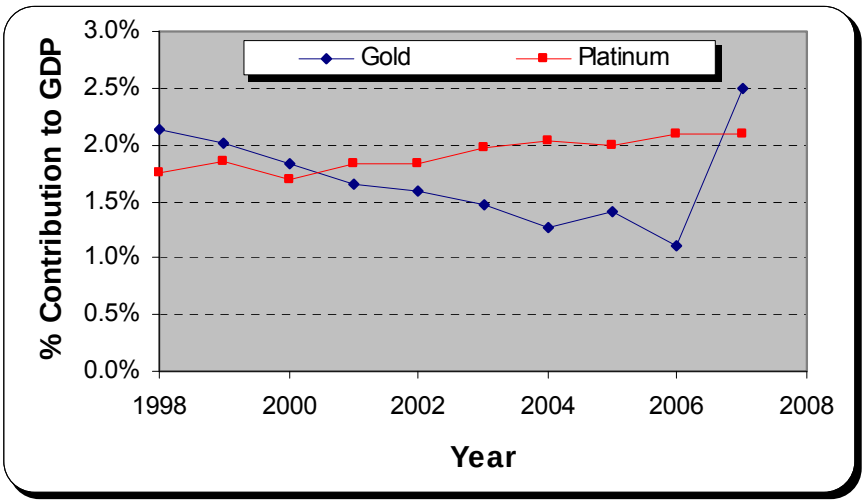


Figure 1.8: Relative contributions to South Africa's GDP by gold and platinum sectors
(Adapted from: Njowa, 2006; Chamber of Mines, 2005; Chamber of Mines 2006; Chamber of Mines, 2007)

Platinum's importance is also seen in its contribution to infrastructural development in South Africa as evidenced by the rapid growth of the town of Rustenburg. Lastly, community upliftment of the Royal Bafokeng Nation in the North-West province, supported initially by royalty receipts and subsequently by shareholding of the Royal Bafokeng Holdings (RBH) in Impala Platinum and other platinum mining companies, also portrays platinum as a significant mineral to the South African community.

1.4 Obligation or responsibility for optimal extraction

The obligation or responsibility to optimally extract the PGM resources is founded on principles of sustainable development, the need to maintain South Africa's strategic competitive advantage of being the leading platinum producer, responsibility of mining companies to provide high returns on investment to shareholders, findings of recent research and development studies, and legal requirements. These are briefly discussed below.

The sheer size of the platinum resources and reserves obscures the fact that the mineral resources are a wasting asset and must therefore be extracted optimally in order to ensure sustainable production (Stilwell and Minnitt, 2006). The responsibility to optimally extract the ore lies primarily with the mining companies for the benefit of their shareholders, citizens of the country, the rest of the world and future generations because the country's natural monopoly on PGMs must be used wisely. Optimal extraction can be achieved if the mining methods are optimised. For example optimisation in conventional mining, a prevalent platinum method on the Bushveld Complex, includes optimising level and raise spacing.

The growing strategic and economic importance of platinum mining to the South African economy makes it imperative for producers to optimally extract the mineral if South Africa is to maintain its position as the leading global producer and resource base of PGMs. More work could therefore be directed at ensuring that the mining methods in use on the Bushveld Complex are optimised, particularly for new, expansion or replacement projects. Again, optimisation of conventional mining methods includes, optimising level and raise spacing.

In 2006 South Africa's major mining research organisation, the Council for Scientific and Industrial Research (CSIR-Miningtek), undertook a survey of research and development (R&D) needs of the South African platinum mining companies and noted that out of 19 possible R&D areas, layout optimisation is one of the four priority R&D focus areas (Singh and Vogt, 2006). The four critical R&D areas are optimisation of mining layouts, mechanical rock breaking or cutting, excavation support and pre-concentration of ore in underground workings before it is hoisted to surface. Optimisation of conventional mining layouts includes, optimising level and raise spacing.

South African legislation, through the Mineral and Petroleum Resources Development Act (MPRDA) of 2002, empowers the State with the discretion to force the holder of mineral rights to a development project to suspend operations if the State is of the opinion that the holder is not optimally mining the mineral resources. This provision is contained in Section 51 of the MPRDA. Therefore from a legal perspective, mining companies need to optimally extract mineral resources. Again, the optimisation of conventional mining layouts includes, optimising level and raise spacing.

The foregoing perspectives highlight the imperatives for optimising platinum mining methods in South Africa. In order to optimise the mining methods it is essential to firstly understand the geological setting of the platinum resources and the associated geo-technical challenges *vis-à-vis* the mining methods in use. The next sections therefore describe the general mining geology of the Bushveld Complex and the mining methods used for ore extraction.

1.5 A general mining geology of the Bushveld Complex

The Bushveld Complex is approximately 2 billion years old (Walraven, Armstrong and Kruger, 1990; Jagger, 1999; Ackerman and Jameson, 2001). It is a saucer-shaped layered igneous intrusion into the Transvaal Supergroup and has a thickness of about 7km-9km (Pincock, 2008; Research Channel Africa, 2009). The Bushveld Complex is regarded as the world's largest layered igneous intrusion with an areal extent of over 65,000km^2 (Cawthorn, 1999; Research Channel Africa, 2009). Its surface footprint is characterised by two main arc-shaped limbs that measure about 300km along the north-south axis and are separated by a distance of about 450km along the east-west axis at their furthest points (Cawthorn, 1999; Jagger, 1999; Ackerman and Jameson, 2001; Pincock, 2008). Figure 1.1 illustrated the aerial appearance of the three limbs of the Bushveld Complex. The reserve estimations of the Bushveld Complex done by Cawthorn (1999) indicated that there could be about 204 million ounces of platinum and 116 million ounces of palladium in the proven and probable reserves categories, respectively. The inferred resources extrapolated down to a depth of 2km were estimated at about 939 million ounces of platinum and 711 million ounces of palladium (Cawthorn, 1999). The proven and probable ounces equated to about 40 years of mining at production rates prevailing at that time.

The Bushveld Complex is exploited for its three distinct reefs namely the Merensky reef named after Dr. Hans Merensky, the Upper Group 2 (UG2) chromitite reef and the Platreef. The Platreef which occurs on the northern limb only, is mined by the Mogalakwena Section of RPM (Cawthorn, 1999; Bye, 2003; Research Channel Africa, 2009). It can be traced for about 30km along the longer axis of the northern limb (Cawthorn, 1999; Research Channel

Africa, 2009). The Merensky and UG2 reefs occur on both the eastern and western limbs of the Bushveld Complex but not on the northern limb. In terms of stratigraphic succession, the UG2 underlies the Merensky reef and the middling between the two reefs varies within the range 25m-200m, from location to location along the Bushveld Complex (Figure 1.9).

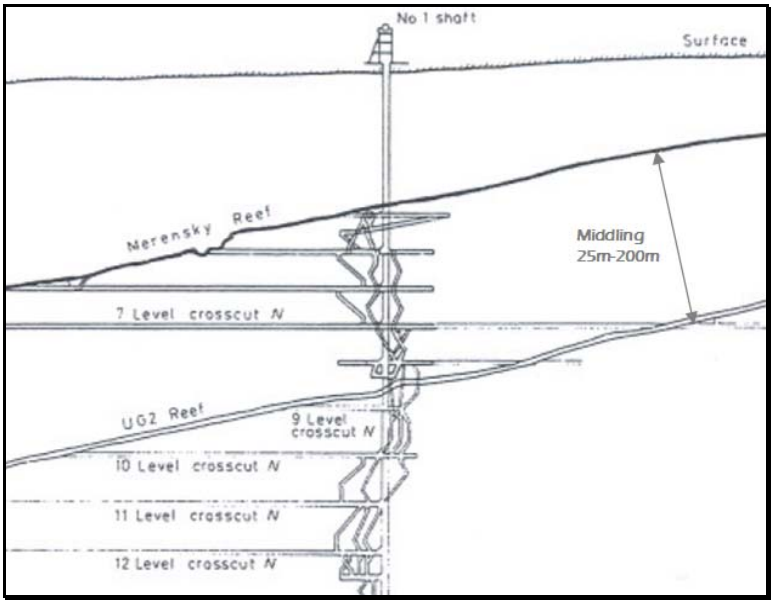


Figure 1.9: Typical section across a Bushveld Complex mine showing the Merensky overlying the UG2 reef
(Adapted from Buchanan, 1987)

The middling poses geo-technical stability challenges when extracting both reefs, particularly when the middling is small. These challenges are real because the Merensky reef, which is almost depleted on most shafts, has historically been the preferred reef that mining companies have been mainly extracting due to the metallurgical-related reasons given in Section 5.11, but due to technological developments the magnitude of UG2 mining has been increasing. There are also other geo-technical challenges posed by the rock type succession within the UG2 stratigraphy that require the footwall development to be optimally located as indicated by Figure 1.10. Figure 1.11 illustrates the actual rock types so that the reader can correlate the stratigraphic column presented in Figure 1.10 to the actual mining environment encountered underground.

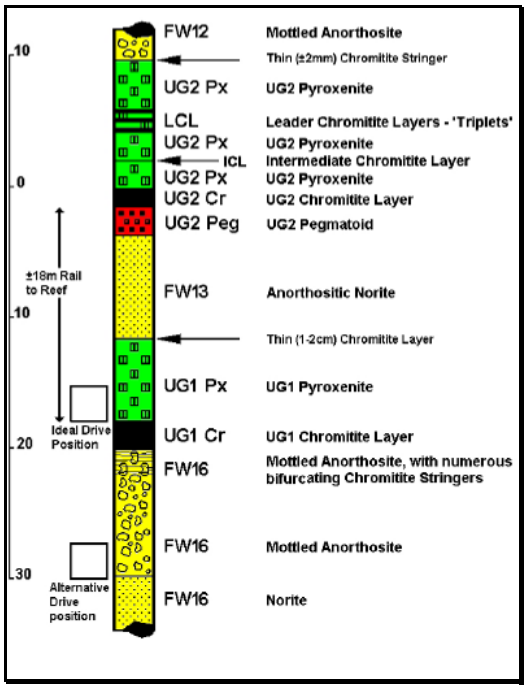


Figure 1.10: General UG2 stratigraphy showing optimal location of footwall development
(Source: Impala Platinum Mining Projects, 2003)

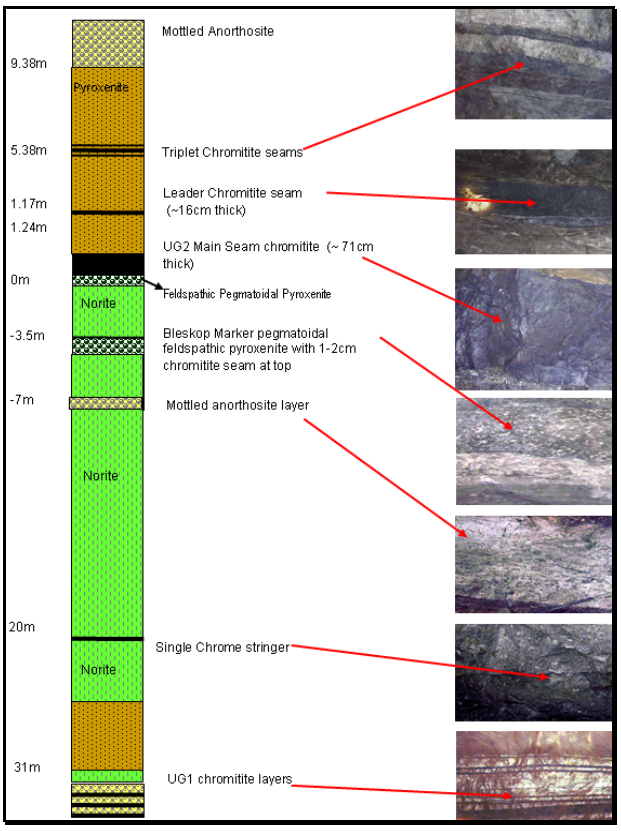


Figure 1.11: Typical UG2 stratigraphy showing the actual rock type appearance
(Source: Anglo Platinum, Boschfontein Shaft, Geology Department, 2004)

The Merensky and UG2 reefs are examples of shallow-dipping, inclined narrow tabular reefs. The reefs are shallow-dipping because they display a fairly consistent dip that ranges from 9^0 - 25^0 with an average dip of about 10º (Watson, 2004). Unlike mining methods on steeply-dipping reefs that rely solely on gravity to move blasted ore material, mining methods on shallow-dipping reefs must use mechanical means of ore movement, such as scraper winches, to clean blasted ore from the stope faces because gravity alone will not be sufficient. Moxham (2004) says that platinum reefs are regarded as narrow because their average thickness is typically less than 1m. The reefs are tabular because they exhibit lateral continuity on a regional scale characterised by fairly consistent planar geometry both on strike and dip. However, the lateral continuity is often disrupted at local mine scales by essentially four major geological structural disturbances namely potholes, iron-rich ultra-mafic replacement pegmatite intrusions (IRUPs), faults and dykes (Schoor and Vogt, 2004). The 'Triplets' shown in Figure 1.10 and Figure 1.11 are chromitite stringers which represent a geo-technical plane of weakness that poses a danger of large scale hangingwall collapse. Every effort is therefore made to ensure that the hangingwall is adequately supported. Typically when the parting from the top contact of the UG2 chromitite reef to the base of the triplet package is less than 50cm, the stoping width will have to include the triplet package, thus increasing dilution. If the triplets are not mined in such a case, large scale hangingwall collapse usually occurs with accompanying employee fatalities and equipment damage.

1.5.1 Potholes

A pothole is a geological disturbance in which the reef and its associated stratigraphy would have slumped or subsided to elevations below the expected footwall horizon of a uniformly dipping continuous reef plane (Figure 1.12). The slumping is usually in the form of a nearly circular or elliptical block of ground when viewed in plan view. Ralph Morris and Associates (1994) estimated median pothole sizes to be about 28m to 37m across the widest points for a sample of 1,000 potholes on the Bushveld Complex. The same study revealed that 85% of the potholes were less than 5,000m^2 in areal extent and that 76% of the potholes were less than 11m deep. Potholes as wide as 100m or even greater in diameter can occasionally be encountered. Most potholes are not mined because they fall outside the reef horizon for which development and production infrastructure is designed to service. Generally UG2 potholes tend to be erratic and are not mineable. Re-raising around potholes is often done to circumvent the potholes.

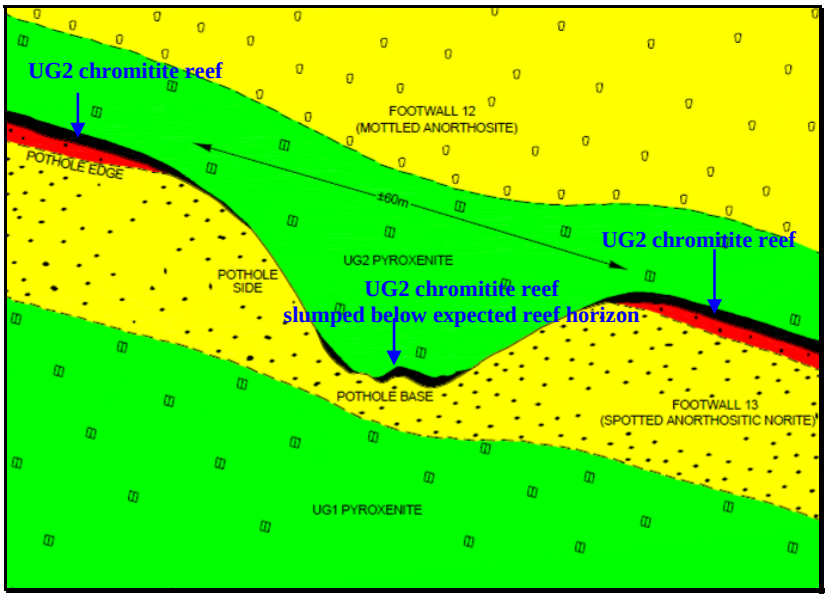


Figure 1.12: Section across a typical UG2 pothole
(Adapted from Impala Platinum Mining Projects, 2003)

1.5.2 Rolling reef

Rolling reef describes the geological disturbance in which the reef rolls in an irregular wave-like manner such that the reef occurs repeatedly below and above its expected datum. The amplitude and wavelength of reef rolls is variable but generally is of the order of 3m to 30m in plan and 1m to 4m in depth (Ralph Morris and Associates, 1994). Reef rolls tend to be shallow and mineable, while most potholes are not.

1.5.3 Dykes and faults

Reef faulting on the Bushveld Complex is generally of relatively small magnitude in the order of a few metres of vertical displacement. However, faults with vertical displacement in excess of 25m can be encountered (Ralph Morris and Associates, 1994).

1.6 A brief history of mining methods on the Bushveld Complex

Mining of the Bushveld Complex platinum reefs has historically progressed through four distinct generations of mining (Ralph Morris and Associates, 1994; Lanham, 2005). The first generation shafts were mainly inclined shafts developed on reef from reef outcrops and the 'hand-got' herringbone mining method was the method used initially for ore extraction. As depth of mining increased, second and third generation shafts provided access to mine these down-dip reef extensions and conventional breast mining was introduced to replace the 'hand-got' herringbone mining method. With the development of trackless mining

machinery, mechanised room-and-pillar mining was subsequently introduced into narrow reef tabular mining. Zindi (2008) described a 'fourth generation' shaft project currently being pursued by Impala Platinum to extract reefs at deeper mining levels which have more geo-technical challenges. Current mining depths on the Bushveld Complex range from outcrop to about 2,300m below surface (Watson *et al*, 2008a). Northam is the deepest mine with mining activities occurring between 1,300m-2,300m below surface on the down-dip side of Anglo Platinum's Amandelbult mine (Northam, 2009). Mining methods currently practised on the Bushveld Complex can be categorised as illustrated by Figure 1.13.

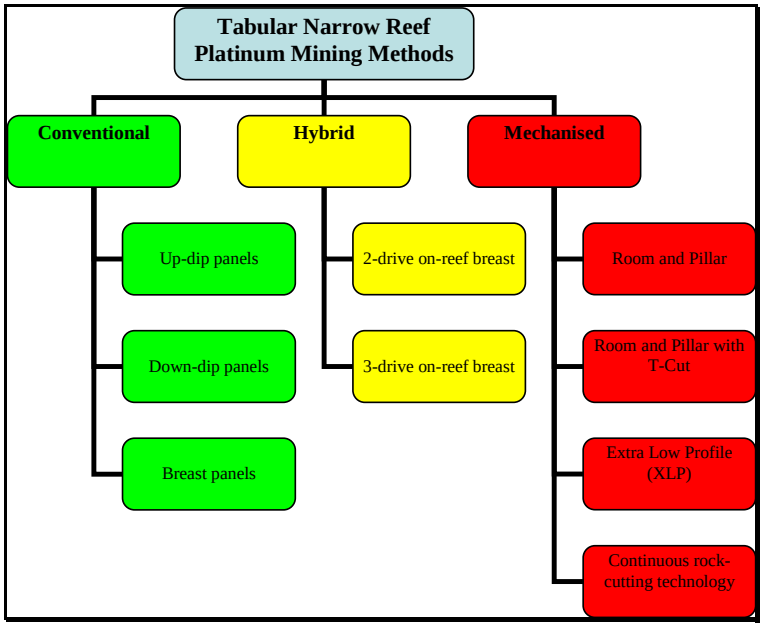


Figure 1.13: Topology of South African platinum underground mining methods

The current mining methods used to extract the platinum reefs include conventional mining (with its variants such as scattered breast mining, up-dip or down-dip mining), room-and-pillar mining, modified room-and-pillar (the *T-cut* method) and the most recent Extra Low Profile (XLP) mechanised room-and-pillar. The conventional mining methods, which rely heavily on scraper winches as a mechanical means for ore cleaning in production stopes, are labour intensive and have low productivity. Hybrid mining methods were developed to maintain the advantages of conventional mining on reef such as low dilution and a higher shaft head grade, while adding the many advantages of mechanised development such as faster development rates and safer operating procedures (Egerton, 2004). Typically hybrid mining utilises scrapers for panel cleaning, but use conveyor belts or LHDs for strike transportation of ore. LHDs and conveyor belts have higher productivity than scrapers. Mechanised mining methods have higher productivity and are considered to be more efficient than conventional mining methods. Trials of non-explosive continuous rock-cutting technology such as the ARM 1100 machine at Lonmin's Rowland Shaft in 2002 and Impala

6# in 2004 (Moxham, 2004; Pikcering, Smit and Moxham, 2006) and the most recent rock-cutter introduced at Anglo Platinum's Townlands shaft by DBT, can be considered to be the future fourth generation mining technology. Sandvik mining, as part of its strategy to market its equipment, regularly undertakes industry surveys and predictions based on feedback from mining companies. In 2007 Sandvik undertook a survey from which Pickering (2007a) reported that conventional mining was the most prevalent mining method on the Bushveld Complex. The distribution of production output from the Bushveld Complex by mining method in 2005 and the prediction for 2010 is illustrated in Figure 1.14 and Figure 1.15.

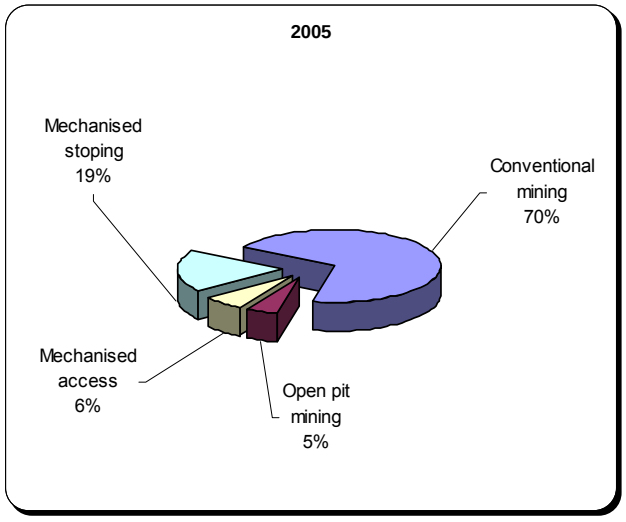


Figure 1.14: Distribution of PGM production output by mining method in 2005
(Pickering, 2007a)

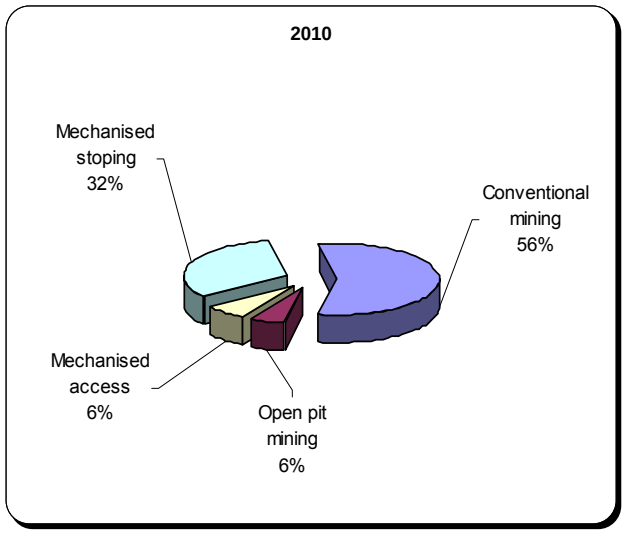


Figure 1.15: Forecast 2010 distribution of PGM production output by mining method
(Pickering, 2007a)

Figure 1.14 and Figure 1.15 indicate that conventional mining is likely to remain the principal platinum mining method in the medium to long term, a paradigm that is further supported by the following cues:

- Moxham (2004) and Pickering, Smit and Moxham (2006) noted that despite efforts to mechanise the South African narrow hard rock reef mining industry in the last 40 years, almost all mechanised mines have reverted back to conventional mining.
- Lonmin announced its intentions about 10 years ago that by 2010 at least 50% of their PGM production would be coming from mechanised mining. However, it is no longer certain if Lonmin is still on course since a mechanisation project at their Saffy Shaft is now being converted to conventional mining. The proportion of production attributable to conventional mining in 2010 could be higher than the 56% shown by Figure 1.15.
- Northam, the deepest platinum mine is still using conventional breast mining but have adapted it to use hydro-powered equipment (HPE) instead of pneumatic equipment. This observation suggests that conventional mining can still be practised at deeper mining levels.
- If a cue is to be taken from the Witwatersrand gold mines using a modified form of conventional breast mining called Sequential Grid Mining (SGM) at depths of about 3,000m-5,000m below surface and the fact that the recent 'fourth generation' Impala Platinum 16# and 20# projects were planned on conventional mining (Jagger, 2006; Zindi, 2008), then platinum mines might be expected to be practising some form of conventional breast mining when they progress to similar depths of mining.
- Egerton (2004) analysed eight different mining methods to mine the UG2 reef. Musingwini and Minnitt (2008) further analysed the results using the Analytic Hierarchy Process (AHP) and noted that conventional mining ranked highest.
- The orebody will always dictate the mining method. The extreme hardness and high abrasivity of particularly the UG2 reef, due to the presence of chromite crystals making up the structure of the UG2, make it difficult to introduce rock cutting technology (Moxham, 2004; Pickering, Smit and Moxham, 2006). The rolling reef nature of the UG2 and Merensky as described in Section 1.5.2 make it difficult to implement mechanised mining. Again, this favours the continued use of conventional mining.
- The reality of mineral price cycles, intermittent strengthening of the ZAR/US$ exchange rate, and the associated cost-cutting measures have invariably resulted in occasional mothballing of mechanised mining projects because they are capital intensive and hence sensitive to such real changes which occur from time to time (Egerton,2004). Again, this favours the continued use of conventional mining.

The continued use of conventional mining also derives from some of the advantages it has over mechanised or hybrid mining methods which include:

- Access to the orebody is in the footwall allowing the flexibility to hoist development waste separately from the reef ore (Egerton, 2004).
- Unlike other mining methods, conventional mining allows for greater flexibility in negotiating prevalent faults in a tabular orebody (Carter, 1999).
- Conventional mining provides the mining engineer with more flexibility to carry out *ad hoc* advance exploration ahead of the advancing stope faces to locate the actual positions of smaller geological structures that were not picked up during initial exploration hence selective mining can be done (Jager and Ryder, 1999).
- Unlike other methods, conventional mining is better suited to follow the rolling reef that was described earlier in Section 1.5.2 and commonly encountered on the Bushveld Complex.
- Conventional mining has lower dilution and therefore delivers a higher shaft head grade compared to other mining methods (Egerton, 2004; Moxham, 2004).
- Conventional mining has lower capital costs compared to other mining methods and so can be implemented even during times of low mineral prices or strong ZAR/US$ exchange rate. The decline in mineral prices experienced from mid-2008 are expected to impact negatively on mechanisation efforts and support the continued use of conventional mining in the South African platinum industry.
- Downgrading events such as fire, rock bursts or explosions are less disruptive in conventional mining compared to hybrid or mechanised mining (Fleming, 2002).

Although there are significant disadvantages which weigh against conventional mining, particularly being very labour intensive and giving lower productivities (Egerton, 2004; Moxham, 2004; Pickering, Smit and Moxham, 2006); its advantages seem to weigh more in its favour, hence its continued use. The next section describes the variations to conventional mining and explains why the laybye access variation was selected for this study.

1.7 Conventional mining

Conventional mining is, and has been, used widely in the extraction of shallow-dipping, narrow tabular reefs in the gold, platinum and chromitite sectors in the South African mining industry (Egerton, 2004; York, 1999). It is an example of a partial extraction mining method because part of the reef is not extracted and left *in-situ* as pillars. A brief description of conventional mining now follows to orientate the reader.

Primary access to the reef horizon is by vertical or inclined shafts, or via declines (or a cluster of dedicated declines) located in nearly the centre of gravity of the orebody in order to minimise haulage distances from the production areas. After shaft sinking or decline development is completed and associated capital infrastructure development has been established, lateral tunnels called main crosscuts or main haulages are developed from the shaft area in a direction approximately perpendicular to the strike direction and are stopped at a pre-determined distance below the reef horizon. The distance below the reef horizon at which the crosscuts are stopped is mainly dictated by geo-technical considerations to ensure that the development is located in geo-technically stable and competent ground within the stratigraphic column as indicated earlier on in Figure 1.10. The distance should also enable boxholes to be long enough to have sufficient capacity for handling ore from the reef horizon (Impala Platinum, 2007b). Another set of lateral tunnels called footwall strike drives or haulages are then broken away from the crosscuts and developed along strike direction, below the reef horizon. All lateral haulages are developed on a gradient typically about 1:200 to allow water to gravitate back to the main sumps or dams in the shaft area and also take advantage of gravity in aiding fully loaded locos tramming ore from the mining horizon back to the main shaft area for eventual hoisting to surface. The development of the orebody on reef horizon is then done on a grid pattern of footwall drives spaced at pre-determined level spacing on dip and raises (and winzes) spaced at pre-determined spacing on strike (Figure 1.16). The smallest self-contained production unit encompassing all mining processes such as development, ledging, equipping, stoping, vamping, reclamation, and ancillary services such as tramming, maintenance, service and construction, and lying within an area demarcated by two levels on either side of the central shaft position is called a half-level (Anglo Platinum MTS, 2005; Smith and Vermeulen, 2006). For example for the two main levels shown in Figure 1.16, there are four half-levels. However, a main level can consist of more than two half-levels if mining is on more than one reef plane as the case experienced on platinum mines concomitantly extracting the UG2 and Merensky reefs.

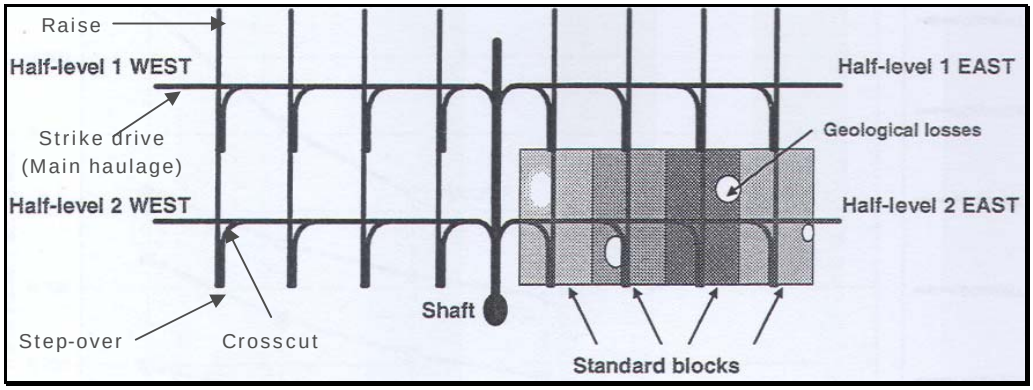


Figure 1.16: A schematic illustration of crosscut access conventional mining in plan view
(Adapted from Smith and Vermeulen, 2006)

There are basically three conventional mining variations which can be distinguished by their different half-level configurations. These variations are the crosscut access up-dip or down-dip mining (mainly practised by Lonmin), crosscut access breast mining (mainly practised by Anglo Platinum and Northam) and laybe access breast mining (mainly practised by Impala Platinum). However, all the variations use similar stope rock handling systems which rely on scraper winches for ore movement. Typically a small duty scraper winch is used for cleaning the panel face to tip into the Advanced Strike Gulley (ASG) while a higher duty scraper winch is used for moving ore along the ASGs and the main gulley (i.e. the raise) as shown in Figure 1.17. Key features of each half-level variation are described in the next sub-sections.

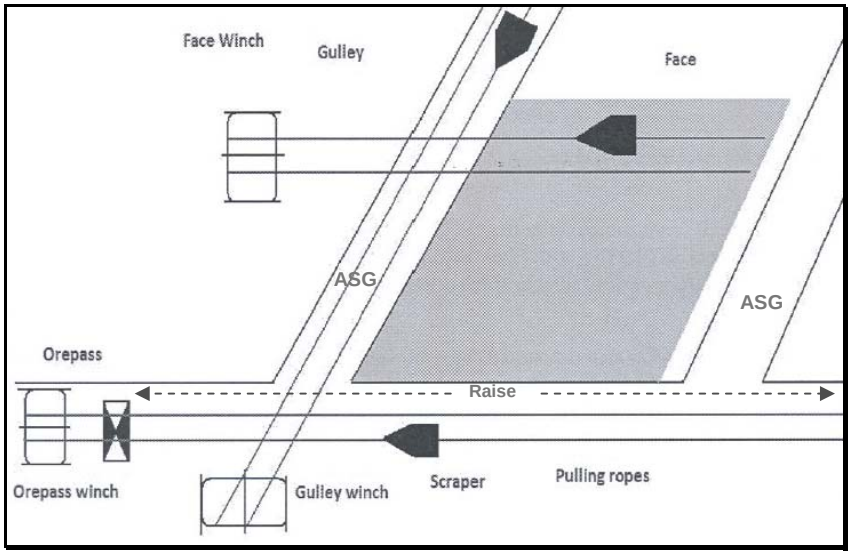


Figure 1.17: Typical scraper winch arrangement in a conventional mining stope panel layout
(Courtesy of Anglo Platinum MTS)

1.7.1 Up-dip and down-dip mining

Conventional up-dip mining is the predominant mining method practiced in most of the underground mining operations at Marikana Division in Rustenburg, a subsidiary of Lonmin Platinum (van den Berg, 2007). A typical up-dip half-level layout is illustrated by Figure 1.18. A footwall strike drive (or main haulage) is developed from the central primary shaft or decline access area. Short 15m crosscuts are broken away from the footwall strike drive towards the reef horizon at 70m intervals. From each crosscut two travelling ways, each about 20m long, are developed to intersect a raiseline position. From the strike drive boxholes are developed every 35m to hole into a raiseline position. Step-over and tip areas are then established at the intersections of the reef horizon and, travelling and boxhole holing positions, respectively. Once the tip area has been equipped with scraper winches, raises each about 200m long are developed up-dip until they reach a holing position on a

consecutive upper level. Stope Preparation Drives (SPDs) are subsequently developed to connect the raises (Figure 1.19).

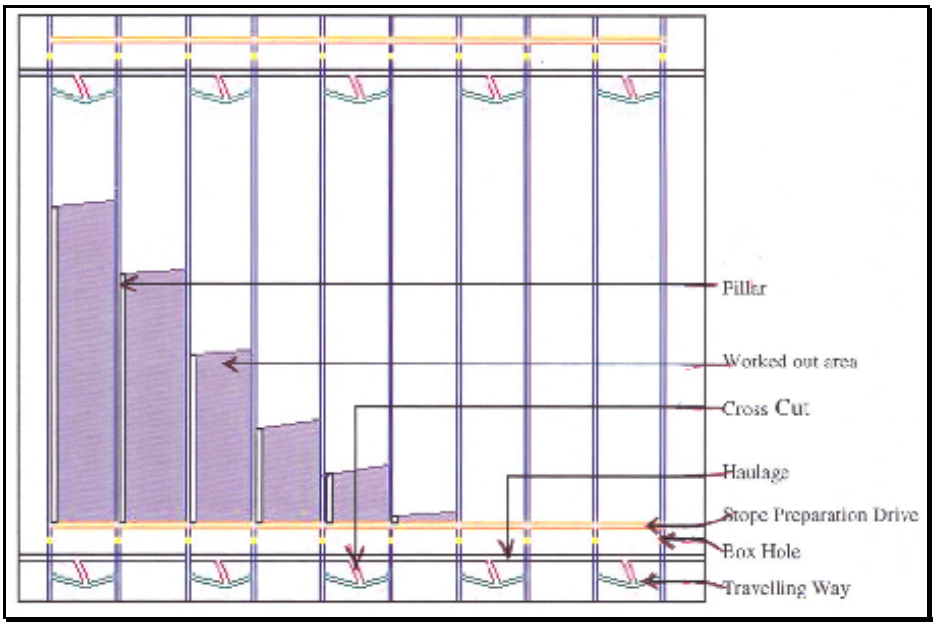


Figure 1.18: Schematic layout of a typical half-level for conventional up-dip mining
(van den Berg, 2007)

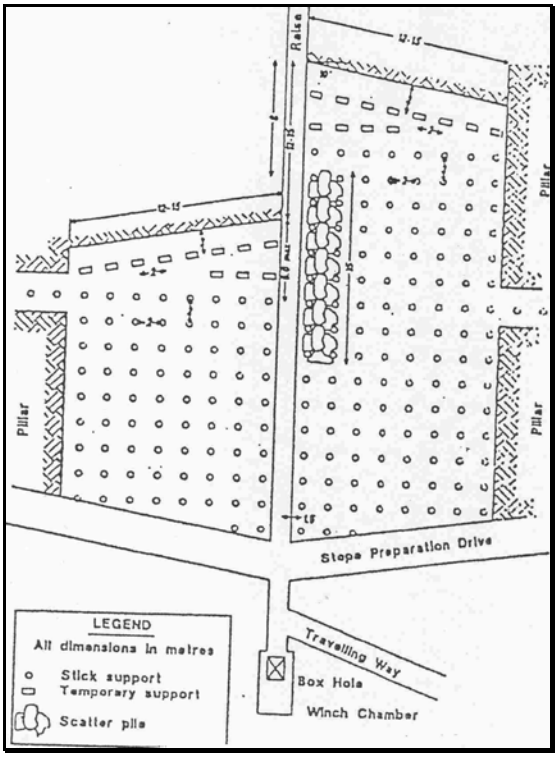


Figure 1.19: Schematic illustration of position of SPD relative to production panel
(Fleming, 2002)

The raise is then ledged to prepare panels for production which are then advanced up-dip (i.e. in the opposite direction to dip direction). In other words panels advance up-dip in an overall direction that is almost parallel to the raiselines. Stope faces are stopped when they hole into the stope above, reach a stopping line, or encounter a fault, dyke or pothole. Geological disturbances are circumvented by developing new SPDs beyond the boundaries of the geological structure. Typically panels are about 28m long with a provision for 5m wide dip pillars that serve as regional support. In this layout, raises and crosscuts are spaced about 35m apart. Therefore raises are spaced about 70m apart while levels are spaced about 35m vertical interval if an average dip of 10º is assumed. In other words, levels and raises are closely spaced in conventional up-dip mining layouts.

1.7.2 Crosscut access breast mining

Egerton (2004) provides a succinct description of the crosscut access conventional breast mining method which is illustrated in Figure 1.20 and Figure 1.21. Main crosscuts from the shaft are developed to about 10m-80m below the reef horizon depending on local mine geology and standards in use. Footwall strike drives then break away from the main crosscuts and are developed on strike.

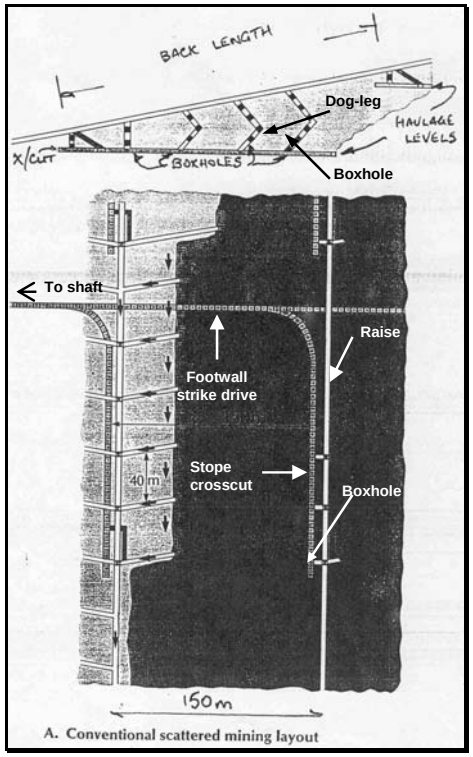


Figure 1.20: A typical crosscut access conventional mining layout
(Fleming, 2002)

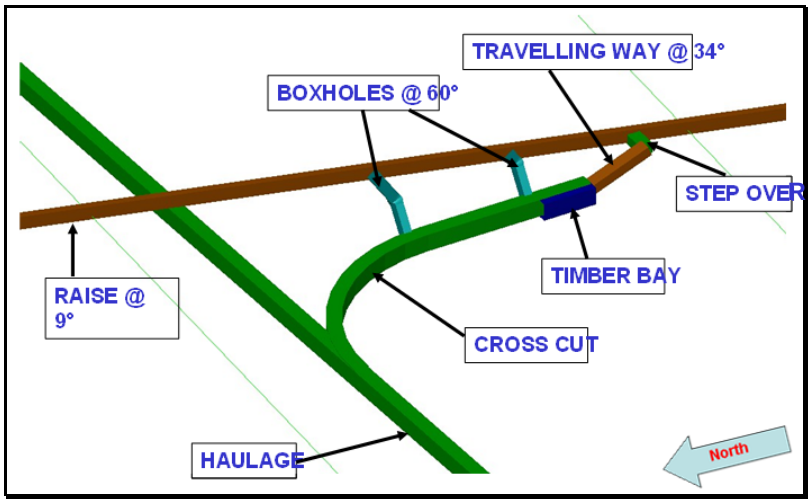


Figure 1.21: A 3-D perspective of a crosscut access layout at a raise connection in Mine 2-4D®

From the footwall strike drives, stope crosscuts to reef are broken away at intervals of about 200m and stopped just below the reef horizon so that a timber or transformer bay is developed at the end of the stope crosscut and a travelling way about 10m-20m long at 34.5º to the horizontal is also developed from the end of the stope crosscut to intersect reef. At the intersection of the travelling way and reef horizon a lateral tunnel called a step-over is developed to connect the travelling way and raiseline position. After a step-over is completed, a chamber called a tip area is excavated usually on the down-dip side of the step-over and along the raiseline.

Once the tip area is completed, a boxhole which would have been started from the stope crosscut below will come and hole into the tip area. Subsequently a grizzley and a gulley scraper winch are installed on the tip area. The first boxhole must be located such that the minimum over-run distance along the stope crosscut is about 25m in order to accommodate a train of 8 hoppers during shunting when loading hoppers from the boxhole. A raise is then developed on dip, from the tip area for about 240m to hole into the next upper level. The dip distance along a raiseline between two consecutive levels, the 240m in this case, is called the backlength (Figure 1.20). As the raise is advancing up-dip, boxholes are also started from the stope crosscut to hole into the raiseline. Typically a maximum of three boxholes are developed as shown in Figure 1.20 and are equipped with manually controlled pneumatic chutes for loading the hoppers. The number of boxholes per stope varies from one per panel to one per entire stope. Typically one boxhole per panel ensures higher face advance. Layouts which provide a boxhole per panel require long cross-cuts and small inter-level spacing and the cost of footwall development must be weighed against economic benefits of increased advance rate (Fleming, 2002). The boxholes are 'dog-legged' as shown in Figure 1.20 so that the 'dog-leg' portion (i.e. where the boxhole bends and looks like an elbow) can

absorb the energy of ore falling down the boxhole, hence slow down the ore and avoid damaging the boxfront, alternatively called the chute (Anglo Platinum MTS, 2005).

Once the raise has holed, the raise shoulders are then ledged for about 6m on either side of the raiseline, and supported with 200mm diameter sticks on pre-stressed pods. Ledging secures the raise area for mining, scraping and ventilation. Advanced strike gullies (ASGs) and winch cubbies are then marked off and blasted. Typically ASGs are developed at about 20º-25º above strike in order to facilitate easier negotiation of reef rolls and to allow water to drain away from the stope to reduce water-logging in the stope. The ASGs are developed about 3m-8m ahead of panel faces for four main reasons. Firstly, to allow the ASG scraper to overrun face scraper so ensure that all the ore cleaned from the panel face can be scraped into the main boxhole. The second reason is exploratory because reef orientation ahead of the panel face can be known in advance of production drilling and proactive planning steps instituted. Thirdly, the ASG will serve as a free breaking face for the production panel nearest to the ASG. Lastly the limit of 8m is dictated by ventilation constraints because beyond this distance the ASG will have gone too far in, to require a force ventilation system to be installed. The ASG width of 1.3m is just wide enough to accommodate the scraper scoop but the height of 2.2m-2.5m is deep enough to provide storage capacity for ore scraped by the face scraper before being scraped into the centre gulley. The same logic applies to dimensions of raises and winzes; however gulley height depends on the rate and extent of stope closure depending on the depth below surface. For example, at depths of 1,500m below surface, raise excavations can start to scale off and deteriorate quickly if ledging is not done early enough (Fleming, 2002).

Panels which are about 25m-40m long and between 0.9m-1.8m in height each, are then mined advancing in strike direction, using the ASGs as free breaking face. Panel advance rates are typically in the range 0.6m-0.9m per blast and in a 23-day production month give the typical industry average monthly face advance rates of about 10m-15m. The stope scraper winch arrangement as illustrated earlier on in is a 37kW (≈50hp) face scraper winch, 56kW (≈75hp) ASG scraper winch, and 56kW gulley scraper winch (Egerton, 2004). Additional equipment which varies from mine to mine typically includes four hand-held pneumatic rock drills (and their ancillary attachments) per panel and a mono-rope winch per raiseline for transporting material into the stopes.

Stope support is in the form of in-stope (or in-panel) pillars with lengths ranging between 4m-6m and widths between 3m-4m depending on the official mine standards. The in-stope pillars are spaced with ventilation holings typically 2m-3m wide to allow ventilating air to pass through the stope. Temporary support is installed in the form of mechanical props down the face between the permanent support and the panel face, and spaced no further than 1.5m

apart on dip. Permanent support is in the form of 200mm diameter pre-stressed, non-yielding timber elongates, typically spaced 1.5m on dip by 2m on strike and never further than 4m from the face or tendons may be installed spaced 1.5m on dip and 1.6m on strike (Egerton, 2004). Figure 1.22 illustrates a typical conventional breast stope with the mined out area shaded in grey and hatched. The figure shows that barrier pillars as denoted by the number '1' are usually located to incorporate unpay ground or geological structures such as dykes and potholes. Geological structures are also circumvented by re-raising around the structure. The number '3' denotes support installed in the back-area behind the active stope face which is usually sandwich packs and elongates (Jager and Ryder, 1999; Egerton, 2004).

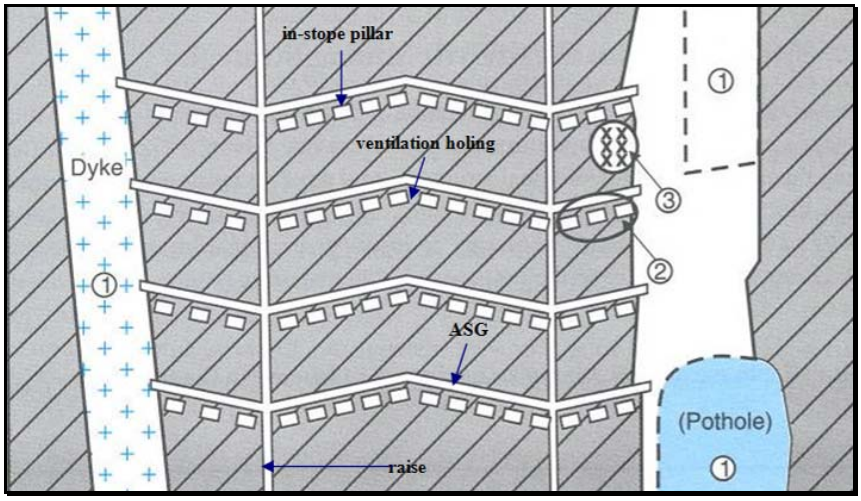


Figure 1.22: Longitudinal plan of a typical conventional breast layout
(Adapted from Jager and Ryder, 1999)

1.7.3 Laybye access conventional breast mining

The laybye access conventional breast mining is very similar in approach to the crosscut access conventional breast mining only that it differs in the way in which the reef horizon is accessed from the footwall strike drives as illustrated by Figure 1.23, Figure 1.24, and Figure 1.25 that are based on current Impala Platinum mining standards. Ackerman and Jameson (2001) described the laybye access conventional breast mining method based on an older version of Impala Platinum mining standards.

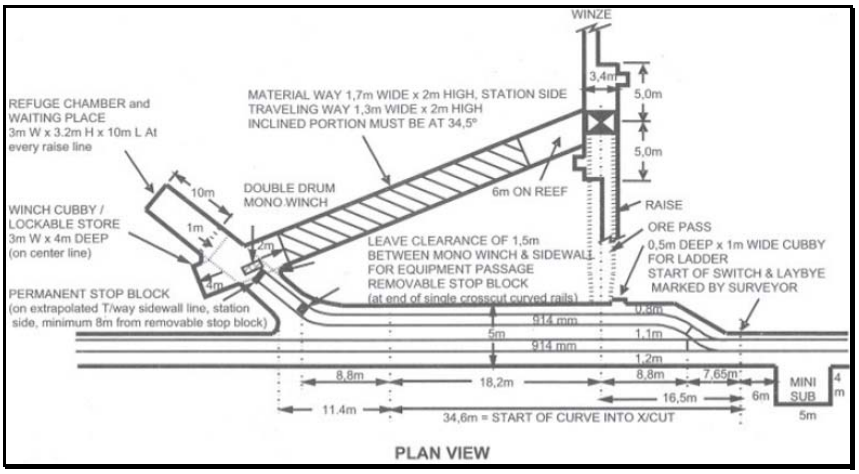


Figure 1.23: Plan view of a laybye access layout at a raise connection
(Impala Platinum, 2007a)

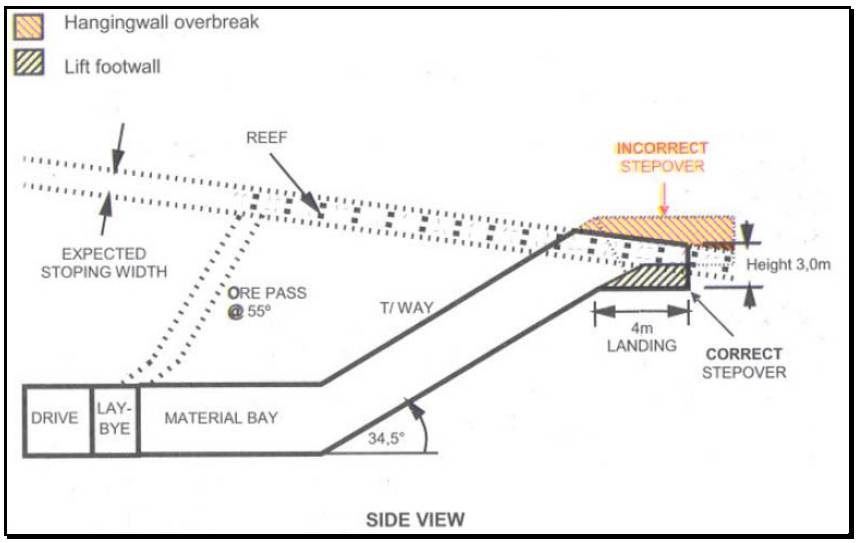


Figure 1.24: Side view across a laybye access raise connection
(Impala Platinum, 2007a)

At a laybye, the footwall strike drive is widened from 3m to 5m to cater for the loco and hoppers, boxhole loading chutes, and a second shunting rail line used for loading ore from the boxhole. A winch cubby 4m long is developed to house a mono-rope winch for material transportation into and out of the stope during stoping operations. The travelling way is developed at a maximum of 34.5° above the horizontal otherwise it becomes legally too steep to require a ladder and platforms to be installed. The travelling way is developed to the reef horizon and stopped at about 5m-8m away from raiseline and a step-over mined to connect the travelling way and raise. From the step-over a tip area is mined out along the raiseline. A boxhole stub is started off as a 2m stub at 37.5° above ground and then raise-bored at 55° - 60º to hole into the tip area once it has been completed.

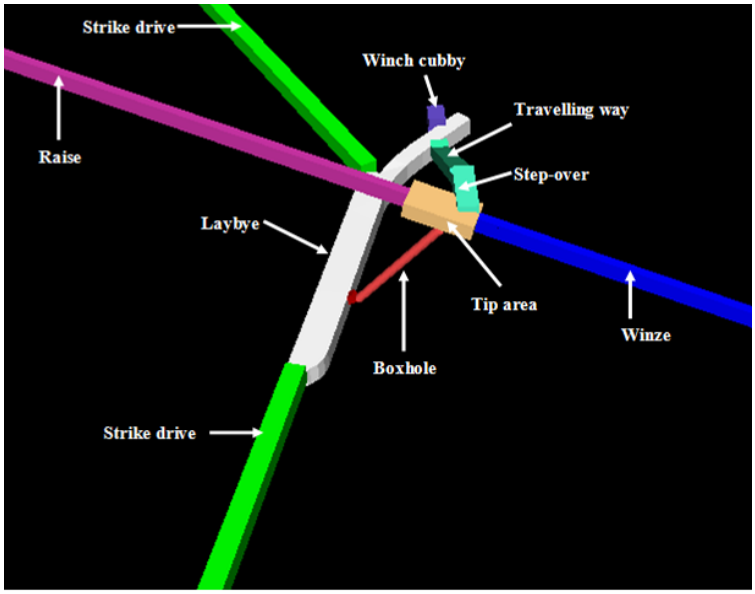


Figure 1.25: A 3-D perspective of a laybye access raise connection in Mine2-4D®

The inclination of 55º-60º is necessary to ensure that ore will flow because steeper inclinations will cause the ore to compact, particularly if it is the UG2 ore (Mohloki, 2007). Ideally, the travelling way should hole into the same position as the boxhole in the tip area or just above the boxhole so that water can drain down the travelling way and not into the boxhole. Water is undesirable in the boxhole, particularly with the friable UG2 ore because it can lead to mud rushes and loss of ore as fines. After holing the boxhole, it takes about 3 months to prepare the tip by installing a grizzley, and raise and winze scraper winches (Mohloki, 2007). Thereafter the raise is developed for about 200m up-dip while winzing is done for about 100m down-dip to give a total of 300m backlength. The up-dip scraping distance of 200m and down-dip scraping distance of 100m are dictated by scraper rope and gravity limitations as explained in Chapter 5. The ledging and stoping are carried out as in the crosscut access method but the scraper winch sizes used are slightly different. A 37kW scraper winch is used for face scraping, and 56kW and 75kW scraper winches are used for the ASG and gulley, respectively.

The laybye access method was the method chosen for this optimisation study for the following two reasons:

- A visual comparison of Figure 1.20 and Figure 1.23 shows that the laybye broken away from the footwall strike drive in a laybye access method is only about 46m long compared to a crosscut broken away from a strike drive, which can be about 100m long, in a crosscut access method. Additionally, the laybye access method has only one boxhole compared to the crosscut access method which can have up to four

boxholes per raiseline. It was therefore prudent from an optimisation point of view to choose the laybye access method because it cuts out a lot of access development without necessarily compromising productivity. As will be discussed in Chapter 2, one of the key issues in level and raise spacing optimisation is to reduce the amount of off-reef waste development because it is a cost item that does not generate any revenue. Further, one of the bottlenecks in conventional mining is the time taken to complete the development that connects the strike drives and the reef horizon and any re-design that can lead to less development is always desirable (Mitchell, 2009). This is also the reason why Anglo Platinum is currently re-designing the crosscut access method so that the crosscuts are much shorter (Mitchell, 2009).

- Impala Platinum uses the laybye access conventional method and was willing to provide a copy of the mining standards. The company was also willing to second a person to help with explaining the mining standards and thereafter review the Mine2-4D® designs and EPS® schedules. Mr. Lefu Mohloki, the then Projects Manager in Impala Platinum, was assigned to provide this help and regular visits were undertaken by the author to Impala Platinum Projects Offices in Rustenburg in 2007 and 2008 to have the designs reviewed.

1.8 Level and raise spacing on conventional mining layouts

The preceding sections indicated that level and raise spacing on conventional mining operations are quite variable for a number of reasons. This section describes a brief history and current practices that have contributed to the variability in level and raise spacing on conventional mining operations.

1.8.1 Brief history on level spacing on inclined tabular reefs

Primary access to inclined tabular reefs has historically been either by developing inclined shafts or declines on-reef or in the reef footwall, or vertical shafts in the reef footwall around the centre of gravity of the orebody. In the early to mid-1900s when imperial units were still used in the South African mining industry, main levels were cut at 100ft (≈30m) vertical intervals for convenient reference, as it would be easier to count in multiples of 100 (Cruise, 2005). Later, it became necessary to construct water reservoirs underground for supplying drilling machines with water at pressures of at least 90psi. This new demand required levels to be cut at longer vertical intervals of 200ft (≈60m) and reference them in multiples of 200 (Cruise, 2005). Since then main levels on inclined tabular reefs have been cut at vertical intervals in the 30m - 70m range (Cruise, 2005).

1.9 Current level and raise spacing practices narrow reef tabular mines

The Witwatersrand gold mines typically have average dips around 22º while the Bushveld Complex platinum mines typically have average dips around 10º but, both are categorised as inclined narrow tabular reefs. The Witwatersrand gold mines are regarded as being more mature mines or a 'sunset industry' as indicated by Ruffinni (2005), compared to the platinum mines which are still mining in the shallow to medium mining depths. Consequently, most practices on the platinum mines have taken cues from the gold mines. It was therefore appropriate in this study to consider practices on level and raise spacing on the Witwatersrand gold mines as a cue to understanding level and raise spacing practices for the platinum mines. Fleming (2002) undertook a survey on level and raise spacing planning for several Witwatersrand gold mines practising scattered or longwall mining. The summary results are indicated in Table 1.1. It can be seen from Table 1.1 that vertical level spacing ranges between 45m-77m (i.e. backlengths between 99m-280m) and raise spacing ranges between 50m-200m. It can also be noted that the choice of level and raise spacing, which is a key input for planning inclined narrow tabular reefs, is mining method dependent (Vieira, Diering and Durrheim, 2001). Eaton (1934:29) also pointed out the same principle that in laying out underground development, the intervals between levels *"are determined by the size, shape, and position of the orebody and the mining system to be used"*. This is the reason why in the work of Fleming (2002) only the level and raise spacing for mines using scattered mining were considered. In a separate study, Woodhall (2002) reported raise spacing range of 150m – 180m for some Witwatersrand gold mines using scattered breast mining. Ragoonanthun (2003) reported raise lengths of up to 400m at Mponeng gold mine exploiting the Ventersdorp Contact Reef (VCR) on the Witwatersrand Basin. The conclusion that can be drawn here is that level and raise spacing varies from mine to mine even though the same mining method is used.

Table 1.1: Level and raise spacing ranges for Witwatersrand gold mines using scattered or longwall mining
(Fleming, 2002)

STOPING AND DEVELOPMENT LAYOUT

MINE AND DISTRICT		INT.LVL SPACING	DIP	X.CUT SPACING	BACK LENGTH	PANEL LENGTH	NO.OF PANELS	NO.OF BOXHOLE
CEN.RAND								
GR'TVLEI	S	58	13	180	253	30	8	1/CG
WRCONS	S	50/60	15	120	130	9	12	1/CG/SG
E.R.P.M.	L	40	22	60	180/270	45(60)	4/6	4/2
WESTWITS								
DEELKR'L	L	72	25	60	170	34	5	3/2
DOORNF'N	L	91	24	60	224	28	8	3/5
EASTDRIE	L	48	22	60	132	33	4	3/1
WESTDRIE	L	91	22	60	240	30	8	3/5
KLOOF	L	68	30	60	136	27	5	3/2
LIBANON	L	65	38	60	105	35	3	2/1
WESTDEEP	L	50	21	60	240	40	6	5/1
KLERKS'P								
BUFFEL'N	S	70	17	150	240	20	12	8/4
HARTIE'N	S	45	10	150	260	37	7	7
STILFO'N	S	70	30/45	150	99/140	20/25	4/7	2/CG
VAALREEF	S	47	15	180	175	35	5	5/0
O.F.S.								
BEATRIX	S	50	5	100	280	30	9	8/1
PRESBRAN	S	63	27	120	139	35	4	4/0
STHELENA	S							
WESTHOLD	S	58	15	120	224	37	6	3/CG
EVANDER								
BRACKEN	S	71	34	180	127	9	14	1/CG
KINROSS	S	61	15	180	240	24	10	5/CG
LESLIE	S	66	19	200	187	9	22	1/CG
WINKEL'K	S	77	29	175	159	9/18	18/9	1/CG

NOTE : S - SCATTERED MINING
L - LONGWALLS

If focus is turned to the platinum mining sector, a pattern similar to that in Table 1.1 is revealed as follows:

- Impala Platinum, the third largest platinum producer plans its conventional breast mining layouts on standard 300m backlengths (equivalent to 55m vertical level spacing at average reef dip of 10º) and 180m raise spacing (Ackerman and Jameson, 2001). The company has, since 2006, been contemplating changing raise spacing from 180m to 220m while maintaining the 55m vertical level spacing, but no formal study has been done to date to evaluate and quantify the merits of making such a decision (Zindi, 2007).
- At Northam mine, the deepest platinum mine with operations at between 1,250m to 2,160m below surface (Lanham, 2006; Northam, 2009), the conventional breast mining layout grid standard is 150m backlengths to fit about five 30m production panels and 180m raise spacing (Northam, 2009). However, the raise spacing can vary from the standard grid in some parts of the mine in order to locate development

away from water fissures. The geology at Northam is such that mining can sometimes encounter water from fissures that can release water quantities ranging from anywhere between 50,000 litres per day to about 370,000 litres per hour (≈16 mega litres of water per day) and any development that is located in a water fissure area may not be usable due to high water inflows, so development must be sited away from water fissures. (Lanham, 2006; Northam, 2009).

- Technical visits undertaken by the author to some of Anglo Platinum's operating shafts in 2004 indicated that backlengths ranged between 150m-180m and raise spacing ranged between145m-340m.
- Spacing tends to be limited by the scraper winch reach which is typically 90m either side of raiseline when using 56kW strike scrapers and 200m down-dip raise scraping or 100m up-dip winze scraping when using 75kW scrapers. The down-dip scraping reach is higher than the up-dip scraping reach because down-dip scraping is aided by gravity while up-dip scraping. The limit is due to the maximum amount of scraper winch rope that can be wound on a scarper drum.

The above observations show that level and raise spacing for conventional mining varies from company to company and from operation to operation. These observations can be explained through two main reasons. Firstly, it is not possible to have a "one-size-fits-all" level and raise spacing because of the differing degree of geological complexity of each mine operation. Secondly, there has not been any scientifically proven optimal level and raise spacing for conventional mining and mines therefore select spacing based on company policy which derives from empirical knowledge peculiar to the company. In order to fully understand the whole concept surrounding level and raise spacing, the next section addresses the objectives underlying the choice of level and raise spacing followed by a section on the effects of varying level and raise spacing.

1.10 Objectives in planning level and raise spacing

Once level and raise spacing have been selected, medium to long-term mine plans such as 5-year business plans or 30-year Mining Right plans are based on this selection. South African legislation makes the State the custodian of the country's mineral wealth and mining companies are granted mining rights for 30-year renewable periods (MPRDA, 2002), hence the 30-year Mining Right plans. Therefore optimisation of level and raise spacing becomes part of the strategic mine planning process. It is important therefore, that the selected level and raise spacing must satisfy long-term planning objectives which are financial, technical and safety related.

The financial objectives include the need to minimise operating costs (by spacing out development to minimise development cost per centare mined), minimise capital costs, maximise Net Present Value (NPV) and minimise payback period (by having a short build-up period).

The technical objectives in which mining engineers desire to demonstrate their technical excellence include the need to maximise shaft head grade by minimise dilution and selectively mining the orebody (by reducing level and raise spacing); maximise extraction ratio; maximise productivity (by reducing level and raise spacing); minimise build-up period to full production; minimise tailing-off period and maximise replacement ratio or replacement factor (by spacing out levels and raises). For example, development is more expensive than stoping per m^3 of rock mined because blasting in stoping most often has a free breaking face unlike in development ends where the free breaking face has to be created by initially blasting out a cut or utilising a large diameter relief hole. Therefore it is desirable to minimise development which is most often done in waste from which no revenue is derived. For inclined narrow tabular reefs of the Witwatersrand Basin and Bushveld Complex of South Africa, this objective is measured by a decision criterion called the Replacement Factor (RF) or Replacement Ratio (RR). A mining method or layout that gives a higher RF is more desirable. The RF can be maximised by minimising development.

The safety factors include the need to minimise the line of sight to reduce accidents (by reducing level and raise spacing), concentrate production to areas close to each other to improve supervision and minimise unsupported spans to achieve better geotechnical stability (by reducing level and raise spacing). In order to understand the interaction among these objectives when different level and raise spacing are assumed for a conventional mining operation, it is necessary consider the impacts that arise from increasing level and raise spacing.

1.11 Effect of increasing level and raise spacing

When considering increasing level and raise spacing, thousands of permutations of possible layouts can be designed, thus making the process an extremely complex one due to the large number of options that have to be considered. A single point estimate for level and raise spacing is therefore insufficient. Rather an optimal range is more appropriate in such a case. This is why this research study considered an optimal range of level and raise spacing. When level and raise spacing are increased in a conventional mining layout, some of the associated desirable and undesirable impacts that occur concomitantly are outlined below:

- The stope size increases resulting in a decrease in the number of stopes per unit area of the orebody. When the number of stopes per unit area decreases this reduces the number of points of attack for production making it more difficult to relocate production teams if say falls of ground (FOGs) occur in a stope or a stope area becomes unsafe. This is an undesirable outcome. Lawrence (1984) expressed by arguing that when raise spacing is increased the ratio of the distance advanced by straight stoping to the strike distance ledged increases, implying a reduction in the number of blocks (or stopes) to be mined per unit time.
- Operating flexibility decreases as a result of reduced points of attack caused by fewer stopes per unit area. This is an undesirable outcome. Operating flexibility is a decision criterion that is referred to quite often in the narrow reef mining industry but there has not been any consensus on how to measure it and what level of flexibility would be desirable.
- Time to establish a stope is increased because the stope size has increased. In other words longer backlengths take longer to build-up to full production (Vermeulen, 2009). This is an undesirable outcome because it slows the build-up period to full production for a stope.
- Replacement factor (RF) increases since development is now more spaced out per unit area. Lawrence (1984) also noted the same point that there will be a reduction in the amount of development metres per centare of production. This has a financially desirable outcome because the development cost per centare mined decreases (Lewis, 1941; Zambó, 1968; Anglo Platinum MTS, 2005).
- Mining of raises and winzes involves taking out a waste portion below the reef horizon to create adequate storage capacity for ore from production faces, therefore the dilution from raise and winze development ore decreases slightly because raises are more spaced out. This is a desirable outcome because it leads to slightly higher shaft head grades.
- The slight decrease in dilution leads to slightly higher shaft head grade. This is a desirable outcome since it improves revenues.
- By increasing the raise spacing, the strike scraping distance is increased resulting in a decrease in scraper productivity as noted by Brassell (1964) and Lawrence (1984). This is an undesirable outcome because as Lawrence (1984) noted, the decrease in productivity offsets the potential saving in development costs.
- An increase in raise spacing reduces the number of raises which are used for grade sampling of stopes. This is undesirable because a decrease in sampling density from raises compromises the quality of ore reserves estimation which has serious implications at company board level and on shareholders because of reduced confidence in ore reserves estimation.

- When level and raise spacing are increased, the required line of sight for communication purposes is extended. This is undesirable because it compromises safety particularly with 'blind' scraping as it is practised in conventional mining where communication is by 'pull wires' that send bell signals to the scraper winch operator.
- By increasing level and raise spacing, the production stopes are more spread out, making concentrated mining difficult to achieve. When working areas are more spread out, supervision becomes increasingly difficult because supervisors must travel longer distances to visit working places. This is undesirable because as Brassell (1964:461) noted, *"productivity can be improved by increased mechanisation and improved techniques but can best be gained by the concentration of mining activities on the minimum of working face"*. Bullock (2001:17) also highlights the need for concentrated production because, *"any entrepreneur planning a mining operation and who is not familiar with the problems of maintaining high levels of concentrated production at low operating costs per tonne over a prolonged period is likely to experience unexpected disappointments in some years when returns are low (or there are none)"*.
- The larger the stope size, the more cumbersome the logistics because crews have to travel longer distances within a stope. This is undesirable because it increases the risk of losing a blast resulting in lower average monthly face advance rates per crew.
- The life of each stoping connection (i.e. raiseline or stope) increases due to the combination of a decreased rate of face advance and increased spacing (i.e. increased stope size) (Lawrence, 1984). This is desirable because production crews can stay longer per raiseline thus simplifying the logistics of moving crews to new raiselines.

From the discussion above it is evident that the main criteria that must be considered in optimising level and raise spacing include, development cost per centare mined, project Net Present Value (NPV) that captures the interaction of cost savings against loss in productivity as spacing is increased, build-up or ramp-up, Life of Mine (LOM) and project payback period to capture the impact of timing associated with the changes, replacement factor or replacement ratio, dilution, shaft head grade, productivity and production rate. These criteria are consistent with the criteria used by Egerton (2004) to compare different mining methods to mine the UG2 reef, except for extraction ratio which is nearly constant in this research study as explained in Chapters 5 and 6 and operating flexibility which is discussed in Chapter 3. The above outcomes or decision criteria exhibit intricate interdependencies and in some cases outright contradictions. For example, by increasing level and raise spacing, the RF increases at the expense of flexibility which becomes compromised because fewer blocks are now available for mining. Trade-offs must be made among decision criteria in order to arrive at an optimal solution that satisfies all the criteria. The ideal solution should

result in the minimisation of undesirable impacts and maximisation of the desirable ones. Optimising level and raise spacing is therefore a complex multi-criteria decision-analysis (MCDA) exercise in which delicate trade-offs must be made between competing decision criteria depending on the importance attached to each criteria by the decision maker(s). Chapter 2 provides a deeper understanding of why a MCDA approach is more appropriate to solve this optimisation problem by highlighting gaps in past and current work done on optimising level and raise spacing.

1.12 Research question and relevance

From the foregoing sections it can be noted that current narrow reef platinum mines use different selections of level and raise spacing. The degree of geological complexity of each mine, which makes each mine a unique project, is a factor contributing to this variability. Other factors contributing to this variability include the subjectivity of the importance of each decision criteria by each practitioner or group of practitioners. This variability precludes the adoption of a standard level and raise spacing for all Bushveld Complex mines. In other words it is not possible to have a 'one-size-fits-all' level and raise spacing. Therefore it is meaningless to even have a company standard selection of level and raise spacing. Rather a range of level and raise spacing would be more meaningful if the optimal solution can be proved to be fairly stable after a post-optimality sensitivity analysis. This is further confirmed by the fact that previous and current work has not been able to produce a convergent solution. It is therefore a challenge for each new or expansion project to choose an optimal level and raise spacing given that there are legal, economic and technical criteria that must be simultaneously satisfied to derive an optimal spacing. This study was therefore premised on the question that, "Is there an optimal range of level and raise spacing for a given Bushveld Complex platinum reef mine using conventional breast mining considering that current operations using conventional breast mining are planned on different combinations of level and raise spacing?". This research study therefore developed a methodology that can be applied to answer this question by applying it to a typical UG2 reef project code-named OB1. The optimal range derived is therefore applicable to OB1 only and any other project would have to be evaluated separately to obtain an optimal range applicable to it.

The research is quite relevant to the platinum mining industry as indicated by the feedback responses on the research findings (Appendix 10.1). For example one of the feedback comments was that (Impala Platinum Review Team, 2009), *"level spacing and raise line spacing has been a controversial topic in the mining industry for decades. No two mining engineers will agree on this issue as there has been no way to scientifically calculate the best option."* This thesis represents novel research in that for the first time in decades, level and raise spacing have been jointly optimised and the AHP has been used as aid to

optimisation not as a selection tool. After the research findings were presented to the two largest platinum mining companies in South Africa, Impala Platinum and Anglo Platinum, on the 10th and 13th of July 2009 respectively, the research re-kindled interest in the subject matter as evidenced by the invitation to present the findings to the Association of Mine Managers of South Africa (AMMSA) on the 6th of August 2009. A copy the AMMSA programme showing the presentation as an Agenda item is shown in Appendix 10.4. A copy of the presentation will be made available on the AMMSA website soon. The AMMSA website is: http://www.ammsa.org.za. After the presentation was made, AMMSA requested to have the presentation compiled as a technical paper for inclusion into their annual Proceedings.

1.13 Structure of thesis

This thesis is divided into eight chapters, followed by reference and appendix sections. A USB memory stick with the Mine 2-4D designs and EPS schedules, is also included with the thesis. This introductory chapter provides background information pertaining to the historical, techno-economic and strategic contexts within which the research question is addressed, and the relevance of the research question. The chapter concludes that the problem of determining an optimal range of level and raise spacing is a multi-criteria decision analysis (MCDA) problem and should solved using an MCDA technique.

Chapter 2 is devoted to an analysis of previous and current methods on optimising level and raise spacing for inclined narrow reef deposits. The chapter notes that these methods have not sufficiently addressed the issue of level and raise spacing optimisation, thus requiring a new approach to solving the problem. The new approach is an MCDA technique called the Analytic Hierarchy Process (AHP) methodology and the justification for its selection is provided in the chapter.

Chapter 3 gives an appreciation of the concept of technical operating flexibility and its relevance as one of the key optimisation decision criteria in the study. The fourth chapter describes the OB1 geological model which was used for testing the behaviour of decision criteria under variable level and raise spacing. Chapter 5 gives a treatment of design, scheduling and economic assumptions applicable in designing the layouts under variable level and raise spacing, at a pre-feasibility level of study at which this research study was undertaken.

Results of design and scheduling, and financial valuation of the layouts are reported and analysed in Chapter 6. In Chapter 7 the AHP methodology is discussed and the summary decision criteria results for each layout analysed in Chapter 6 are integrated to determine the

optimal range of level and raise spacing. A sensitivity analysis is subsequently done to establish the stability of the solution obtained. Conclusions and recommendations are made in Chapter 8.

2 REVIEW OF LEVEL AND RAISE SPACING OPTIMISATION AND MULTI-CRITERIA DECISION ANALYSIS (MCDA) TECHNIQUES

2.1 Introduction

The current chapter presents the findings of a literature search conducted to trace the progression of techniques applied to the problem of optimising level and raise spacing for inclined narrow reefs or veins. The gaps in the techniques applied or solutions obtained from these previous and current optimisation methods are discussed to justify the need for a more holistic multi-criteria decision analysis (MCDA) technique to solve the problem. MCDA techniques are subsequently reviewed and from the discussion it emerges that the Analytic Hierarchy Process (AHP) is best suited to solve the problem of optimising level and raise spacing for the shallow-dipping narrow tabular platinum reefs of the Bushveld Complex. The AHP methodology is then discussed in more detail. In this chapter the terms 'drift' and 'drive' maybe be used interchangeably to mean the same thing, and so are the terms 'spacing' and 'interval' and the terms 'raise', raiseline, 'raise connection' and 'connection'.

2.2 Review of previous and current work

Due to the complex nature of the problem of level and raise spacing optimisation for inclined narrow reef or vein deposits, the subject has received intermittent attention over the years, with generally inconclusive solutions being derived. This could be the reason why only a few directly relevant previous and current references on the subject matter were identified. These are in chronological order, Eaton (1934), Lewis (1941), Brassell (1964), Zambó (1968), Lawrence (1984) and Anglo Platinum MTS (2005). Carter, Lee and Baarsma (2004) highlight the same viewpoint in their argument that the design and planning engineer for underground metalliferrous mines has had to rely on experience and a limited number of design heuristics in order to optimise underground mine plans because unlike open-pit mine designs, the underground mine design optimisation problem has numerous permutations of mining layout alternatives. Other work which addresses the planning of location and timing of development in general, is also discussed. The few previous and current methods on optimising level and raise spacing for inclined reefs or vein deposits are individually addressed in the next sub-sections.

2.2.1 Level spacing optimisation by Eaton (1934)

One of the early researchers to pay attention to the subject of optimisation of level spacing was Eaton (1934). Eaton (1934:29) argued that in laying out underground development, the intervals between levels *"are determined by the size, shape, and position of the orebody and*

the mining system to be used". From an economic point of view, Eaton (1934:29) argued that a level should be opened at an elevation such that *"sufficient ore is above it to justify the development"*. Eaton (1934:29) further argued that in order to *"keep down the cost per ton for development and level equipment, the interval between levels is made as large as is compatible with convenience, safety, and economy in mining"*. It follows therefore that the higher the cost of excavating and maintaining a level, then the greater the level spacing that must be made. The current focus by mine planners in advocating longer backlenghts in the design of inclined narrow tabular reef mines concurs with this argument.

Eaton (1934) further argued that as the level spacing is increased, a point is reached where the saving per tonne of ore mined is more than offset by the cost of mining at longer distances. This argument can be deduced from Brassell (1963), who carried out extensive on-mine test work on improving stoping efficiencies for a narrow, tabular reef gold mine and observed that mining at longer distances reduces productivity asymptotically, thus increasing the cost of mining at longer distances. From this perspective, Eaton (1934) was implicitly acknowledging that other factors other than the cost per tonne do affect the decision to select an optimal level spacing. Although Eaton (1934) did not clearly show quantitatively how he arrived at optimal level spacing, he estimated the economic limit on level spacing to be 100ft-200ft (≈30m-60m) based on the mining practices on mines at that time. This range of values is consistent with current practices as highlighted earlier in Section 1.9. Two examples of mines that exceeded the limit of 200ft for level spacing had level intervals of 300ft (≈90m) and 600ft (≈180m), respectively. The major reason for the departure from the norm was that the shaft was a long distance away from the orebody on the first mine with level spacing of 600ft while the second mine was using a caving method where 300ft was the most geo-technically optimal limit for level spacing.

Eaton (1934) gave another hypothetical example of a mine where the orebodies are small and scattered, thus placing a demand for a large amount of haulage excavation to be done for a small tonnage throughput per level. In this example, the temptation is therefore to increase the level interval. This temptation however, is at the expense of the exploratory value of development as discussed earlier in Chapter 1. For example, with increased level spacing it becomes more difficult to find the downward extension of the orebodies intersected on upper levels since veins can rapidly thin out and terminate. Additionally, with increased level spacing, it becomes more probable that some small veins might be missed as the mine is extended further down. Therefore, Eaton's (1934) work highlights the importance of the role that geology plays in optimising level and raise spacing, because the more complex geology requires that levels and raises be spaced closer together resulting in higher the development cost per centare mined.

Some criticisms are worth noting on Eaton's (1934) work. Although Eaton (1934) gives a compelling qualitative argument, the work does not provide a quantitative treatment on how the economic limit of 100ft-200ft (≈30m-60m) for level spacing was derived. The argument is also silent about the effect of the timing of the development costs, yet the timing of development changes once level spacing has been changed. Lastly, the approach considers the economic factor as the overriding factor (i.e. the cost per tonne ore mined), yet the problem is actually a multi-criteria decision analysis optimisation problem as will be discussed in later sections of this chapter.

2.2.2 Optimisation of level spacing by Lewis (1941)

Lewis (1941) approached the optimal spacing of levels as an exercise to minimise the sum of excavation and haulage costs. When these two sets of costs are charged to a tonne of ore mined, the optimal level spacing is the one giving *"the least cost per ton of ore mined for the method of mining chosen"* (Lewis 1941:416). The excavation and haulage costs considered in the exercise were separated into two categories. The first category was made up of the cost of shaft-sinking and equipping from one level to the next, costs of drifts (i.e. haulages) and crosscuts to access the ore and cost of level equipment such as fans, tracks, air and water lines and power lines. The second category was made up of costs of raises and ore passes needed; costs of maintaining the drifts, crosscuts and raises over the life of the level; costs of hoisting ore to surface; costs of pumping; costs of ventilation, waste filling supervision, and interest that could have been earned on capital spent on developing stopes that are ready for mining but not being mined. It can be interpreted that by factoring in interest, Lewis (1941) was actually accounting for the timing of development costs.

The cost analysis was done for a 4ft (≈1.2m) continuous thick vein of scheelite with an average dip of about 60°, serviced by an inclined shaft dipping at 75° in the same direction as the orebody, so that the bottom level was at 500ft (≈150m) below the collar of the shaft and the distance from the shaft to the orebody at that level was 1,200ft (≈360m). As expected, the first category costs decreased as the level spacing was increased and the second category costs increased in proportion to the distance between levels. The overall cost per tonne of ore mined (i.e. both category one and two costs) showed an asymptotic decrease with increasing level spacing, that followed a power function (Figure 2.1). The study underscored the economic motivation for the largest possible level spacing as the overall cost per tonne asymptotically decreased with increasing level spacing. Thus, Eaton's (1934) suggestion for longer level spacing had once again been confirmed. This finding remains true today for mine planning on narrow reef tabular platinum and gold mines of the Bushveld Complex and Witwatersrand Basin, respectively. There is a constant striving to increase backlenghts for conventional mining layouts, with backlenghts of close to 400m

being reported by Ragoonanthun (2003) for Mponeng gold mine, where excavations called 'slushers' are developed underneath and parallel to the raiseline to provide adequate ore storage capacity.

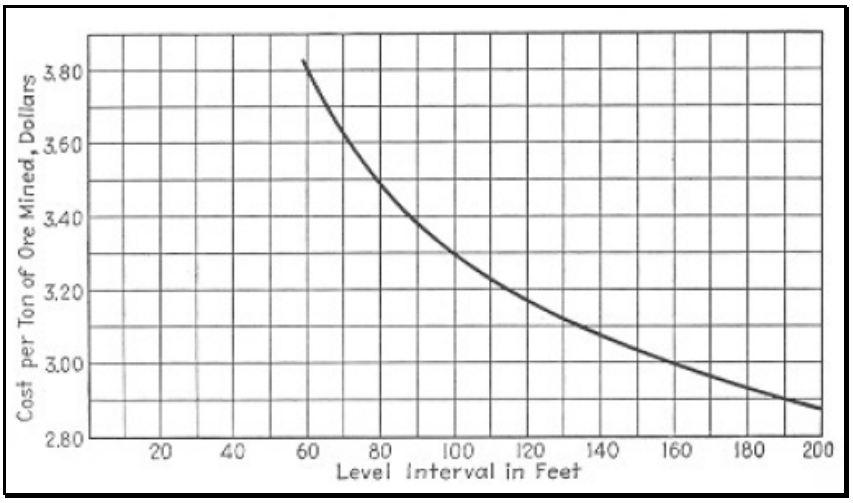


Figure 2.1: Variation of cost per tonne ore mined with increasing level interval or spacing
(Lewis, 1941)

Some criticisms can be drawn on the work by Lewis (1941). Firstly, the work does not consider the impact of geological factors such as spatial grade variations and loss of mining areas due to geological discontinuities. These are important as they impact on the net contribution in value from a development working. Secondly, the work is inconclusive on what would be an optimal level spacing for the scheelite vein deposit. Rather, Lewis (1941:417) concludes that:

> *"In the final analysis, the above comparative costs must be weighed against other factors, such as the relative advantages of various level intervals for prospecting, the time required to open the level before stoping can be started, the life of the level, and the structural features of the ore body and its environment, since these determine the method of mining and thus indirectly the distance between levels"*.

In this comment Lewis (1941) was in fact acknowledging that the problem of optimal level spacing in inclined reefs or veins is a multi-dimensional problem yet he had solved it as a mono-criterion decision problem of minimising the cost per tonne of ore mined. For example, when the level spacing is increased, the cost per tonne mined decreases, but the backlength increases, reducing the cleaning efficiency or productivity in the production workings. Brassell (1964) noted that productivity decreases asymptotically with increasing raise spacing or backlength, following a power function. Consequently, by treating the problem as a mono-criterion decision analysis problem, Lewis (1941) failed to find a convergent solution because as Figure 2.1indicates, the cost per tonne decreases *ad infinitum* with increasing level interval.

It is also worth noting that the train of thought followed by Lewis (1941) concurs with contemporary optimisation models in planning level spacing. For example, optimising the planning of level spacing on some platinum mines is currently guided by the *Half-Level Optimisation Model* concept (Ballington, *et al*, 2005), which assumes that the development cost per centare mined decreases asymptotically with increasing level interval. It should also be noted that the work of Lewis (1941) ignores Eaton's (1934) argument that as the level spacing is increased, a point is reached where the saving per tonne of ore mined is more than offset by the cost of mining at longer distances; therefore the cost per tonne cannot continue decreasing indefinitely without being countered by other negative effects.

2.2.3 Scraper winch productivity and raise spacing by Brassell (1964)

Brassell (1964) carried out extensive stope productivity improvement field trials and related time studies at the then Vaal Reefs Exploration and Mining Company over a period of six years. The mine was a gold mine using conventional breast mining. Two of the several trials conducted are of relevance to this research study. One of the trials was on panel face length variation and its corresponding effect on cleaning time using a 30hp (≈25kW) scraper winch. In a 7-hour cleaning shift the effective cleaning time was about 3½ hours to 4 hours. The trial indicated that at typical slow-speed scraping, the optimum face length that could be cleaned in a single shift ranged between 100ft (≈30m) to 120ft (≈36m). This finding concurs with current conventional breast mining operations on narrow reefs as indicated in Section 1.7.2 that panel lengths are in the range 25m-40m. The second relevant trial was on variation of advanced strike gulley (ASG) length as a panel face advanced away from the raise position. A 50hp (≈37kW) scraper winch was used for strike gulley scraping. The results from this trial were tabulated by Brassell (1964) but are presented here in a graphical from (Figure 2.2).

These findings indicate that scraper productivity decreases asymptotically with increasing ASG length (i.e. increasing raise spacing), following a power function. Brassell (1964) also noted that the breast stoping layout that evolved as a result of these trials, laid out raises 500ft-600ft (≈150m-180m) apart on strike, a raise spacing which is currently used on the Witwatersrand gold mines and Bushveld Complex platinum mines as indicated earlier in Section 1.9.

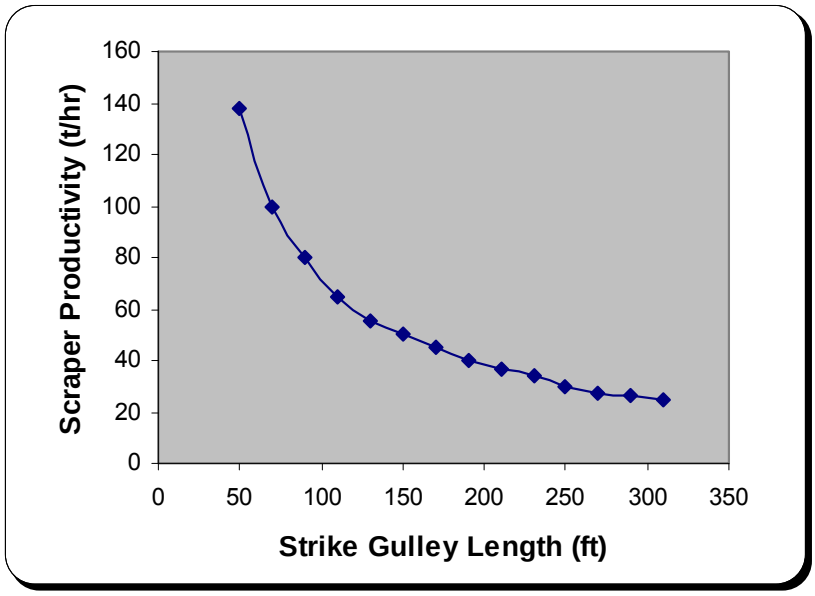


Figure 2.2: Variation of scraper winch productivity with increasing strike gulley length on a panel
(Adapted from Brassell, 1964)

However, the results from the Brassell (1964) study need to be understood in the context of present day conventional breast mining by noting that:

- Brassell's (1964) paper deals with a single panel in a raise connection. However in current practice there could be up to five stoping crews in a single raise connection blasting up to five panels a day resulting in the productivity being dependent also on the capacity of the centre gulley scraper winch to clean all the ore from the ASGs.
- Productivity will also be affected by the distance of the face scraper winches from the panel faces. Typically face scraper winches are 'leap-frogged' regularly so that they are not more than 30m away from the panel face.
- Productivity will also depend on the configuration of the scraper winch sizes in use as indicated in Sections 1.7.2 and 1.7.3.
- The stope boxholes are cleaned by loco-and-hopper tramming systems on each level, which is a batch transportation system and therefore can reduce the cleaning capacity of the centre gulley winch.
- Productivity will also depend on how the full mining cycle for the raise connection is arranged. Poor shift arrangement and supervision negatively affect productivity even if scraping is being done at short scraping distances.
- The productivity will also depend on the frequency of lost blasts which dictate the balance between how much ore will be available per cleaning shift against how much the scraper can move in a shift. For example Jiyana (2009) reported that the

average lost blast frequency at Turffontein shaft is currently about 29%, caused by a combination of factors ultimately affecting stope productivity.

- The productivity will also depend on the geo-technical stability of the panels in the raise connection. Geo-technically poor ground conditions negatively affect productivity because significant shift time can end up being used for stope support thereby compromising the stope cleaning capacity and productivity.

Brassell's (1964) work has some implications on optimising level and raise spacing. Firstly, scraper productivity decreases with increasing scraping distance (i.e. with raise spacing) and so does stope panel advance. Similarly, increased level spacing directly leads to longer backlengths resulting in more panels per raise connection and longer centre gulley scraping distances thus, decreasing the centre gulley scraping productivity and panel face advance.

2.2.4 Optimisation of level and raise spacing by Zambó (1968)

During the 1960s, Zambó (1968) analysed the problem of optimising the location of a shaft in both plan view and section (discussed further in Chapter 5), and optimising level interval and panel strike length (i.e. raise spacing) for tabular, gently-dipping vein deposits. In all cases Zambó (1968) used graphical and mathematical procedures to illustrate how to make the optimal selection by simultaneously minimising excavation and haulage costs. The work was originally written in Hungarian in 1966 but was later translated into English in 1968. Zambó (1968) used the Hungarian monetary unit, the Forint (F) for all cost calculations. In his work, Zambó (1968:126) argued in a similar manner to Eaton (1934) and Lewis (1941) in that:

> *"The greater the level interval, the less the specific investment expenditure, the smaller the number of levels to be kept open simultaneously, the more fully the hoist of the shaft can be exploited, and the less the mineral reserve to be tied down eventually in the pillars of the haulageways of the levels. Conversely, the less the level interval, the less the specific cost of displacing personnel, timber and supplies at large and between two levels in particular, the simpler the driving of raises and winzes… . Of the possible level intervals, that one will be considered an optimum here which makes the specific production cost of the mine a minimum"*.

By specific production cost, Zambó (1968) was referring to the cost per tonne of ore mined. In deriving the optimum level interval, Zambó (1968) made several simplifying assumptions. Firstly, the investment expenditure (i.e. development cost), K_A, on a level includes the cost of driving and equipping the permanent facilities of the level such as the shaft stations, haulage tracks and pipes. K_A is related to the production capacity, q, of the level through the relationship given by Equation 2.1.

$$K_A = aq^{\mu}$$

Equation 2.1

Data on annual output, q, and total investment expenditure, K_A per level were obtained from mines operating under similar geological and mining conditions and compiled as shown in Figure 2.3. The constants, a and μ in Equation 2.1 were then derived by regression analysis. Typically μ must lie in the interval, 0 to +1, if the power function depicted in Figure 2.3 is to remain valid.

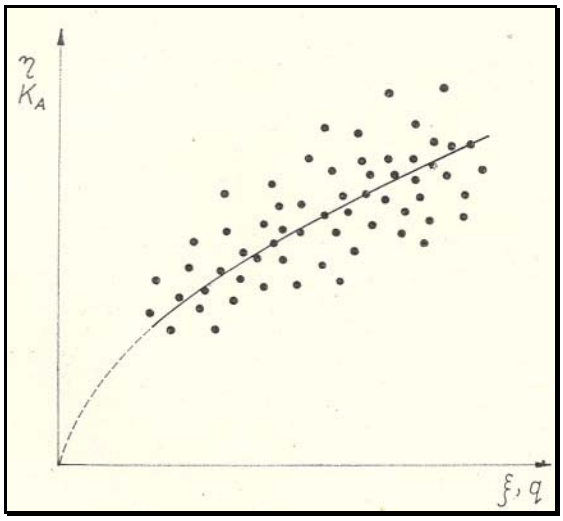


Figure 2.3: Typical plot of K_A and q values for mines operating under similar conditions
(Zambó, 1968)

Zambó (1968) further assumed that the investment expenditure, K_A, could also be expressed as a function of level interval, h, considering the logic that the further apart the levels are, then the higher the investment required per level. Based on this assumption, the specific investment cost function took the form indicated by Equation 2.2.

$$KA = \frac{H}{Q} a h^{\mu-1}$$

Equation 2.2

where H is the maximum economic depth to which mining will occur, measured along reef dip and projected onto the vertical plane; Q is the total workable mineral reserve per unit area in the plane of the reef and projected onto the vertical plane and, a and μ are derived from the regression analysis of Equation 2.1.

For the haulage cost function, K_B, Zambó (1968) assumed that it was related to the average transportation distance, L, on a level through the relationship indicated in Equation 2.3.

$$K_B = b q^{\nu} L^{\omega}$$

Equation 2.3

Again, data from mines operating under similar mining and geological conditions were used to carry out a regression analysis to determine the constants *b*, *v* and *ω*. The constant *v* must lie between +1 and +2 for Equation 2.3 to be valid. Typically, the initial investment in the haulage system would be written-off annually using pre-determined percentages, until it is completely redeemed at zero interest rate. The annual writing-off of the investment expenditure is captured in the haulage cost function through a constant *c*. If the investment expenditure is written-off using an annual percentage rate denoted by ϑ_i, then the constant *c* is defined by Equation 2.4.

$$c = \frac{\vartheta_{i+1}}{\vartheta_i}$$

Equation 2.4

When the initial investment is written-off in equal annual repayments then *c = 1*, and Zambó (1968) called this '*uniform amortization'*. When the initial investment is written-off using decreasing percentages applied to the initial sum, then *c < 1* and Zambó (1968) referred to this approach as '*digressive amortization'*. When the initial investment is written-off using increasing percentages applied to the initial sum, then *c > 1* and Zambó (1968) referred to this approach as '*progressive amortization'*. These approaches are equivalent to straight line depreciation and accelerated depreciation methods in contemporary economic and financial valuation terminology.

Zambó (1968) further assumed that the haulage cost function, K_B, could also be logically expressed as a function of level interval, *h*, as indicated in Equation 2.5.

$$KB = \frac{b}{c} h^{v-1}$$

Equation 2.5

where *b* is the constant derived from regression analysis of Equation 2.3 and *c* is the constant derived from Equation 2.4.

The total specific cost function, *K*, is the sum of K_A and K_B as given by Equation 2.6.

$$K = \frac{H}{Q} a h^{\mu-1} + \frac{b}{c} h^{v-1}$$

Equation 2.6

The classical optimisation approach to obtain the minimum specific cost requires as a first step, taking derivatives of Equation 2.6 with respect to *h* and setting them equal to zero to obtain Equation 2.7.

$$(1-\mu)\frac{H}{Q}ah^{\mu-2} = (\nu-1)\frac{b}{c}h^{\nu-2}$$

Equation 2.7

The relationship in Equation 2.7 permits the expression of the optimum level interval, *h*, as given by Equation 2.8:

$$h = \left[\frac{(1-\mu)Hac}{(\nu-1)bQ}\right]^{\frac{1}{\nu-\mu}}$$

Equation 2.8

It must be noted that Equation 2.8 is applicable when interest on investment is ignored or assumed to be zero. However, when interest is factored into the investment and the investment subsequently amortised, then the formula is modified and the optimum level interval obtained, h_r, applies to a case of amortisation with interest. It can be further noted that h_r is always greater than *h* because interest increases the value of *K*. Zambó (1968) then applied the above procedure for determining the optimal level interval for a hypothetical mine using typical industry data at that time and obtained the results shown by Figure 2.4.

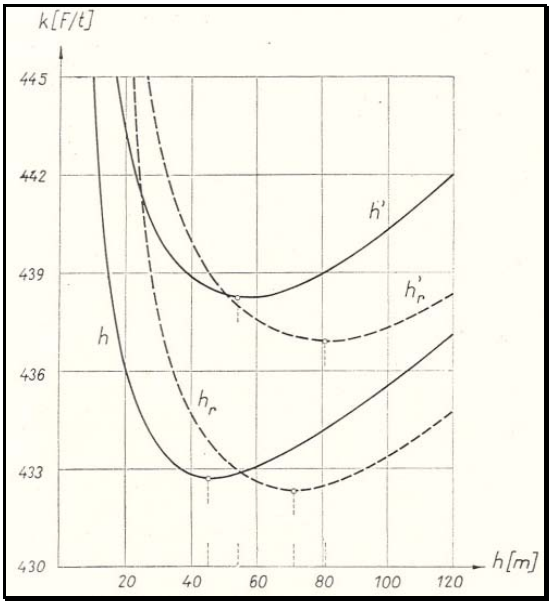


Figure 2.4: Variation of the specific cost function, *K*, with level interval, *h*
(Zambó, 1968)

Figure 2.4 shows the optimum level interval under different amortisation conditions; *h* (=45m) is the optimal level interval when the specific cost is not amortised; h_r (=54m) is the optimal level interval when the specific cost is amortised at an interest rate of 5%; h' (=71m) is the optimal level interval with uniform amortisation without interest and h'_r (=81m) is the optimal level interval with uniform amortisation at 5% interest rate. By considering *h* and h_r it can be

seen that a 5% change in interest rate results in a 20% change in level interval, which is quite significant. Similarly, a 15% change in optimal level interval is obtained when h' and h'_r are considered. Thus, the choice of the interest rate, or project discount rate as done in Chapter 5, is important, and should be done as carefully and realistically as possible to avoid erring on the choice of optimal level interval. Hajdasiński (1995) also emphasised the importance of careful and realistic selection of the interest rate when optimising the location of mining facilities. These findings concur with Eaton's (1934) argument that as the level interval is increased, a point is reached where the saving per tonne of ore mined is more than offset by the cost of mining at longer distance and the overall cost per tonne starts rising again, because other factors, such as the associated decline in productivity, negate the cost benefits derived from wider spacing of levels. It is also worth noting that the cost per tonne varies with increasing level interval following a power function.

Zambó (1968) similarly analysed the optimum strike length of a panel (i.e. optimum raise spacing) and obtained the results as shown in Figure 2.5 indicating an optimum raise spacing of 0.46km for the typical industry data prevailing at that time. This distance is in close agreement with the practical limit for level (or backlength) and raise spacing as noted earlier in Sections 1.8 and 1.9 and later in Chapter 5 where the practical level and raise spacing limits of 400m are used when designing layouts for the OB1 case study. Again, the cost per tonne varies with increasing raise spacing following a power function (Figure 2.5).

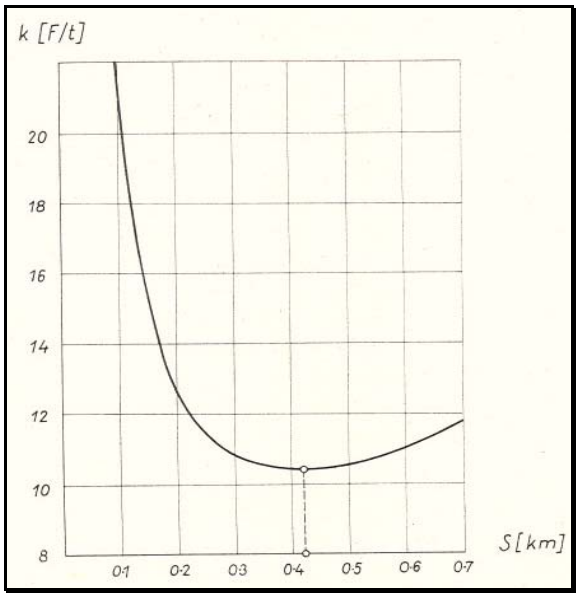


Figure 2.5: Variation of specific cost function, *K*, with crosscut panel length, *S*
(Zambó, 1968)

One of the conclusions made by Zambó (1968:134) is that the optimal level interval that was derived best served as a guide only, *"indicating that value of h in the vicinity of which a more*

detailed examination of the cost function may be worthwhile". The work also concluded that the specific cost function for both level interval and raise spacing *"varies rather slowly in the vicinity of the optimum point, while its rate of change increases quite rapidly with growing distance from the optimum",* thus proving that *"it is not worthwhile to aim at an exaggerated accuracy in optimum computations"* (Zambó, 1968:144). If this finding is interpreted within the context of this study, it implies that deriving a precise optimal level and raise spacing might be an exaggerated degree of accuracy, but rather a range of optimal level and raise spacing may be more appropriate.

There are some criticisms to Zambó's (1968) work. Firstly, as Lizotte and Elbrond (1985) noted, Zambó's procedures did not provide solutions to the generalized problem and required the analyst to "visually" eliminate certain possibilities to get to the final solution. Secondly, Zambó (1968) did not jointly optimise level interval and panel strike length, yet the spacing selection of one will directly impact the spacing selection of the other, thus the solution were sub-optimal solutions. Lastly, the approach structured the problem as a mono-criterion optimisation problem based on cost per tonne alone, yet the optimisation problem is in fact a multi-criteria optimisation problem.

2.2.5 Optimisation of raise spacing by Lawrence (1984)

Lawrence (1984) developed a computerised method to calculate an economic optimum spacing of raise connections (i.e. raise spacing) in conventional 'scattered' (i.e. breast) mining layouts for shallow-dipping, narrow tabular gold reefs. In the computation, Lawrence (1984) only considered the cost saving associated with changing raise spacing as the key determinant in comparing different raise spacing. The savings were then converted to present value (PV) terms using opportunity interest rates between 3% and 7% applicable at that time, in order to draw up a meaningful comparison since changes in raise spacing affect timing of the development costs. The PV of cost savings were further annualised to give an equivalent annual cost saving by dividing with the annual tonnes or centares mined and reported in R/t or R/ca, respectively. The calculation procedure or method was programmed using the programming language, BASIC, and run on an HP9845 desktop computer.

A number of key assumptions were made in setting up the model. Firstly, Lawrence (1984) assumed that strike scraping productivity decreased with increasing raise spacing as shown in Figure 2.6. The loss in productivity would offset potential development cost savings arising from the reduced number of raise connections. In making this assumption, Lawrence (1984) also referred to the work of Brassell (1964). This figure shows that, productivity as an optimisation criterion varies non-linearly with increasing raise spacing, following a convex power function, such as a quadratic function.

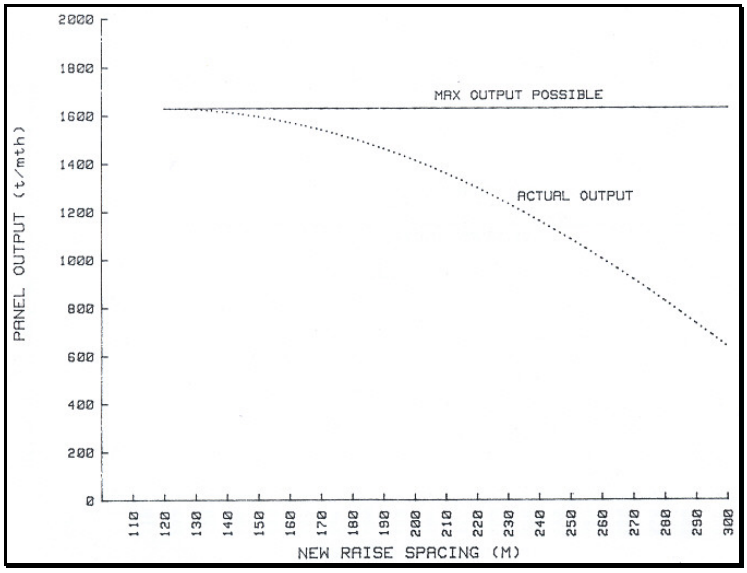


Figure 2.6: Convex quadratic relationship between panel productivity with raiseline spacing
(Lawrence, 1984)

However, if the trend shown in Figure 2.6 is extrapolated, the result will be as shown in Figure 2.7 suggesting that beyond 350m raiseline spacing, negative productivity will be obtained. Negative productivity is unrealistic because in the worst case productivity can only be zero. This could be a possible flaw in Lawrence's (1984) model because the expected relationship from the work by Lewis (1941) and Zambó (1968), suggests a concave inverse relationship, not a convex relationship as perceived by Lawrence (1984). Most probably, Lawrence (1984) considered the reality that for the Witwatersrand deep level gold mines, when raise spacing is beyond 350m, gulley closure is experienced resulting in more waste tonnes from hangingwall and footwall closure being extracted than ore tonnes in order to keep the stopes open (i.e. mining more waste than ore resulting in a negative net ore tonnes). This explanation is inferred from one of the main assumptions that, *"the closure in the centre and strike gullies follows the elastic theory of convergence. A closure exceeding the tolerated closure is costed in terms of an increase in the size of the gullies"* (Lawrence, 1984:11).

A second major assumption was that ventilation requirements would change because altering raise spacing will mean that the ventilation network is altered also. This is the reason why ventilation planning had to be done in Chapter 5. Thirdly, the range of raise spacing considered was 120m-300m, the upper limit being dictated by the strike scraping capacity of 56kW strike scraper winches in use at that time. The lower limit could probably have been due to the fact that when the raises are too close, the working areas become too congested and this ultimately complicates the production logistics.

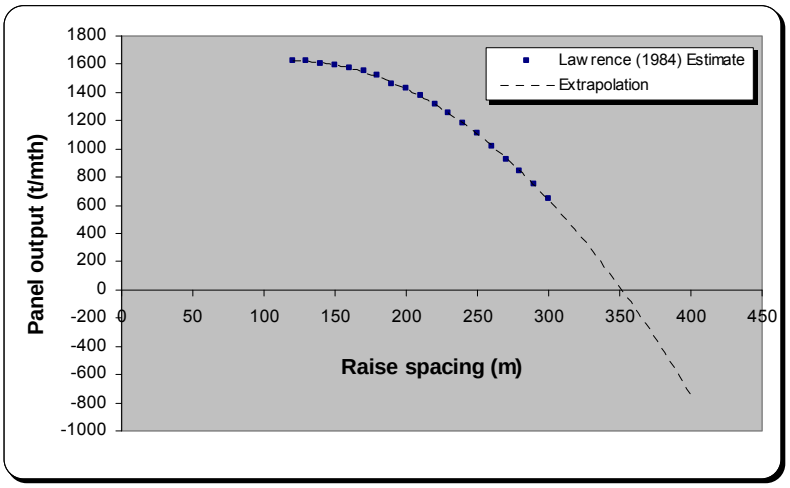


Figure 2.7: Extrapolation of the Lawrence (1984) model at longer raiseline spacing

Lawrence (1984) then applied the model on a scattered mining layout for a hypothetical gold mine, using typical industry data prevailing at that time. The mine's production rate was 80,000ca/month at an average stoping width of 1.3m, located some 2,300m below surface and with an average reef dip of 23º. The initial layout had raises spaced at 150m along strike and the backlength was kept constant at 180m (i.e. vertical level interval was fixed at about 70m); crosscuts were assumed to be 170m long; travelling ways were assumed to be 30m long; and each raise connection had four boxholes with a total length of 120m. Lawrence (1984) also assumed an initial stope face advance rate of 15m/month at 150m raise spacing. The stope face advance rate was assumed to decrease in proportion to the square of the distance in excess of the 150m. All cost calculations were based on 1982 cost figures. The results of the analysis of PV cost savings are shown in Figure 2.8.

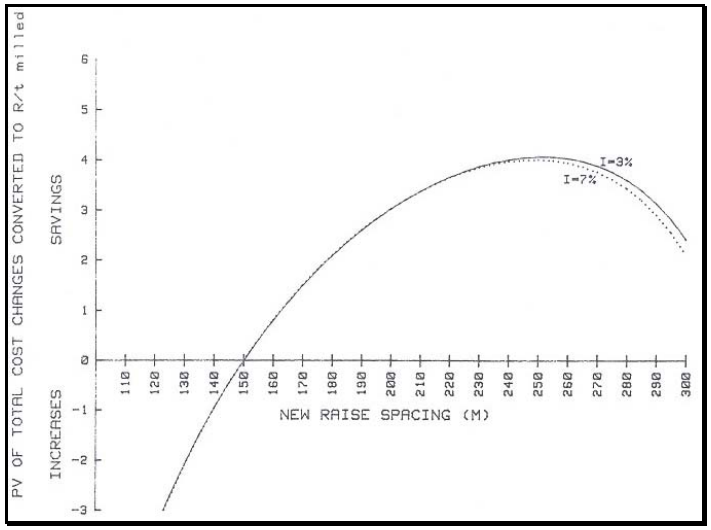


Figure 2.8: Variation of PV of cost savings with increasing raise spacing
(Lawrence, 1984)

The dotted curve in Figure 2.8 is for an interest rate of 7% while the curve with a solid line is for interest rate of 3%. Figure 2.8 indicates that the relationship between the annualised PV of cost savings and raise spacing follows a quadratic function. It is also discernible from Figure 2.8 that *"the economic optimum raise spacing would be either approximately 240m or 250m…. For the purpose of this example, the economic optimum spacing is taken as 245m."* (Lawrence, 1984:13). Strangely, this optimum has to date not been officially adopted by the Witwatersrand gold mines as seen earlier in Section 1.9 that the gold mines are using a range of raise spacing for their conventional mining layouts. The fact that the optimal raise spacing of 245m has not been widely adopted by industry raises the question of its validity as an optimal spacing. Therefore the following opinions and criticisms on Lawrence's (1984) work are worth noting when interpreting the derived optimal raise spacing:

- The main constraint limiting raise spacing that was noted by Lawrence (1984) and still remains true today was that, *"the most influential factor involved in the determination of the economic optimum raise spacing is the system used for strike tramming"* (Lawrence, 1984:17). The same sentiment is expressed by Woodhall (2002). The strike tramming system used when the study was undertaken was scraper cleaning, which is still used to the present day in conventional breast mining.
- Lawrence's (1984) work was based on varying raise spacing for a fixed level spacing that was equivalent to a backlength of 180m, yet the mines practising scattered mining (or scattered breast mining) use different level spacing as noted earlier in Section 1.9. Therefore a 'one-size-fits-all' solution as derived by Lawrence (1984) is in adequate under such circumstances, unless a mine is planned on the same fixed backlength of 180m used by Lawrence (1984).
- For a fixed vertical level spacing, the backlength varies from one raise to the next due to the variable dip and surface terrain of the reef horizon caused by geological variations from point to point over the entire orebody. The fact of variable geology was noted by Schoor and Vogt (2004) as mentioned earlier in Section 1.5. In Chapter 6 it is noted that the backlengths for OB1 were variable for each layout although the vertical level spacing had been fixed for each layout. Therefore it is incorrect to assume a fixed backlength as was done by Lawrence (1984).
- Lawrence's (1984) model did not incorporate geological variations because it assumed a constant geology. This is highlighted in one of Lawrence's (1984) main assumptions that, *"the layout is not affected by geological conditions such as faults and dykes or by areas of low payability"* (Lawrence, 1984:11). This assumption runs counter to the important fact raised by Eaton (1934) that geology cannot be ignored or assumed to be constant throughout the orebody when carrying out level and raise spacing optimisation. Lawrence (1984) also noted this weakness in his model by saying that, *"in a real situation, the ground would be divided into irregular blocks,*

each with a different strike width" (Lawrence, 1984:16), therefore requiring that *"each block would be treated separately but in the same way as described above to give local optimum (economic and practical) spacings for raise connections"* (Lawrence, 1984:16). Therefore, it is more appropriate to include geology when optimising level and raise spacing in conventional mining layouts.

- In addition, the method by Lawrence (1984) did not jointly optimise level and raise spacing, yet these two are the basis for defining the boundaries of a mining block, which is the smallest production unit in a mine that is replicated to produce the mining pattern or method. Therefore, this research study jointly optimised the level and raise spacing.
- The model assumes that raise spacing is a mono-criterion optimisation problem based on economics alone because, *"the economic optimum spacing is that at which the overall savings are at a maximum"* (Lawrence, 1984:11). This is inadequate because as was discussed earlier on, optimisation of level and raise spacing is a multi-criteria optimisation problem.

2.2.6 Anglo Platinum MTS (2005) Half-Level Optimisation Model

When Anglo Platinum was formed through the unbundling of JCI, it acquired other PGM assets that were not part of the JCI group and in the process ended up with mines that had different standard operating procedures and mine planning guidelines (Rogers, 2005). In order to standardise the operations, the company has over the years developed the *Group Guideline: Mine Technical Services* manual for reference by individual mines. Part of the guideline addresses optimising backlength (i.e. optimising level spacing) for conventional breast mining and this is done through the *Half-Level Optimisation Model*. The Half-Level Optimisation Model assumes that the primary drivers of a half-level output are the average panel advance per month multiplied by the backlength equivalent of the sum of the panel lengths, less the sum of geological and mining losses. This output is then adjusted using secondary drivers (or constraints to production) to achieve an optimum backlength, that include ventilation constraints, geo-technical constraints that may limit pre-development, availability of services such as power, compressed air and water, and state of logistics such as rock handling capacity, men, material and equipment transportation, and capacity of shaft infrastructure such as tips and stations. As such the Half-Level Optimisation Model appears to be configured as a Linear Programming (LP) model, with backlength maximisation as the objective function and the production constraints as the LP constraints. Typical output information from the model is illustrated by Figure 2.9 and Figure 2.10 which provide justification for why longer backlengths are preferred in designing conventional breast mining. In fact, one of the sections in the *Group Guideline: Mine Technical Services* manual is, "Why longer backs and development focus".

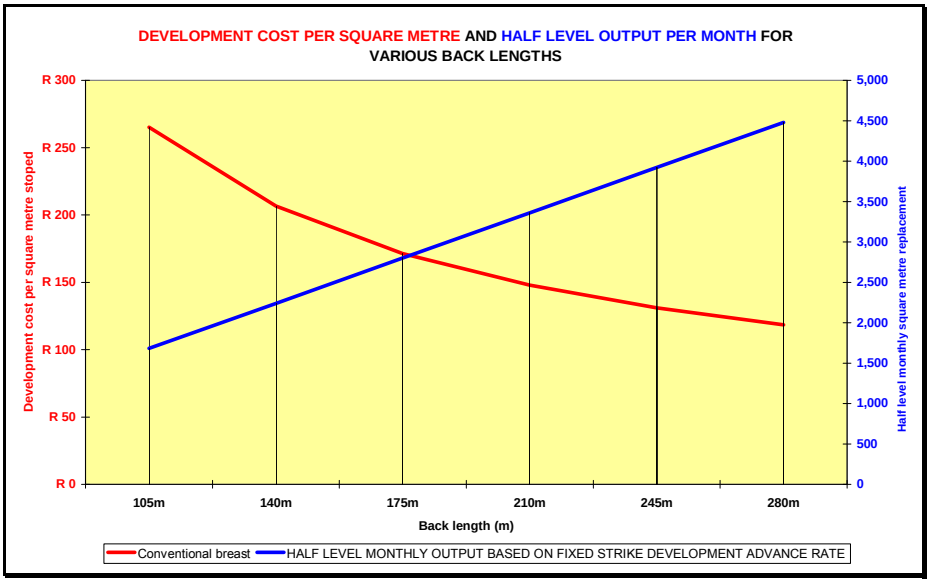


Figure 2.9: Typical Half-Level Model Output
(Anglo Platinum MTS, 2005)

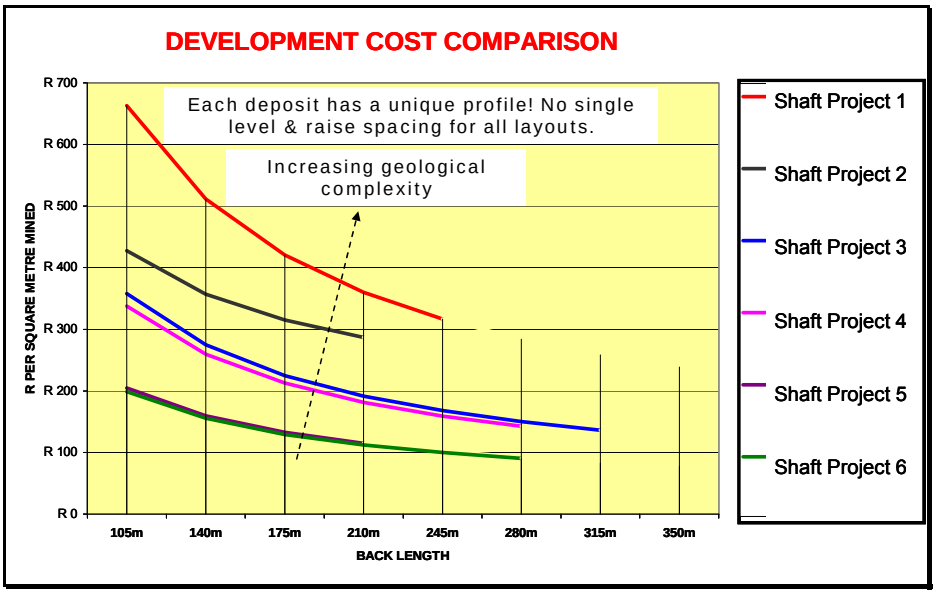


Figure 2.10: Half-Level Model output for different shafts
(Adapted from Anglo Platinum MTS, 2005)

Some points are worth noting on the output from the Anglo Platinum Half-Level Optimisation Model. These are:

- Figure 2.10 is based on one deposit but with different configurations of mining layouts but was adapted to represent typical output obtained if different deposits using the same conventional breast mining method were compared.

- As Figure 2.9 and Figure 2.10 indicate, the development cost per centare mined decreases asymptotically as backlength is increased, following a power function, confirming the work of Lewis (1941) and Zambó (1968).
- Figure 2.10 shows that each deposit has a unique profile of development cost per centare mined versus backlength, although all profiles are asymptotic, following a power function. The difference in the gradients of the profiles is explained by the differing geological complexity of each deposit. For example geological complexity increases from Shaft Project 6 to Shaft Project 1 as more development is required to negotiate more geological structures on Shaft Project 1 than on Shaft Project 6, in order to expose a centare of stoping. Therefore there can be no single level and raise spacing for all mining operations, hence this study considered a range of optimal level and raise spacing.
- The half-level monthly output increases linearly with increasing backlength, suggesting that although the development cost per centare mined decreases asymptotically with increasing level spacing following a power function, other criteria may have other types of relationships with increasing level spacing (or raise spacing). The relationships between other criteria and, increasing level and raise spacing are analysed in Chapter 6.

The only criticism that can be made on the Anglo Platinum Half-Level Optimisation Model is that it ignores the timing of development costs because it reports the development costs per centare mined not the PV of these costs and that this criterion is prioritised over other criteria treated as secondary drivers, although no justification is provided for its high priority.

2.2.7 Lonmin's Six-Sigma Ideal Ore Reserve Replacement Rate Project

Lonmin undertook a project in 2006 code-named Six-Sigma-Project-174, for the B2# Shaft UG2 Section at Western Platinum Mine. The purpose of the project was to determine the Ideal Ore Reserve Replacement Ratio (IRR) per half-level for the conventional up-dip mining layout through reducing the amount of development by eliminating excess Stope Preparation Drives (SPDs) per stope. The IRR is a measure of the total amount of the development that includes raises, SPDs, boxholes and haulages, required per centare of stoping. It is the equivalent of the Replacement Factor (RF) or Replacement Ratio (RR) in conventional breast mining. The IRR is used to predict the amount of development metres required per centare mined for each particular stope for input into a mine planning Technical Budget (TB). The acceptable IRR rule of thumb that has been used by the company for decades says that the total development (inclusive of all on-reef and off-reef development) required is equal to 10% of the area mined in ca that is being budgeted for in the TB (Nkosi, Kruger and Tyobeka, 2006). The IRR has often triggered debate on whether an ideal ratio can be

determined because geological and geo-technical conditions are never fully known in advance often resulting in the development of unnecessary, unplanned or unauthorized development such as too many SPDs in one raiseline (Nkosi, Kruger and Tyobeka, 2006). Such excess development increases the budgeted cost per ounce of PGEs mined and lowers the ROM grades because there are more development ore tonnages mined. Additionally, mining activities end up deviating significantly from the pre-defined mine standards and long term plans. Nkosi, Kruger and Tyobeka (2006) used linear regression techniques based on historical data of mined-out stopes to set lower specification limits (LSL) and upper specification limits (USL) for the number of SPDs per raiseline. This is equivalent to setting a range of SPD development metres per centare mined in order to control SPD development. The geological structures that were modelled include rolling reef, potholes and dykes. The findings from the study that are relevant to this research study are:

- It is not adequate to set a precise value for planning parameters such as RF (determined from level and raise spacing) because geological and geo-technical conditions are never fully known in advance. Rather, based on past history, an optimal range defined by a LSL and USL can be determined and used whenever unexpected mining conditions are encountered. This is why in this research study it was more appropriate to determine a range of level and raise spacing in order to cater for uncertainty in mining conditions.
- Planning models for development are inadequate if they ignore geology or assume uniform geology throughout the entire orebody.

2.3 Explanation of power relationship between development costs and spacing

When level and raise spacing are varied, a number of observations and implications for development planning can be made. Firstly, a power relationship between level or raise spacing, and development costs is observed. The underlying explanation for this relationship is that raiselines divide strike distance in a way that is analogous to dividing a line into equal parts as shown by Figure 2.11. Consider starting off with a line (or strike distance) and the entire strike distance is equal to the planned raise spacing. Only one stope is possible. If each half is further divided repeatedly into two equal parts (akin to repeatedly halving the raise spacing), the number of stopes increase following the series $2^0, 2^1, 2^2, \ldots, 2^n$ where n is the number of times the line or strike distance has been divided. This division process is not convergent because the number of stopes keeps getting larger and larger. The stope size therefore decreases following the power function $\left(\frac{1}{X^n}\right)$ as shown in Figure 2.11where X is the number of equal parts into which the strike distance is being divided at a time. This is a power relationship which can be applied to dividing dip distance into levels.

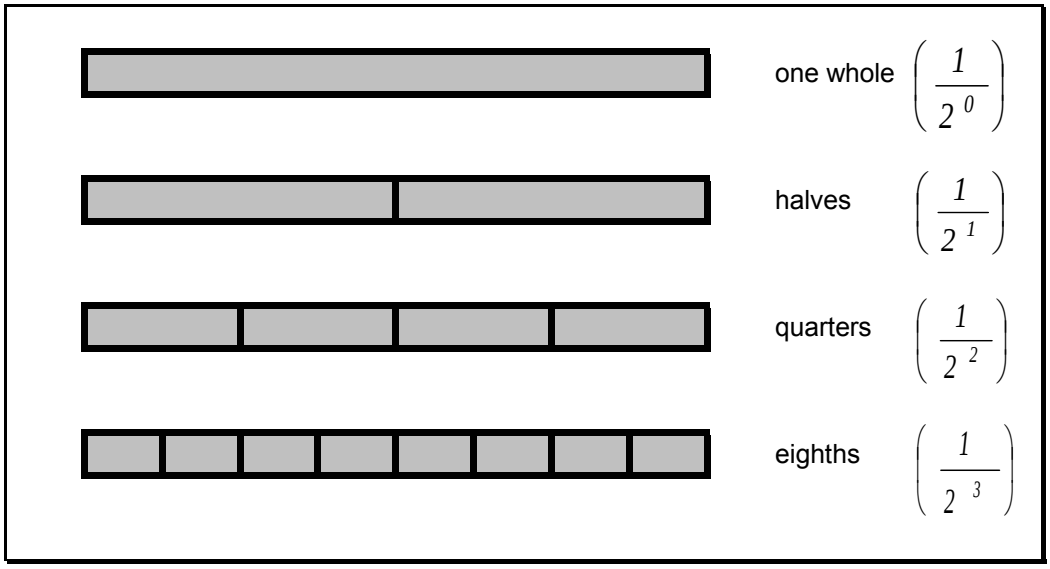


Figure 2.11: Analogy of line and strike distance being repeatedly into equal parts

2.4 Other related studies on planning the location and timing of development

Other studies not specific to level and raise spacing optimisation in inclined narrow reefs but addressing the general planning of the location of underground mine development include Young (1923), Lizotte and Elbrond (1985), Hjadasiński (1995), Macfarlane (1997), Kirk (1997), Diering (1997), Nilsson (1998), Bullock (2001), Brazil *et al* (2003), Brazil *et al* (2004), Brazil *et al* (2005) and Ballington *et al* (2005). The key issues coming from these studies relevant to this research study are:

- Economic and technical considerations sometimes tend to be contradictory when planning development for underground mining and a compromise must be made between these two to achieve optimal extraction. Financial wisdom demands that development, which is an expense and locks up capital, be deferred as far into the future as possible yet on the contrary technical knowledge suggests that developing well ahead of stoping is practically desirable because it generates additional geological information required to improve planning of the remainder of the unmined orebody thus, creating better operational flexibility. The concept of operating flexibility is further addressed in Chapter 3.
- Mine operators and planners tend to focus more on costs than any other value drivers when looking at maximising margins usually leading to sub-optimal solutions. This has been identified in earlier sections as a major shortcoming in most of the methods used to optimise level and raise spacing.

- Geological constraints are not incorporated in most optimisation models, whereas in practice layouts are designed to honour geological boundaries and structures.
- The timing of development costs is critical to the economic success of a mining project because as Bullock (2001:18) contends, *"timing of a cost is often more important than the amount of the cost"*. This is why in this research study the PV of development costs per centare and not development costs per centare was used as one of the optimisation criteria.
- For an open pit deposit, the direction of mining is essentially down and an outward to the pit limits (Hatch Associates, 2004). The mining direction is the basis upon which the "nested pit" approach in Whittle-4D was developed. However, for the underground mining situation, there are numerous permutations of the direction of mining, such as advance or retreat mining, depending on the mining method chosen (Carter, Lee and Baaarsma, 2004). The lack of extensive optimisation analysis in underground mining layouts and schedules is largely attributable to the increased complexity of the problem when compared to open pit layouts and schedules.
- Mining of a mineral block in an open pit is constrained by following relatively simple logical sequences rules for the removal of overburden and the mineral blocks above it and adjacent lateral blocks to form stable slope. For an underground mineral block, there is no single logical sequence for tunnelling through the overburden and adjacent blocks can at times be left unmined only to be recovered later in a retreat sequence, thus sometimes making the problem an unconstrained optimisation problem.

It is therefore clear to see why models, algorithms and software are a common routine for the optimisation of open pit mine designs, and have been well-developed and been in use for many years. Examples include the Lerchs-Grossman algorithm and Whittle-4D commercial software. However, the design engineer for underground metalliferrous mines has had to rely on experience and a limited analysis of design alternatives due to the increased complexity of the underground optimisation problem (Alford, 1995; Brazil *et al*, 2004; Carter, Lee and Baarsma, 2004; Ballington *et al*, 2005; Smith and O'Rourke, 2005). A consequence of this difficulty has been that literature on the optimisation of underground mine designs is relatively scant and fragmented when compared to the abundant literature available on open pit optimisation (Alford, 1995; Brazil *et al*, 2004; Carter, Lee and Baarsma, 2004).

2.5 Overview of multi-criteria decision analysis (MCDA) methodology

The paper Musingwini and Minnitt (2008) on MCDA methodology that was published from this research is partly based on this section and also on Chapters 1, 5 and 7. The strategic mine planning process is not a once-off event, but an on-going process because mining plans need to be continually re-optimised because consideration must be taken of new information about the orebody, technological, economic and social changes (McCarthy, 2006). Optimisation and re-optimisation of mine plans currently occurs in an environment that is characteristically multi-criteria and increasingly complex due to rapid economic, technological, environmental, social, political and legal changes. The decision criteria associated with these changes are inextricably linked and sometimes inherently contradictory, for example the contradiction between financial demands and technical knowledge mentioned in Section 2.4. Consequently mine management and strategic mine planners are faced with the challenge of delicately balancing all these criteria when executing strategic mine planning, in order to achieve optimal mineral extraction. This is the reason why good governance and compliance reporting requirements in South Africa, that were compiled by the statutory King Committee on Corporate Governance, now require mining companies to report not just on the single bottom line (i.e. economic performance) but on the triple bottom line (IOD, 2002). The triple bottom line embraces the economic, environmental and social aspects of the company's activities (IOD, 2002). Pursuant to the principles of sustainable development, the Minerals and Petroleum Resources Development Act (MPRDA), No.28 of 2002 advocates *"the integration of social, economic and environmental factors into planning, implementation and decision making so as to ensure the mineral and petroleum resources development serves present and future generations"* (MPRDA, 2002:16). Optimising level and raise spacing for a conventional breast mining layout is part of the strategic planning process and must therefore be treated as a multi-criteria decision analysis (MCDA) optimisation process.

The challenges faced in a MCDA optimisation process include but are not limited to the following:

- The optimal decision must be one that carefully balances conflicting objectives (or criteria) by selecting the best trade-off among the competing objectives or criteria (Ballington *et al*, 2005; Vieira, 2004; Chen, 2006).
- The optimisation criteria have different units of measure and the challenge is to integrate more than two different criteria that are measured in different units. For example when raise spacing is increased, it is difficult to configure how to achieve an optimal trade-off between a decrease in productivity that is measured in centares/man/month with an increase in RF that is measured in m^2/m, unless the

importance attached to either criterion is known. The trade-offs will be too complex to configure if the criteria display a mixture of relationships that take other non-linear forms. For example it is difficult to configure a trade-off between two criteria if one is varying logarithmically while other one is varying quadratically with increasing level and raise spacing.

- The human brain can easily configure an optimal decision such as deriving maximum benefit or minimum loss when faced with a 2-dimensional problem expressed as a quadratic function in an x-y Cartesian plane, or when the decision problem is 3-dimesnional expressed as a surface in 3-D x-y-z space. Ballington *et al* (2005) alternatively refer to such a 3-D surface as a 'Hill-of-Value'. When optimisation decisions involve decision criteria that exceed 3-dimenions, humans have to rely on abstract thinking or attempt to simplify the problem back to 2-D or 3-D for easier configuration. However, as Saaty and Ozdemir (2003), Yavuz (2007), Yavuz and Pillay (2007a), Yavuz and Pillay (2007b) and Saaty (2008) noted, there are general limitations on human performance on abstract thinking.

MCDA methodologies are premised on addressing the above challenges. The following sub-sections explain how the MCDA methodologies are structured to meet these challenges.

2.5.1 Structure of the MCDA decision problem

The basic structure of a generic MCDA problem (Table 2.1) is premised on requiring a decision-maker (DM) to select an alternative, A_i, from a set of alternatives, $A = \{A_1, A_2, \ldots, A_m\}$, such that A_i gives the best trade-off among decision criteria defined by a set, $C = \{C_1, C_2, \ldots, C_n\}$. In total there are m alternatives and n criteria. The efficiency of alterative 1 against criterion 1 is expressed as the outcome O_{11}, that of alternative i against criterion j, as outcome O_{ij} and so on.

Table 2.1: The structure of a generic MCDA problem

		Criteria					
		C_1	C_2	…	C_j	…	C_n
Alternatives	A_1	O_{11}	…	…	…	…	…
	A_2	…	…	…	…	…	…
	…	…	…	…	…	…	…
	A_i	…	…	…	O_{ij}	…	…
	…	…	…	…	…	…	…
	A_m	…	…	…	…	…	O_{mn}

2.5.2 Decisions within MCDA framework

The decision framework for MCDA is that there are several alternatives (or feasible solutions) to a decision problem. Decisions must then be made using several criteria to evaluate the merits of each alternative. Typical MCDA decisions can take any one of the following forms (Chen, 2006):

- **Choice.** Choosing one alternative (the best compromise alternative) from a set of alternatives. For example in Figure 2.12, alternative A_2 is the best alternative from the set, A, with seven alternatives.
- **Sorting.** Arranging the alternatives into homogeneous groups starting with the most preferred group of alternatives and ending with the least preferred group of alternatives. For example in Figure 2.13 Group 1 containing alternatives A_2, A_1 and A_6, is a preferred group to Group 2 containing alternatives A_5, A_4, A_7 and A_3. Sorting is useful when more than one alternative must be chosen or as a preceding step for a choice decision when enough information is not available to reach a final choice directly or there are too many alternatives that must be considered. Figure 2.13 illustrates how sorting would precede choice using the information in Figure 2.12.
- **Ranking.** Arranging alternatives in an order that starts with the most preferred alternative and ending with the least preferred alternative. For example in Figure 2.12, the ordered sequence of alternatives from A_1 to A_7 follows the preference sequence $A_2 \succ A_1 \succ A_6 \succ A_5 \succ A_4 \succ A_7 \succ A_3$, where $\succ$ means 'preferred to'. Ranking usually precedes a choice or sorting decision when enough information is not available to reach a final choice directly or there are too many alternatives that must be considered. Figure 2.12 illustrates how ranking would be done.
- **Description.** Describing alternatives in terms of their major distinguishing features (Figure 2.12).

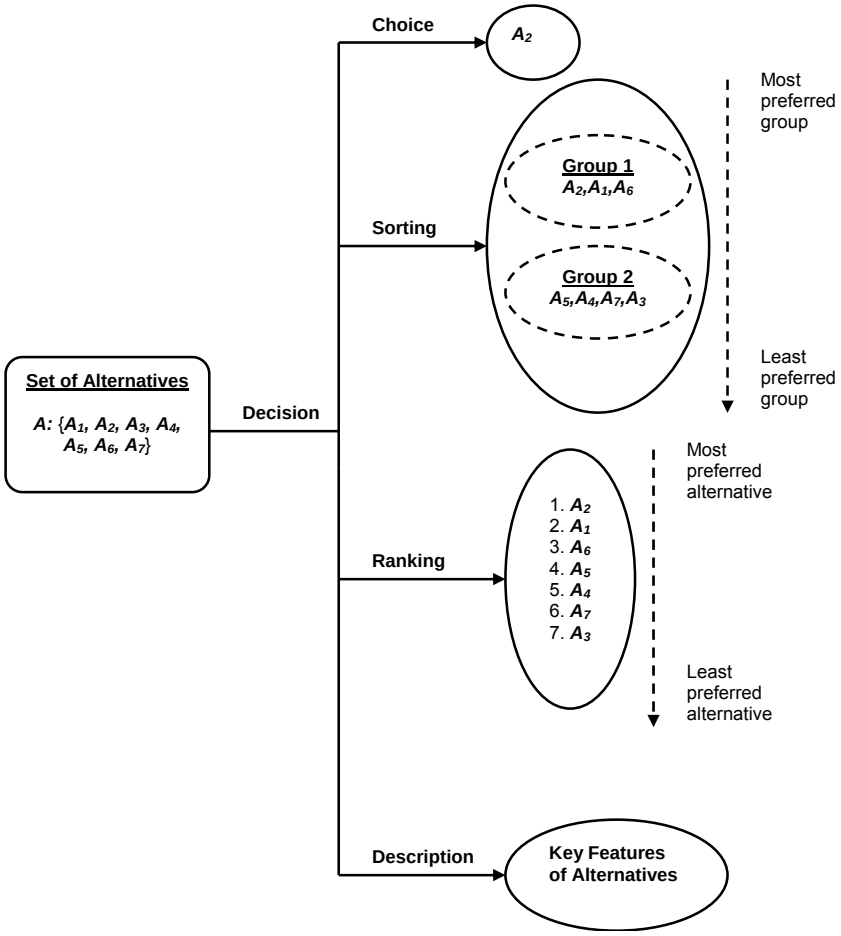


Figure 2.12: Range of MCDA decisions
(Adapted from Doumpos and Zoupodis, 2005; Chen, 2006)

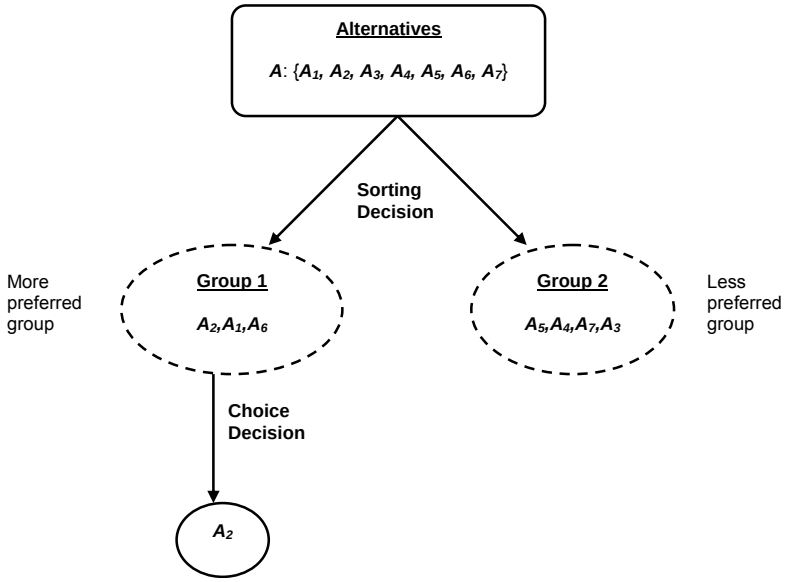


Figure 2.13: Relationship between Sorting and Choice decisions for a large set of alternatives
(Adapted from Chen, 2006)

In this research each layout at a particular level and raise spacing is an alternative that must be evaluated and the decision is a choice decision to select the best compromise or optimal layout. In total 15 layouts were evaluated as described in Chapters 5, 6 and 7 by taking a cue from Chen (2006:14) who conceded that:

> *"Most DMs would like to limit the number of alternatives for analysis. The number that is reasonable may vary greatly according to the circumstances. Twenty may be too many, and two is likely to be too few… In fact, the number of alternatives to be identified may depend on the Problématique. For ranking and sorting problems, all possible alternatives within pre-specified boundaries should be considered. … For choice problems, it may not be necessary to give comprehensive evaluations of all possible alternatives, because some inferior alternatives are not worth further consideration"*.

2.6 The generic MCDA process

The generic MCDA process begins with defining objectives, mapping them into decision criteria, assigning weights to the criteria to indicate the importance of each criterion to the overall objective, identifying all possible alternatives, measuring the efficiency of each alternative against each criterion, synthesising the performance of each alternative for all the criteria and lastly making a decision of either choice, sorting, ranking or description of the alternatives. These stages which are briefly discussed in the next sub-sections can be condensed into basically four steps:

- Determining objectives and mapping them into criteria
- Assigning weights to criteria
- Aggregating the O_{ij} values using the weights
- Execute decision

2.6.1 Mapping objectives into criteria

There are three kinds of criteria namely, natural criteria, constructed criteria and proxy criteria (Keeney, 1992). Natural criteria can be measured directly and physically. For example in this research study the objective, 'maximise production rate' was mapped into the criterion, 'production rate' and measured in 'tpa'. Constructed criteria cannot be measured directly and physically but are measured through a derived index. For example in this research study the objective, 'maximise technical operating flexibility' was mapped into the criterion, 'Flexibility Index, *FI*' using a dimensionless scale. Proxy criteria are indirect measures of an objective when it is difficult to identify a natural or constructed criterion for that objective. For example, if the objective is to, 'minimise the amount of Acid Mine Drainage (AMD) released by a tailings dam' then water draining from the dam is measured

for the proxy criteria acidity and Iron III Hydroxide, which are the products of AMD formation. Acidity is measured on a pH scale while Iron III Hydroxide is measured in mg/l of drainage water and both are then used in conjunction to indicate the degree of AMD. In this research study no proxy criteria were identified as all were either natural or constructed criteria.

Irrespective of the category that a criterion falls into, all criteria must be measured. Criteria can be measured qualitatively (as non-numerical data or linguistic data) or quantitatively as numerical data. Linguistic scales are usually converted to equivalent numeral scales to facilitate analysis. For example a linguistic scale can be assigned numerical values on a scale of 1-10, such that 'low' is equivalent to 1-3, 'medium' to 4-6, and 'good' to 7-10. The numerical data on efficiency of alternatives when measured against decision criteria can be further divided into any of the following three broad classes (Chen, 2006):

- **Cardinal data**. Data is cardinal if the outcome O_{ij} is a real number. Layout efficiency data used in this research study belongs to this group of data.
- **Ordinal data**. Data is ordinal when it is reported using linguistic scales.
- **Interval data, Probabilistic data, Fuzzy data**. This type of data accounts for uncertainty and is expressed as a probability function.

2.6.2 Assigning weights to criteria

The importance of each decision criterion is measured by assigning a weight to the criterion. There are three fundamental principles that must be fulfilled before weights can be assigned to criteria (Chen, 2006). The first principle is the principle of *preference availability* which requires that a DM should be able to express the preference between any two outcomes on a criterion, implying that data on O_{ij} must exist for all criteria and alternatives. The second principle is that of *preference independence* which requires that the DM's preference on one criterion must not have a bearing on the DM's preference on other criterion. The third and last principle is that of *preference monotonicity* which states that a criterion is a positive preference criterion if and only if larger O_{ij} values are preferred; it is a negative preference criterion if and only if smaller O_{ij} values are preferred and it is monotonic if it is either positive or negative. The criteria used in this study met all these principles and therefore could be assigned weights.

There are two broad categories of criteria weights namely, trade-off based weights and non-trade-off weights (Belton and Stewart, 2002). Trade-off based weights require the pair-wise comparison of criteria, thus creating some kind of 'compensation' across criteria. Non-trade-off based weights do not require trade-offs to be made across criteria. MCDA approaches called outranking methods and discussed in Section 2.7 generally use non-trade-off based

weights. The most common trade-off based methods are the Swing, Geometric ratio weighting and ordinal ranking. In this study, the geometric ratio weighting was used because it is integrated as part of the AHP process which is discussed in detail in Section 2.7.4. The raw data is usually obtained from questionnaire surveys or by techniques such as the Delphi technique or the Vicekry-Groves-Clarke method (Darwish and Butt, 1989; Gordon, 1994; Cox, Alwang and Johnson, 2000; Linstone, Turoff and Helmer, 2002).

The Delphi technique uses anonymous questionnaires to obtain criteria weights from a group of decision-makers. The group is chosen such that they remain anonymous to each other. Each decision-maker uses an ordinal scale to rank the decision criteria and clearly states any assumptions made in arriving at the ranking. A statistical analysis is then performed to analyse the assessments using such statistical measures as medians and quartiles. The results are then distributed to the group and each decision-maker requested to revise their earlier assessment based on the summary results of the group. The revised results are then analysed statistically and the process repeated until a consensus is reached. The demand-revealing voting process (also called the Vickrey-Groves-Clarke method) requires each decision-maker to choose the criterion he/she prefers from a set of criteria and how much he/she will be willing to pay to have that criterion over others. The weight of a criterion is then the sum of the dollar amounts for each criterion from each decision-maker. The criterion with the highest dollar amount is the most preferred criterion. For this study the raw data for criteria weights was more appropriate to use a questionnaire survey as described in Chapter 7 and details of which are contained in Appendix 10.2 in order to obtain the weights.

Weights measure the relative importance of a criterion to the overall objective. The weight of a criterion $\boldsymbol{C_j}$ is denoted $\boldsymbol{w_j}$ where $\boldsymbol{w_j} \in$ R and $\boldsymbol{w_j}$ >0 for all criteria. Weights are normalised to sum up to 1 as shown in Equation 2.9 in order to assist DMs interpret the relative importance of each criterion.

$$\sum_{j=1}^{n} w_j = 1$$

Equation 2.9

The weight vector for each alternatives is then defined as **w =** ($w_1, w_2, \ldots, w_j \ldots, w_n$) since the number of weights should be equal to the number of criteria.

2.6.3 Aggregate the weights and O_{ij} values

The outcomes of the efficiency of alternatives against criteria are normalised using a value function so that each $\boldsymbol{O_{ij}}$ score corresponds to a dimensionless value $\boldsymbol{v(A_i)}$. The weights of

the criteria and the outcomes of the performance of alternatives are then aggregated as a linear additive value function, $V(A_i)$, defined by Equation 2.10.

$$V(Ai) = \sum wj.v(Ai)$$

Equation 2.10

2.6.4 Execute decision

The aggregate values are then used to derive the specific decision required by the MCDA problem. This can take the form of a choice, sorting or ranking decision as discussed in Section 2.5.2.

2.7 Main categories of MCDA methodologies

There are four broad categories of MCDA methods and these are the **El**imination **a**nd **C**hoice **T**ranslating **Re**ality (ELECTRE), **P**reference **R**anking **O**rganisation **Meth**od for **E**nrichment **E**valuation (PROMETHEE), **M**ultiple-**A**ttribute **Ut**ility (MAUT) and **A**nalytic **H**ierarchy **P**rocess (AHP) and its subsequent version the **A**nalytic **N**etwork **P**rocess (ANP) (Almeida, Alencar and Miranda, 2005; Geldermann and Rentz, 2005; Saaty, 1980; Saaty, 2008). The methods are classified according to the type of information given by the DM and its salient features depending on whether it is ordinal or cardinal scale information (Geldermann and Rentz, 2005). MAUT and AHP methods are most often applied when the information is cardinal while ELECTRE and PROMETHEE methods are applied to mostly ordinal scale information (Geldermann and Rentz, 2005). The ELECTRE and PROMETHEE methods are founded on the outranking procedure. Outranking is done to account for the fact that preferences are not constant in time, are not ambiguous, and are not independent of the process of analysis (Geldermann and Rentz, 2005). Saaty (2008:7) concurs with the argument that human preferences are fluid because, *"people, then, not only have different feelings about the same situation, but their feelings change or can be changed by discussion, new evidence, and interaction with other experienced people"*. The outranking argument is that an alternative $\mathbf{A_i}$ outranks or is superior to alterative $\mathbf{A_j}$ if the DM strongly perceives $\mathbf{A_i}$ to be at least as good as $\mathbf{A_j}$. A comparison of two alternatives is called a pair-wise comparison. A brief description of each group of methods is discussed in the next sub-sections.

2.7.1 ELECTRE methodology

The ELECTRE group of methods comprises of the versions ELECTRE I, ELECTRE II and ELECTRE III. The ELECTRE methods are more difficult to explain to decision-makers in

industry because they work on thresholds that have no realistic meaning (Geldermann and Rentz, 2005). The subtle differences among pair-wise comparisons usually complicate the ELECTRE decision-making process.

2.7.2 PROMETHEE methodology

The PROMETHEE group of methods comprises PROMETHEE I and PROMETHEE II. The PROMETHEE group was developed to overcome the main problem associated with ELECTRE methods, that of nuances in the pair-wise comparisons (Geldermann and Rentz, 2005). The fundamental mathematical model underlying the PROMETHEE methods is that when comparing two alternatives $\boldsymbol{A_i}$ and $\boldsymbol{A_j}$ for each criterion, *k,* a preference function P_k can be defined as indicated by Equation 2.11.

$$P_k(f_k(\boldsymbol{A_i}) - f_k(\boldsymbol{A_j})) = P_k(d) \in [0,1]$$

Equation 2.11

where $P_k(d)$, is the difference in the degree of preference for alternative $\boldsymbol{A_i}$ over $\boldsymbol{A_j}$ and varies from $P_k(d) = 0$, representing indifference in preference through a zone of weak preference, then a zone of strong preference up to $P_k(d) = 1$, representing strict preference. The PROMETHEE algorithm can be summarised into six steps as outlined below (Geldermann and Rentz, 2005):

- For each criterion, *k*, specify a generalised preference function, $P_k(d)$. The preference function can take any of six possible forms of function distributions which are the criterion distribution; quasi-criterion distribution; criterion with linear preference distribution; level criterion distribution; linear and indifference area distribution; and Gaussian distribution.
- Define a vector of weights that indicate the relative importance of each criterion given by, $w^T = [w_1, \ldots, w_k]$ as expressed by the DM.
- Define the outranking-relation, π, for all alternatives $\boldsymbol{A_1}, \boldsymbol{A_n} \in \boldsymbol{A}$ as indicted by Equation 2.12.

$$\pi(A_i, A_j) = \sum_{k=1}^{k} w_k * P_k(f_k(A_i) - f_k(A_j))$$

Equation 2.12

- Calculate the leaving flow $\Phi^+(\boldsymbol{A_j})$, defined by Equation 2.13.

$$\Phi^+(Ai) = \frac{1}{T} \sum_{j=1}^{T} \pi(Ai, Aj)$$

Equation 2.13

- Calculate the entering flow $\Phi^-(\boldsymbol{A_j})$, defined by Equation 2.14.

$$\Phi^{-}(A_i) = \frac{1}{T}\sum_{j=1}^{T}\pi(A_j, A_i)$$

Equation 2.14

- ❖ Perform a graphical evaluation of the outranking relation. Generally, the higher the leaving flow and the lower the entering flow, the better the alternative.

The PROMETHEE methods use graphical output to show the partial pre-order of the alternatives represented as nodes and the outranking relations depicted as arcs.

2.7.3 MAUT methodology

Vieira (2003), Vieira (2004) and Vieira (2005) used the MAUT methodology to select an optimal mining method from four possible methods to mine ultra-deep gold deposits of the Witwatersrand Basin based on rock engineering risk assessments. The inherent assumption made by Vieira (2003), Vieira (2004) and Vieira (2005) was that each of the four different mining methods was already optimised. The MAUT methodology is based on utility, a concept that evolved from the branch of economics. In economic terms utility is simply satisfaction. Common among individuals or individual groups of people is the need to maximise utility (U) by maximising desirable outcomes and minimising undesirable outcomes.

The MAUT structures the problem as a hierarchy with the primary objective occupying the pinnacle of the hierarchy and having first-layer and second-layer objectives below the primary objective, arranged in terms of hierarchical importance. The objective is measured using attributes (i.e. criteria). Feasible alternatives are represented by, a_i, attributes are represented by x_j, the trade-off weights of attributes are represented by w_{ij} and p_{ij} is the most likely probability of attaining a pre-determined value of efficiency measure which alternative a_i scores against attribute x_i. The overall relative utility (U_i) of an alternative, a_i, is given by Equation 2.15.

$$U_i = \sum_{j=1}^{n} p_{ij} w_{ij}$$

Equation 2.15

2.7.4 AHP methodology

Saaty (1980) developed the AHP methodology. Matrix and vector algebra form the basis of the mathematical framework of the AHP methodology, thus AHP calculations can be easily performed in Microsoft Excel®. Additionally there are generic off-the-shelf software such as

Expert Choice® and *DecisionLens*® that can be used to solve AHP problems. The books, *The Analytic Hierarchy Process,* by Saaty (1980) and *Decision Making for Leaders: The Analytic Hierarchy Process for Decisions in a Complex World,* by Saaty (2008) comprehensively explain the AHP mathematical theory. A brief discussion of the AHP theory now follows.

The AHP methodology is premised on four main axioms which are (Saaty, 1986; Harker and Vargas, 1987):

- **Axiom 1** – the reciprocity axiom. Given any two criteria C_i and C_j, the degree of preference of C_i over C_j is an inverse of the complementary preference decision of C_j over C_i.
- **Axiom 2** – the homogeneity axiom. When comparing two alternatives or two criteria, the scale of the ratio of comparison is bounded (i.e. alternative/criteria one cannot be infinitely better than alternative/criteria two).
- **Axiom 3** – the dependence axiom. The set of alternatives is dependent on the set of criteria if a fundamental scale can be defined to measure each alternative against each criterion (i.e. the decision problem can be formulated as a hierarchy).
- **Axiom 4** – the expectations axiom. All alternatives and criteria which impact a decision-making problem are represented in the hierarchy and assigned priorities compatible with the expectations.

It is not the intention of this study to prove these axioms as such proof can be found in Saaty (1986) and Harker and Vargas (1987). Rather the intention is to recognise that the decision problem in this study satisfies all the four axioms and can therefore be solved using the AHP methodology.

The mathematical procedure starts with a pair-wise comparison of the relative weight or importance of each criterion over another using the reciprocity axiom. The relative weight of C_i over C_j is denoted by w_{ij} such that, $w_{ij} = \frac{1}{w_{ji}}$, $\forall\ i \neq j$, and $w_{ij}=1$, $\forall i=j$, since a criterion is as important as itself. These weights form a square matrix $\mathbf{W} = (w_{ij})$, of order n, corresponding to the number of criteria. The matrix, **W**, is referred to as a reciprocal matrix because the inverse of the weight of one criterion over another is equal to the weight of the second criterion over the first one. For example if capital costs are twice as important as operating costs in choosing a mining method, then logically operating costs will be half as important as capital costs.

The matrix of weights, **W**, is then evaluated for transitivity. A relationship is transitive if the relative importance is multiplicative. For example, if criterion C_2 is twice as important as

criterion C_1 and criterion C_3 is three times as important as C_2, then logically criterion C_3 should be six times as important as C_1. A matrix satisfying the transitive axiom represents consistent judgements. Typical human judgements are characteristically inconsistent to a greater or lesser degree and cannot satisfy the transitive axiom. The AHP methodology provides a way of measuring the degree of inconsistency in judgements.

The transitive relationship between weights can be expressed mathematically as $w_{ik} = w_{ij}w_{jk}$, $\forall i,j,k$. A vector, $\boldsymbol{w}$, of order $\boldsymbol{n}$ can be established such that $\mathbf{W}\boldsymbol{w} = \lambda\boldsymbol{w}$. The vector, $\boldsymbol{w}$, is called an eigenvector of the matrix $\mathbf{W}$ and the constant λ is its corresponding eigenvalue. If the matrix, $\mathbf{W}$, is consistent then $\lambda = \boldsymbol{n}$. For inconsistent human judgements, the eigenvector, $\boldsymbol{w}$, cannot satisfy the earlier condition but will satisfy the condition $\mathbf{W}\boldsymbol{w} = \lambda_{max}\boldsymbol{w}$ such that $\lambda_{max} \geq \boldsymbol{n}$. The difference between λ_{max} and $\boldsymbol{n}$ indicates that there is some inconsistency in the judgements but, if $\lambda_{max} = \boldsymbol{n}$ then logically, the judgements were consistent.

Several methods are available for estimating the eigenvector. Of these, a close approximation of the eigenvector is obtained when geometric means are used to estimate the eigenvector elements. The rationale for geometric means is simple. If a typical scale of 1 to 10 is used to denote the relative weights, then from the reciprocity axiom, the reciprocal weights 0.1 and 10 will differ by an order of magnitude of 100. Costa (2007) indicated that geometric means are meaningful when evaluating data that differs by several orders of magnitude, the minimum order being three (i.e. the largest number is three times as big as the smallest number in the data set). The geometric mean is useful for such data because unlike the arithmetic mean, it tends to dampen the effect of very high or low values, which could bias the mean if an arithmetic mean were calculated (Costa, 2007).

A Consistency Index, *CI*, is then calculated from λ_{max} and $\boldsymbol{n}$ using the relationship defined by Equation 2.16.

$$CI = \frac{(\lambda_{max} - n)}{(n-1)}$$

Equation 2.16

In order to determine if judgements are reasonably consistent a Consistency Ratio, *CR*, is calculated by assessing the calculated *CI* against judgements that are made completely at random. Saaty (1980) simulated large samples of random matrices of increasing order and calculated their corresponding *CI*s which are random indices, *RI*s. For matrices of order between 1 and 15, Saaty (1980) established the corresponding *RI*s as shown in Table 2.2.

Table 2.2: Random Index (RI) for n-ordered matrix
(Source: Saaty, 1980)

Matrix order	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
RI	0.00	0.00	0.58	0.90	1.12	1.24	1.32	1.41	1.45	1.49	1.51	1.48	1.56	1.57	1.59

The *CR* is obtained by dividing the *CI* by its corresponding *RI*. Saaty (1980) suggests that if the *CR* exceeds 0.1 then the judgements are likely to be too inconsistent to be reliable and the assignment of weights to criteria should be redone. The threshold ratio of 0.1 can be interpreted to mean that the judgments are approximately 10% random and a ratio of 1.0 would therefore mean that the judgements are completely too random to be trusted. A *CR* ratio of 0 therefore implies that judgements are perfectly consistent (i.e. not random at all). In practice *CR*s of more than 0.1 are sometimes accepted provided there is adequate justification for their acceptance (Coyle, 2004).

If the degree of inconsistency in judgements is acceptable, the efficiencies of all alternatives on a criterion, O_{ij}, are then normalised to eliminate the effect of different units of measure for each criterion. For m alternatives on a criterion, the normalised O_{ij} values denoted by, O^N_{ij}, are derived as shown in Equation 2.17.

$$O^N{}_{ij} = \frac{O_{ij}}{\sum_{i=1}^{m} O_{ij}}$$

Equation 2.17

The matrix of normalised efficiency outcomes is finally multiplied by the eigenvector to obtain the aggregated AHP priority score. The decision is then made based on the logic that the higher the AHP priority score for an alternative, then the more preferable the alternative.

There are three main limitations of the AHP methodology. Firstly, the AHP only works if the matrix for the criteria weights is a positive reciprocal matrix (Coyle, 2004). Positive reciprocity is satisfied if criterion C_i is x times more important than criterion C_j and correspondingly C_j is $\frac{1}{x}$ times as important relative to criterion C_i. Secondly, when the scale for measuring the relative importance of criteria with respect to each other is changed, say from a scale of 1 to 10 to a scale of 1 to 20, the weight vector will also change, in some cases affecting the final decision (Coyle, 2004). Lastly, as the number of criteria to be compared increases, the number of pair-wise comparisons increases rapidly following a power function as shown in Table 2.3, thus clouding judgement and rendering the calculations more complex. For example, for the recommended maximum number of criteria of 9, a total of 36 comparisons have to be made.

Table 2.3: Relationship between number of criteria and comparisons
(Source: Kardi, 2006)

Number of criteria	1	2	3	4	5	6	7	n
Number of comparisons	0	1	3	6	10	15	21	$\frac{n(n-1)}{2}$

2.8 Structure comparison of the four categories of MCDA methodologies

A summary of the structure comparison of the four broad categories of MCDA methodologies is shown in Table 2.4.

Table 2.4: Comparison of the four MCDA methodology categories
(Adapted from Geldermann and Rentz, 2005; Chen, 2006)

	MAUT	**AHP**	**ELECTRE**	**PROMETHEE**
Foundation	Classical MCDA approach	Hierarchical approach	Outranking procedure	Outranking procedure
Theoretical Basis	Utility Function additive model	Pair-wise comparison (weighted eigenvector evaluation)	Pair-wise comparison (concordance analysis)	Pair-wise comparison (Preference Function)
Measurement of criteria	Numerical (Non-numerical data must be converted to numerical scale)	Numerical (Non-numerical data must be converted to numerical scale)	Numerical (Non-numerical data must be converted to numerical scale)	Numerical (Non-numerical data must be converted to numerical scale)
Determination of weights of criteria	Trade-off based weights (generate weights using Swing, Direct-ratio, or Eigenvector methods)	Trade-off (generate weights using Saaty's Eigenvector & geometric mean)	Non-trade-off (Does not provide procedure to obtain weights)	Non-trade-off (Does not provide procedure to obtain weights)
Result	Relative preference order	Relative preference order	A set of non-dominated alternatives	Partial and complete ranking order

2.9 Choice of AHP methodology

The AHP was selected over other MCDA methods in this research study for three main reasons. Firstly, the method has significant advantages which are:

- When compared with other MCDA techniques, the AHP can detect inconsistent judgements and provide an estimate of the degree of inconsistency in the judgements (Coyle, 2004).
- The AHP is supported by an easy-to-use commercially available software package called ***Expert Choice***® (Geldermann and Rentz, 2005) and more recently, ***DecisionLens***® (Saaty, 2008).
- The AHP has the ability to rank alternatives in order of their effectiveness when conflicting objectives or criteria have to be satisfied (Coyle, 2004).

Secondly, the AHP has been successfully used to solve a wide range of MCDA decision problems in the minerals industry and is gaining gradual recognition because most optimisation and decision-making problems encountered in the minerals industry are of a multi-criteria nature as shown by the examples in Table 2.5. Lastly, the AHP was a preferred choice because the layout efficiency data in this research study was cardinal data.

Table 2.5: Examples of minerals industry problems solved using MCDA techniques

Source	MCDA decision problem
Vieira (2003; 2004; 2005)	Used MAUT to select the best mining method from four possible methods to mine ultra-deep tabular gold deposits of the Witwatersrand Basin. Four mining methods were compared on the basis of 49 attributes clustered into five decision criteria.
Almeida, Alencar and Miranda (2005)	PROMETHEE II used to select the mining method for ornamental rocks that best satisfies a set of evaluation criteria. Six mining methods were compared on the basis of five criteria.
Liquin, *et al* (1995)	Used AHP to select an optimal mining plan from a set of possible mining plans for a generic multi-criteria decision-making model.
Elevli and Demirci (2004)	PROMETHEE I and PROMETHEE II used to select the most suitable underground ore transport system for a chromite mine in Turkey. Five alternative transportation systems were compared on the basis of six criteria.
Dessureault and Scoble (2000)	AHP used by a mine to decide whether to purchase new drill-monitoring technology, maintain status quo, or retrain drillers and surveyors to work more productively and safely. The three alternatives were compared on the basis of six criteria.
Karadogan, Kahriman and Ozer (2008)	Used AHP based fuzzy multiple attribute decision-making methodology to select the most suitable underground mining method for the Ciftalan Lignite Mine in Turkey. Five possible mining methods were compared on the basis of 18 criteria.
Bitarafan and Ataei (2004)	Used two methods, an AHP based fuzzy multiple attribute decision-making method and fuzzy dominance method, to select the optimal mining method for extracting the No. 3 Anomaly at the Gol-Gohar iron mine in Iran. Seven mining methods were compared on the basis of 15 criteria.
Ataei (2005)	Used AHP to select the best location of an alumina-cement plant in Iran. Five possible locations were compared on the basis of five criteria.
Kazakidis, Mayer and Scoble (2004)	Used AHP based *Expert Choice*® software to model mining scenarios for selecting the (i) best rockbolt support system from 14 possible rockbolt support systems on the basis of 10 criteria; (ii) best option from five operational options to improve tunnelling advance rates based on seven criteria; and (iii) mine with the highest risk to mine production performance arising from ground problems, from a set of eight mines in a mining company, based on four criteria.
Uysal and Demirci (2006)	Used a hierarchical multi-dimensional objective system similar to AHP to select the more suitable mining method for the ELI and GLI coalfields in Turkey. Two mining methods compared on the basis of 19 criteria.
Wu, *et al* (2007)	Used AHP to advise the board of directors of Wugang Mining Cooperation on the order in which the company was weakest in terms of core competence for each of the four products (iron concentrates; pellets; copper and sulphur concentrates; and non-metallic concentrates). The products were compared on the basis of eight criteria clustered into three criteria.

2.10 Summary

This chapter has demonstrated that optimising level and raise spacing in a conventional breast mining method for the shallow-dipping narrow tabular reefs of the Bushveld Complex is a multi-criteria decision analysis (MCDA) problem. The problem should therefore be

solved using an MCDA methodology. The AHP was chosen as the appropriate MCDA methodology for this research because of its advantages over the other methodologies and that the layout efficiency data was cardinal. The AHP is subsequently used in Chapter 7 to rank the conventional breast mining layouts at different level and raise spacing and then identify the optimal range of level and raise spacing. As was noted in Section 2.4, the next chapter discusses the concept of technical operating flexibility.

3 TECHNICAL OPERATING FLEXIBILITY IN MINE PLANNING

3.1 Introduction

The preceding chapters indicated that optimisation of level and raise spacing is a multi-criteria optimisation problem and that one of the key criteria to be considered is operating flexibility. However practitioners in the narrow tabular reef mining industry, who often make reference to operating flexibility when discussing mining methods, have not quite developed a methodology to measure this criterion and so tend to overlook this factor in the final analysis of mine layouts and schedules. This tendency could be a consequence of the nebulous nature of operating flexibility. By glossing over operating flexibility the resultant mine layouts and schedules may be sub-optimal. The need to incorporate operating flexibility to become an inherent part of mine plans is however, increasing in importance as demonstrated later in this chapter.

The terms 'operating flexibility' and 'technical operating flexibility' are synonymously used in this chapter. This chapter explores the nature of technical operating flexibility, reviews previous work on measuring operating flexibility, and concludes by proposing a method to quantify technical operating flexibility for tabular reef mines by using a case study based on OB1, which is a UG2 platinum reef deposit that is comprehensively described in Chapter 4. The methodology developed to quantify technical operating flexibility using a Flexibility Index (*FI*), was subsequently applied in Chapters 5 and 6 to compare the operating flexibility of 15 different layouts and schedules developed for OB1. For additional reading on the subject of technical operating flexibility, the reader can also refer to the two papers that were written out of this chapter and Chapters 4 and 5, which are Musingwini, Minnitt and Woodhall (2006), and Musingwini, Minnitt and Woodhall (2007).

3.2 Defining operating flexibility

The New Collins Dictionary and Thesaurus in One Volume (1989:382) defines flexibility as *"adaptability, adjustability, elasticity, responsiveness"*, a definition which can be interpreted to mean the ability to adapt, adjust or respond to changes. This definition is fundamental in understanding operating flexibility in mining operations. Mines operate in a business environment characterised by several uncertainties of a financial, technical, environmental, legal or social nature such as fluctuating exchange rates, cyclical mineral prices, changes in government fiscal policies, changes in geological conditions, changes in technology, evolving environmental legislation, rising wage costs or unexpected labour strikes. The complex multi-criteria business environment, in which mining companies in South Africa have to operate in, was described in Chapter 2. Mining plans have to be adjusted or adapted

quite often over the life of the operation to cope with the multi-criteria changes. The ability of mining plans to adjust, adapt or respond to such changes is operational flexibility.

A flexible mining plan can be considered to be a robust mining plan because it is able to adapt to changes in the operating environment. However, operating flexibility in mining is a nebulous concept, difficult to define and measure, because it means different things to different people. Two different kinds of operating flexibility can be identified at two different levels of mining operations, namely strategic and operational. At a strategic or corporate level, operating flexibility is the ability to meet the required shareholders' return on investment at an acceptable level of risk. This type of flexibility is obtained by structuring the company's portfolio of operations to be able to divest from unprofitable operations, take on board new operations and Greenfield projects that are able to generate the minimum required levels of profitability. Operating flexibility at a strategic level has also been referred to as managerial flexibility in real options analysis (Kajner and Sparks, 1992; Samis and Poulin, 1998; Davis, 1998; Dapena and Fidalgo, 2003; Kazakidis and Scoble, 2003).

Operating flexibility at a tactical level is alternatively referred to as 'technical operating flexibility' in this study in order to distinguish it from managerial flexibility at a strategic level. Technical operating flexibility is the ease with which production crews can be relocated to different production faces within the mining operation to respond to grade control requirements, safety constraints or when unpredicted geological structures are encountered. In this way, production and safety risks are minimised. In an underground narrow reef tabular mining operation, technical operating flexibility ultimately translates into mining face availability (Woodhall, 2002). For example, if a bottleneck develops on the hoisting system, the tramming system will in turn be choked leading to un-cleaned mining faces that become unavailable for mining. Clearly, there is a complex interaction of activities to produce overall operating flexibility implying that adequate face availability in itself does not mean an operation is flexible. The available faces have to be close to areas that become unavailable such that there is no lost blast when the relocating production crews, implying that mining activities have to be concentrated within the mining operation. Therefore operating flexibility can only be guaranteed if adequate face availability is created under concentrated mining conditions. This proposition is revisited in Chapter 8.

Woodhall (2002) also explored operating flexibility as the outcome of the complex interplay among geological complexity, layout geometry and economic circumstances. Woodhall (2002:43) underscored the need to have an *"appropriate level of mining flexibility to manage profitability over time"*. Since long-term profitability is a strategic objective and operating flexibility is a tactical objective, operating flexibility is a strategic-tactical nexus. Based on the experience in the study, Woodhall (2002:43) noted the existence of an occasional "hand-to-

mouth" habit by mines in making available suitable mining face because no mine management team can *"claim they have spare mining face just waiting for someone to come and mine it"*. Woodhall (2002:40) also argued therefore that, *"the only true flexibility in terms of having choice to mine or not to mine is therefore equipped panels waiting to be stoped"*. At a tactical planning level, Woodhall (2002:40) distilled the concept of operating flexibility down to the mining face and defined mining operating flexibility for underground tabular reef gold mines as:

> *"the provision of sufficient equipped mining face to make alternative, profitable work places available to sustain planned production levels…and the only true flexibility in terms of having choice to mine or not mine is therefore equipped panels waiting to be stoped"*.

It is necessary that production personnel on the mines have to exercise strategic discipline regarding operating flexibility because it should not be seen as a 'choice' to deviate from mine plans but as inherent 'options' within the mining plan that are available to respond to changes in the operating environment as and when they occur, so that the same level of production can be consistently maintained even under difficult operating conditions.

3.3 Creating technical operating flexibility in narrow tabular reefs

Technical operating flexibility is created by having mining face length that exceeds the face length required to meet planned production. This is achieved initially by keeping development and stope preparation well ahead of stoping activities. For example, Swanepoel (2002:401) described the strategy for creating operating flexibility at Thorncliffe Chrome Mine on the Eastern Bushveld Complex as, *"this flexibility is created through development where the company currently utilizes, on cycle, 40 panels of the available 100 panels"*. However, Swanepoel (2002) does not explain how the additional 60 panels were specifically derived, but the assumption here is that they could have been empirically determined based on past geological and logistical experience on the mine. Smith and Vermeulen (2006) described a similar but clearer strategy that is practised at Anglo Platinum whereby operating flexibility is created through having spare mining face that is determined by adjusting the required face length, which in their example was 200m by the estimated global geological loss. In this strategy, Smith and Vermeulen (2006:S9.9) noted that, *"If a geological loss of 17% is considered and a simple rule of maintained spare face equivalent to the geological loss is applied a minimum of 234m of face (200 x 1.17) is required to sustain production"*. In this way, inherent operating flexibility that accounts for geological losses, is built into the mine plan but may be adjusted further depending on other factors which the mine planner may consider important.

Figure 3.1 is an illustration of the sequence of typical mining phases in the build-up to technical operating flexibility, starting from the time a raiseline has holed. A change or delay in a mining phase directly affects subsequent phases and ultimately technical operating flexibility. Therefore, it can be argued that developing well ahead of stoping operations does not necessarily create operating flexibility because in some cases development laid out well ahead of stoping may not necessarily be in the correct or optimal areas, again leading to a loss of mining face.

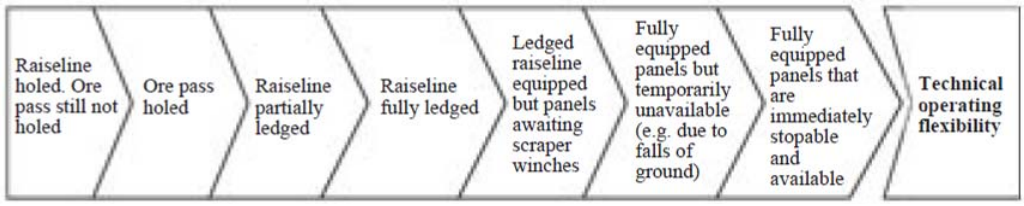


Figure 3.1: Sequence of typical mining phases in the build-up to technical operating flexibility

When the strategies to create operating flexibility are mapped out into a sequence of tactical activities as indicated in Figure 3.1, it becomes apparent that mining operations must have strategies that guide the timing of development ahead of stoping activities. Technical operating flexibility and the timing of development ahead of stoping are therefore two interrelated concepts in the planning of tabular reef mines linked together by the concept of ore availability. Ore availability is a measure of how far development has been kept ahead of stoping operations. It is the amount of ore available for stoping with little or no further development required, expressed in years of production at current rates of production. Ore availability can also be expressed as m^2 of available mining face. Although it is debatable as to what counts as ore availability, a logical reasoning is that all stopes with fully ledged raiselines are available ore, stopes with raises that have holed but not yet ledged are not available for stoping, and stopes with partly ledged raiselines are partially available in the proportion of length ledged to length of raise.

A minimum ore availability of two years is a typical figure for most narrow tabular reef deposits, based on the book by Storrar (1977) and the papers by McCarthy (2002) and Lanham (2004). Storrar (1977:273) referred to ore availability an '*apparent ore reserve life*' and indicated that most tabular reef gold mines on the Witwatersrand Basin considered a figure of two years as being a safe value. McCarthy (2002) discussed rules on keeping development ahead of production and noted that it is usual to keep primary access development two years ahead of production in longhole stoping operations of narrow reefs. In an article on the progress made in extending the life of mine to 18 years at Northam Platinum Mine, Lanham (2004) cited two years as an ideal figure for ore availability for the mine due to its geological complexity, but noted that some platinum mines on the Bushveld

Complex with simpler geology are comfortable with ore availability of 12 months. In the same article, Lanham (2004:19) quoted the then Manager: Projects, Rene Rautenbach as saying that, *"Ore availability last year was 18 months. However, with the delays while the fissures and faults were overcome, this decreased to 15 months. As soon as we get through into good ground, we are going to build up reserves close on to two years to give the flexibility needed when mining a highly erratic orebody such as the Merensky reef"*. It is not surprising therefore that Northam mine has consistently maintained ore reserve availability of between 15months-24months as indicated in their 2006 and 2007 Annual Reports (Northam, 2006; Northam, 2007).

Woodhall (2002) analysed the statistics for a Witwatersrand Basin tabular reef gold mine that was using scattered mining planned on 150m raise spacing and 180m backlenghts and noticed that the mine had, between 1994 and 1999, almost consistently achieved 23 months as the time lag between development effort and the subsequent stoping results. This is an important indicator of the ability to change existing levels of operating flexibility should it be considered necessary, should other factors such as geo-technical factors become more significant. For example, in a study on the development of a just-in-time (JIT) development model for a sub-level caving asbestos mine, Musingwini *et al*, (2003) referred to ore availability as "*buffer reserves"* or "*buffer time"* and noted that the mine had been gradually reducing buffer time from 6 years in 1991 to 4 years by 2001, but the mine was not clear about the minimum level of buffer reserves to keep. By considering past geotechnical experiences on the mine in terms of stand-up times of development openings and associated costs to keep the ends open, and the geological simplicity of the orebodies, the study noted that the figure could be reduced to 6 months. The mine adopted the recommendation and achieved significant cost savings, while still assuring customers of continued medium to long term product supply. The concept of the JIT development model for an operating mine with existing historical data was further explored by Musingwini (2004) to show how operating mines can vary their buffer reserves over time.

Low ore availability tends to lead to reduced technical operating flexibility while high ore availability tends to lead to increased technical operating flexibility. This relationship allows the measurement of technical operating flexibility by varying the levels of ore availability as indicated in Section 3.6.

3.4 Importance of operating flexibility in mine design and planning

The importance of flexibility in mine plans was also highlighted at the *First International Seminar on Strategic versus Tactical Approaches in Mining* held in 2005 in South Africa, where eight of the twenty-four papers presented made reference to and recognised the

importance of flexibility in contemporary mine plans. More recently Elkington, Barrett and Elkington (2006) noted that uncertainty is intrinsic to all mining projects and should be planned for by providing adequate operating and strategic flexibility. The importance of operating flexibility in enhancing project value and profitability, ensuring that optimal mineral extraction paths are obtained and improving safety, is highlighted by in Woodhall (2002), Kazakidis and Scoble (2003), Macfarlane (2005), and Steffen and Rigby (2005) and Johnson (2007), as briefly outline below.

In a study on planning for flexibility in underground mine production systems, Kazakidis and Scoble (2003:34) noted that operating flexibility and strategic adaptability are now increasingly being recognised as critical to long-term corporate success because, *"the ultimate level of profitability of a mining project is enhanced by flexibility in the mine plan"*. Woodhall (2002:43) had also highlighted a similar link between flexibility and profitability because an *"appropriate level of mining flexibility"* was necessary *"to manage profitability over time"*.

Macfarlane (2005:187) underscored the importance of flexibility in mine plans by arguing that, *"where flexibility to deal with changing economic cycles has not been created, (as a value-adding decision) reactive planning has to be undertaken, which is value-destroying"*. Macfarlane (2005) further argued that ideal optimal planning profiles should be those that create value early in the life of a mining project, and part of this value should then be re-invested into building flexibility in the operation. In this way, an optimal path of extraction can be created through the removal of operating constraints, provided the flexibility options are exercised. Steffen and Rigby (2005) argued similarly that flexibility to ensure an optimum production profile from known reserves over the life of the mine is so important that it should warrant executive directive because it involves risk acceptability and directly affects corporate balance sheet capacity.

Woodhall (2005) painted some possible scenarios that can be encountered during planning that link ore reserve development and mine planning. Firstly, *"if we chose a development programme in balance with stoping i.e. mining reserves are created a fast as they are depleted"* (Woodhall, 2005:92) then *"we will find ourselves with a temporary constraint in terms of maintaining volume"* (Woodhall, 2005:92). Secondly, *"if we have chosen to cut back on development and are already limited by availability of reserves, mining is equally constrained"* (Woodhall, 2005:92) because we will *"struggle to maintain volume and cost of production as we open up areas we cannot currently mine"* (Woodhall, 2005:92). Implicit in these conclusions is that the production is constrained due to the absence of flexibility to adapt mine plans. Woodhall (2005:92) therefore recommends that, *"as a strategy, a hesitant approach to reserves development represents a higher risk to production volume and …*

denies a mining operation future opportunity due to lack of flexibility. A robust strategic plan is therefore one that is developed, tested and stands against various scenarios because it has inherent flexibility".

In a discussion with Johnson (2007) it emerged that when some mine accidents were investigated, the reason for the accidents indicated that sometimes employees ended up working in areas that were not as desirably safe because, *"there was not enough flexibility to have safe mining areas available"*. Johnson (2007) was therefore tasked with developing a project to measure level of operating flexibility for Great Noligwa mine and adopted the methodology developed in this chapter and published in the paper Musingwini, Minnitt and Woodhall (2006).

3.5 A metric for measuring technical operating flexibility

Kazakidis and Scoble (2003) noted that by 2003, there was no documented or formalised standard procedure for quantifying or valuing flexibility despite its increasing importance. Kazakidis and Scoble (2003) therefore proposed an index as a metric for measuring managerial flexibility and defined it as indicated by Equation 3.1.

$$Flexibility\ index, \quad F(\%) = \frac{Option\ Value, \quad OV}{NPV\ \ passive} \times 100, OV > 0$$

Equation 3.1

The *OV* is the additional *NPV* over the base case of a project that would be derived from exercising the alternatives made available by the flexibility obtained. However, the flexibility comes at premium that includes additional capital and/or operating costs. Therefore if the flexible option is not exercised, the NPV over the base case will decline, because of the additional costs incurred to acquire the operating flexibility.

It can be observed that the definition of flexibility by Woodhall (2002) is a logistical construct while that by Kazakidis and Scoble (2003) is a value construct, although both constructs are indicative of issues surrounding the measurement of operating flexibility. By considering these two proposals, a metric in the form of a flexibility index, *FI*, can be defined as shown in Equation 3.2.

$$FI = \frac{Available\ Fully\ Equipped\ Stopes + Stopes\ Already\ in\ Pr oduction}{Production\ Stopes\ Re quired\ to\ meet\ Planned\ Production\ Rate}$$

Equation 3.2

The definition assumes that the fully equipped alternative stopes have already been developed, ledged and equipped, and are in close proximity to the working stopes so that no shift is lost in relocating production crews to the alternative working places, should such a need arise. From Equation 3.2, if *FI*<1, then for the period under consideration the operation is inadequately developed for production and has no flexibility at all since there are fewer stopes available than are required to meet the planned production rate. If *FI*=1, then the operation is temporarily inflexible because any unforeseen loss of panels causes the operation to slip back into a situation of no flexibility at all. Accordingly, the development has to be stepped up to bring the operation to an acceptable level of operating flexibility. If *FI*>1, then the operation is flexible. The behaviour of this index with respect to increasing level and raise spacing in conventional breast mining layouts is explored in Chapter 6.

3.6 Behaviour of flexibility index under variable timing of development

The flexibility index (FI) was tested on OB1 using a conventional breast layout that was planned to produce 1.8mtpa on a layout grid of 200m backlength and 200m raise spacing. The design was done in Mine 2-4D® design and planning software. The Mine 2-4D® design was then exported to Enhanced Production Scheduler (EPS®) for scheduling and then to Microsoft Excel® for final analysis. By using the scheduling layer functionality in EPS®, schedules based on ore availability taken in steps of 3 months over the range 0–36 months were then developed, so that flexibility could be investigated 12 months on either side of the customary figure of 2 years. For clarity of illustration, some figures will show only ore availabilities which are in multiples of 3 months. Figure 3.2 shows the production profiles for ore availability taken in steps of 6 months. As ore availability is increased, there is a gradual shift of the production profiles to the right. This is expected because as ore availability is increased, the gap between development completion and production start-up widens, thus pushing the production profile more to the right.

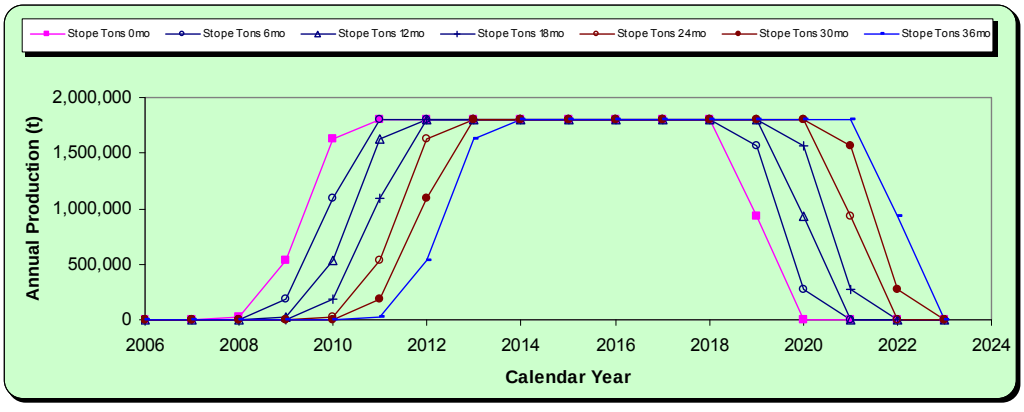


Figure 3.2: OB1 production profiles at variable ore availability

The behaviour of the flexibility index was investigated over the range of 0-36months for ore availability. Figure 3.3 illustrates that from production start-up, the curves steepen with increasing ore availability and the flexibility reaches a peak about halfway through the project life in 2014. This trend shows an increase in technical operating flexibility with increasing ore availability. In the early stages of development, only a few stopes are fully ledged and equipped, but as development progresses, more stopes become available and flexibility increases. Low ore availability does not give the operation enough time to build up beyond a flexibility index of 1.0 as stopes are mined almost as soon as they are developed. This trend can be observed for curves representing ore availability less of than 18 months in Figure 3.3. Beyond half of the project life, flexibility starts to decline because there is less development happening and more stoping occurring. Technical operating flexibility of greater than 1.0 is obtained at ore availability of 18 months or more, suggesting that 18 months should be the minimum ore availability for OB1. This is the ore availability assumed for design and scheduling in Chapter 5. When viewed from a risk perspective, two different risk profiles are evident from Figure 3.3. From start-up to about half of the project life in 2014, there is a higher risk to production because development work is still generating more information about the orebody. Consequently, in this segment of the project, there is a high potential for lost blasts, in turn reducing flexibility build-up, and negatively affecting revenue generation. Beyond 2014, lower risk is expected because most of the development is now complete and there is increased knowledge about the orebody. Most production areas are fully developed, creating the opportunity for fewer lost blasts and increased productivity.

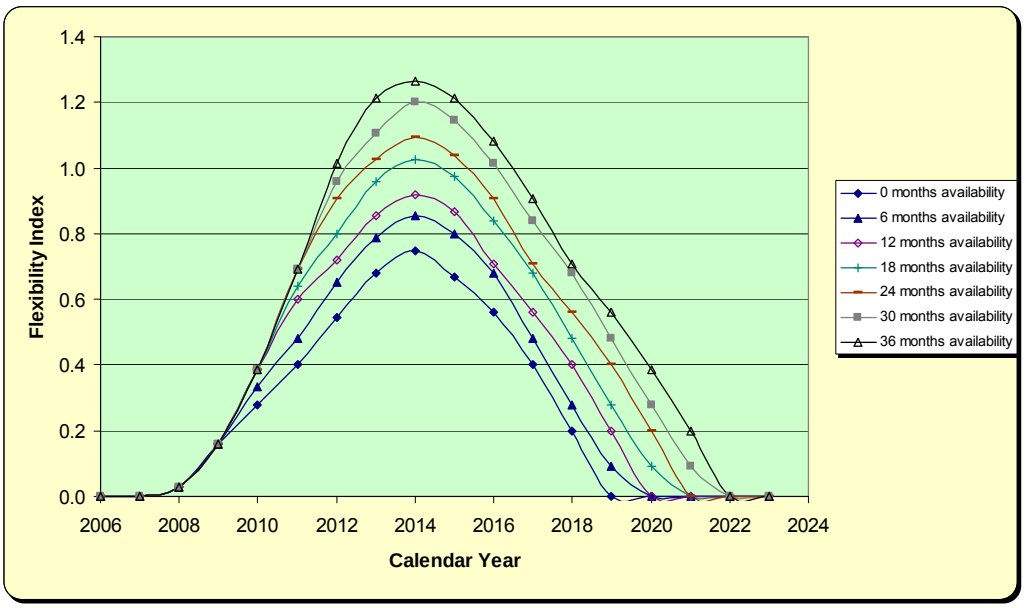


Figure 3.3: Variation of Flexibility Index with ore availability

Finally, the flexibility index and project NPV were plotted for the ore availability range of 0–36 months and the results obtained are shown in Figure 3.4. It is evident from Figure 3.4 that technical operating flexibility increases with increasing ore availability, while project NPV decreases with increasing ore availability, confirming the earlier argument. By delaying the start date for a stope to be scheduled into production, future flexibility is generated. However, this also delays revenue from that stope when costs have been incurred to prepare it for production. As indicated earlier, the longer the delay between completion of stope preparation and production start from that stope, the greater the reduction in project NPV. For each particular level of ore availability the corresponding NPV and flexibility index values were computed. The NPV was based on net of development and stoping costs and revenue from production. However, it is expected that NPV would increase with increasing flexibility if the alternative stopes are utilized, for example, to increase production rate or to take advantage of a spike in mineral price. For example, the benefit of meeting planned production levels can be quantified by valuing production loss that would occur if flexibility were absent.

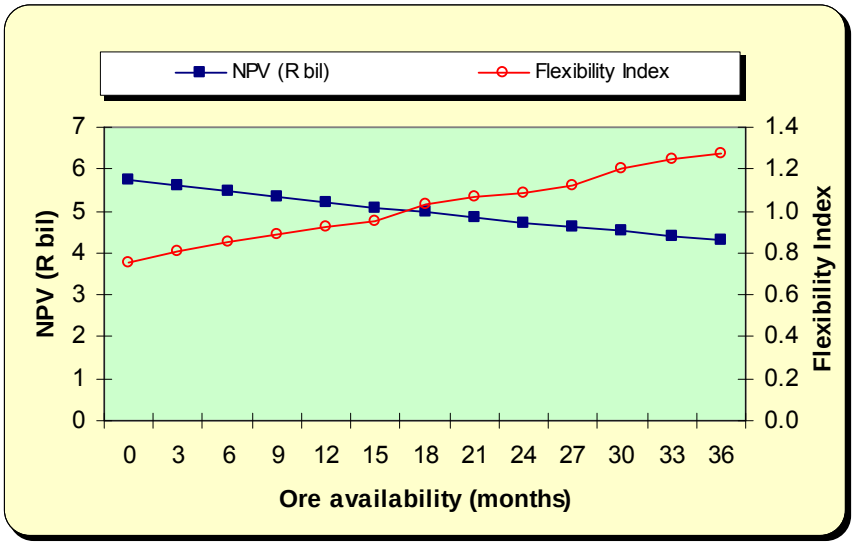


Figure 3.4: Relationship between NPV and flexibility Index

3.7 Concluding Summary

This chapter has defined technical operating flexibility and noted its increasing importance in mine planning. A metric, the Flexibility Index (*FI*), was derived as a unit of measure for technical operating flexibility. The *FI* methodology that was developed in this chapter is used in conjunction with the mine planning parameters discussed in Chapter 5 to analyse different layouts based on a typical UG2 orebody model that is described in the next chapter.

4 GEOLOGICAL OREBODY MODEL (OB1) DESCRIPTION

4.1 Introduction

As mentioned in Chapter 2, most previous studies on optimising level and raise spacing assumed constant geology throughout the entire orebody or completely ignored geological impacts. This assumption is inappropriate for the variable geological conditions that are actually encountered in mining on the Bushveld Complex. In order to overcome the shortcomings of assuming constant geology throughout the entire orebody, a geological orebody model based on real data from the Bushveld Complex was used for this research study. Since the data constituted proprietary information, it was necessary to protect the actual identity of the orebody by masking the data in four ways. Firstly, only a data subset of the entire mineral resource dataset was extracted, but carefully chosen not to exclude the key Bushveld Complex geological features such as dykes, faults and potholes. Secondly, the orebody model was pseudo-named as Orebody 1 (OB1). Thirdly, the actual topography of the project area was not disclosed and a flat terrain was assumed. Lastly, the prill split was also not disclosed and a typical UG2 prill split was assumed for the orebody.

In choosing the orebody data to work with, a cue was taken from Vieira, Diering and Durrheim (2001) and Vieira (2003) where a hypothetical *Iponeleng* orebody, based on data typical of the Witwatersrand ultra-deep level mining environment, was used to compare four different ultra-deep mining methods for the *Deepmine* project. A similar approach was taken in this study but instead real geological data was used for an orebody that represents a typical UG2 mining environment for conventional breast mining on the Bushveld Complex. The design and scheduling considerations in Chapter 5 were applied to OB1 to generate the results presented in Chapter 6. A description of the orebody and its relevant properties are discussed in the next sections.

4.2 Overview of exploration work done to configure and delineate OB1

The exploration work to configure and delineate OB1 that had been done by the mining company that supplied the geological data included borehole drilling and assaying, strike and dip trenching, outcrop and field mapping, high resolution aeromagnetic surveys, land satellite imagery and aerial photography. In the current times, 3-D Seismics could also have been used but had been ruled out of the exploration programme on the basis of cost and the fact that the mine property was contiguous to other already known orebodies whose geology was well-known and could be extrapolated to help configure OB1. The data was captured and modelled in SABLE® and ARCGIS® software packages, and subsequently exported to Datamine® and Microsoft Excel® software packages for further geological and geo-statistical

modelling and evaluation. The exploration boreholes used to delineate and configure the orebody were drilled on a 400m x 400m grid pattern, each hole having a mother hole and three deflections. Hence, according to the South African Code for the Reporting of Exploration Results, Mineral Resources and Mineral Reserves (the SAMREC code), the resource could be categorised as a Measured Resource.

4.3 OB1 structural geology

OB1 is an outcropping UG2 chromitite reef located on the southern part of the eastern limb of Bushveld Complex in a fairly flat topography, since a flat terrain was assumed for proprietary reasons as indicated earlier on in Section 4.1. The orebody strikes roughly north-south for about 4 km and has an average dip of 9.6° (Figure 4.1). As Figure 4.1 indicates, the orebody is traversed by dykes, faults and potholes. Figure 4.1 is a Mine 2-4D® illustration of the Datamine® files *ug2faults, ug2dykes, ug2potholes* and *boundary* contained in the directory *OB1* on the USB memory stick. Another key geological feature is the oxidised zone which extends for about 30m below surface (Figure 4.2). No mining will take place in the oxidised zone for two main reasons. Firstly, the strength of the ore material is weakened by oxidation resulting in geo-technically unstable ground if mining were to be carried out in the area. Secondly, oxidised ore is very difficult to treat and metallurgical recovery factors tend to be low, sometimes even as low as 40% compared to metallurgical recovery factors in excess of 80% for normal un-oxidised ore (Mahlangu, 2007). The oxidised zone was therefore excluded in calculating the mineral resource.

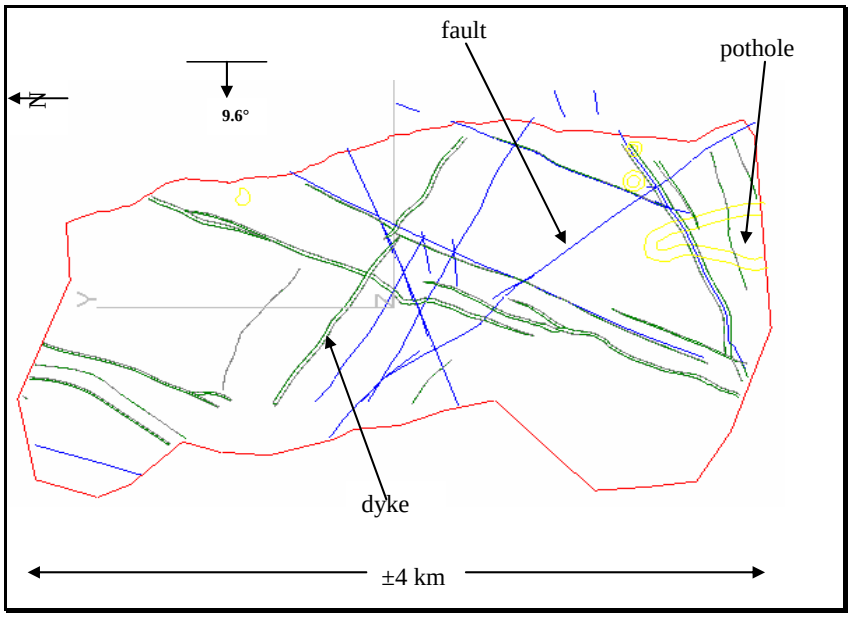


Figure 4.1: A Mine 2-4D® representation of OB1 structural geology

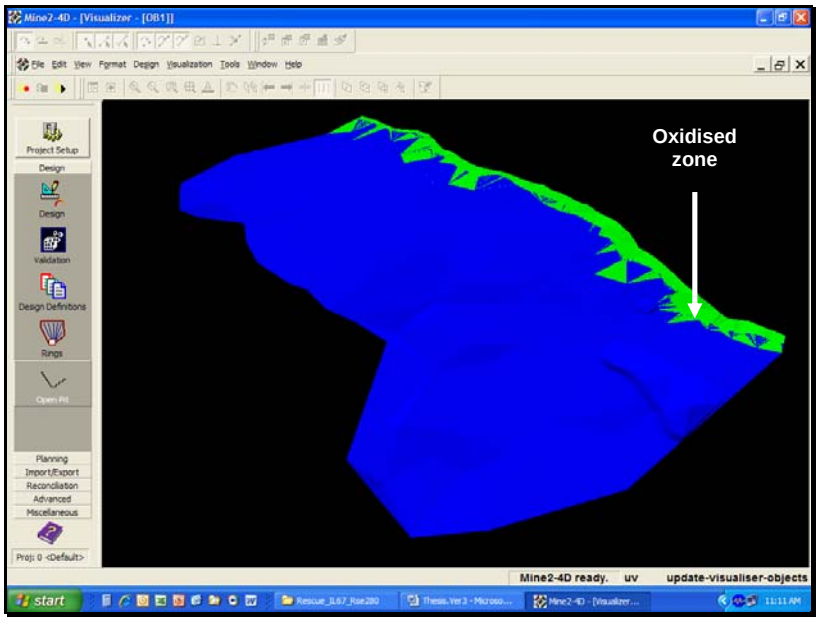


Figure 4.2: A Mine 2-4D® illustration of the oxidised zone (in green) extending for about 30m below surface

4.4 OB1 flat topography assumption

As mentioned earlier on in Section 4.1, a flat topography was assumed for OB1. This flat topography was supplied as the Datamine® wireframe file *topcut* and is included in the USB memory stick for reference. When *topcut* is read into Mine2-4D together with the OB1 wireframe file, their relative positions are as shown in Figure 4.3.

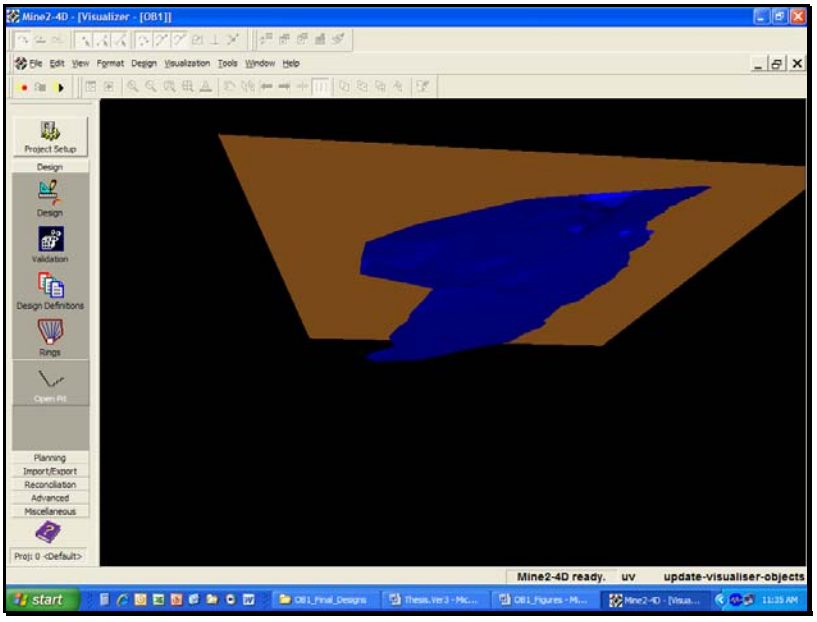


Figure 4.3: A Mine2-4D® illustration of OB1 (in blue) overlain by a flat topography (in brown)

4.5 Prill split

The term ‘prill split’ is used in platinum mining to indicate the relative proportions of the various PGE elements contained in a tonne of platinum ore as determined by an assay analysis of a PGE prill. The prill split is classified as a 4E prill split if it reports on the elements platinum (Pt), palladium (Pd), rhodium (Rh) and gold (Au). A 4E prill split is sometimes called as a 3E+Au prill split. A 6E prill split reports on the elements Pt, Pd, Rh, Iridium (Ir), ruthenium (Ru) and Au. This prill split is alternatively referred to as a 5E+Au prill split. Mining companies have traditionally reported 4E prill splits in the pre-2009 era but some companies have now started reporting on 6E prill splits when reporting resources and reserves because the income derived from the other two elements is also becoming significant such that it should not be left out of public reporting. Additionally, base metals copper (Cu) and nickel (Ni) contained in PGM ores, are reported separately as % grades. Each deposit has a unique prill split. Generally, UG2 ore contains a higher proportion of platinum and lower proportion of palladium compared to Merensky reef. Typical average prill splits reported by Impala Platinum pre-2009 and in 2009 are shown in Figure 4.4 and Figure 4.5, respectively.

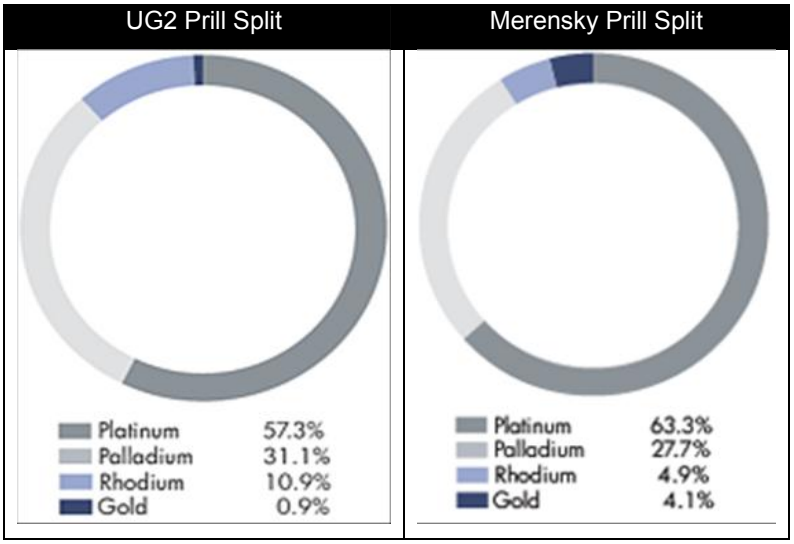


Figure 4.4: Typical average 4E prill splits for UG2 and Merensky reefs
(Source: Impala Platinum, 2008; Impala Platinum, 2009)

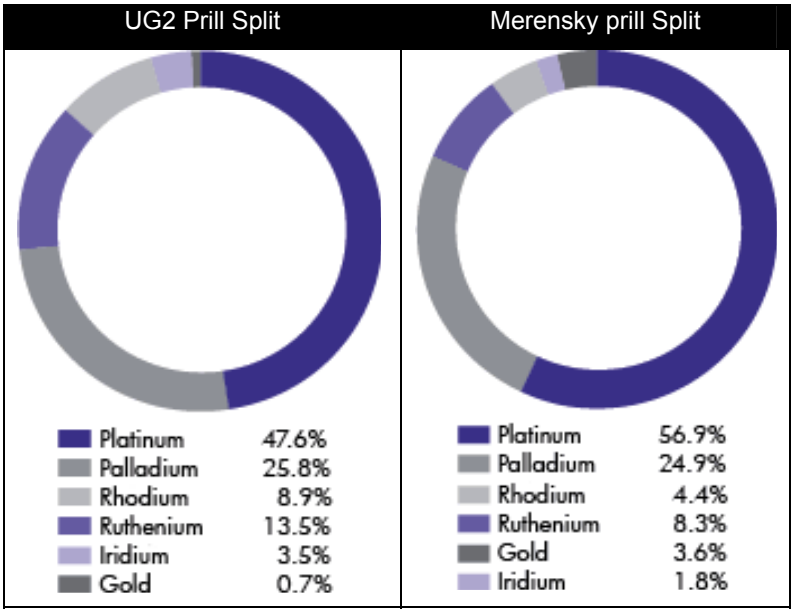


Figure 4.5: Typical average 4E prill splits for UG2 and Merensky reefs (Source: Impala Platinum, 2008; Impala Platinum, 2009)

Since the geological data that was provided reported the PGE grade as a 4E grade and the prill split had to be assumed, it was considered to be appropriate to take the Impala Platinum 4E prill split as the prill split for this research study. A quick check of Figure 4.4 shows that the total UG2 prill split is 100.2% and for Merensky is 100%. The discrepancy on the UG2 prill split can be attributed to rounding up errors. In order to correct this discrepancy for a total of 100% prill split, the Rh and Au components were adjusted to 10.8% and 0.8% respectively as shown in Table 4.1. This is the prill split applied in the cashflow model in Chapter 5.

Table 4.1: UG2 prill split assumed for OB1

PGE element	*Prill Split*
Pt	57.3%
Pd	31.1%
Rh	10.8%
Au	0.8%
Total prill split	**100.0%**

4.6 OB1 geological block model

At the time when the data was obtained the entire OB1 orebody had been drilled and all boreholes had intersected the UG2 reef. However, assay data was only available for boreholes covering only 43 blocks each measuring 400m x 400m. The 3-D block model file based on these blocks was provided as the Datamine® file, *modpge7c*, which is included on

the USB memory stick which has been supplied together with this thesis. In order to help the reader to interpret the codes used for the key headings in the file *modpge7c*, that is relevant to this study, Table 4.2 was compiled to show the meaning of the codes used to represent the key variables that were analysed for OB1. Table 4.3 shows the summary statistics for reef thickness, grade and density for each of the blocks making up OB1.

Table 4.2: Meaning of codes for OB1 key variables

Code	Meaning
PRP0000	Thickness of reef intersection in metres (m)
PGE0000	4E grade or PGE grade in grams per tonne (g/t)
ACC0000	Accumulation content in cm-g/t
CU0000	%Copper (Cu) content
Ni0000	%Nickel (Ni) content
DEN0000	Density in t/m^3
%ER	% error on an estimate of ACC0000
RECORD	Block number
(N)	Implies number format
XC(N); YC(N); ZC(N)	X, Y and Z coordinates of the centre or centroid of a block
XINC(N); YINC(N); ZINC(N)	Length or thickness of a block along the X, Y and Z axes
XMORIG(N); YMORIG(N); ZMORIG(N)	Implied XYZ origin for the block model with respect to the corner of the first block not its centroid
IJK	Unique code allocated to each block by Datamine with respect to the implied origin used to sort blocks during Datamine processes

Table 4.3: Summary statistics for OB1

BLOCK NO	XINC (N)	YINC (N)	PGE0000 (N)	PRP0000 (N)	DEN0000 (N)
1	400	400	6.47	0.99	3.91
2	400	400	6.04	1.13	3.97
3	400	400	5.97	1.36	3.83
4	400	400	6.05	1.26	3.94
5	400	400	6.06	1.21	3.91
6	400	400	5.09	1.30	4.00
7	400	400	5.78	1.30	4.03
8	400	400	6.62	0.98	4.14
9	400	400	7.32	0.88	4.20
10	400	400	6.59	0.95	4.06
11	400	400	6.17	1.08	3.90
12	400	400	6.46	1.03	4.12
13	400	400	6.67	1.05	4.12
14	400	400	6.37	1.06	4.08
15	400	400	6.06	1.17	4.00
16	400	400	5.53	1.27	3.93
17	400	400	4.95	1.20	3.95
18	400	400	5.78	1.11	3.93
19	400	400	6.75	1.04	4.04
20	400	400	7.02	0.92	4.08
21	400	400	6.57	0.88	4.01
22	400	400	6.54	0.95	4.11
23	400	400	6.35	1.02	4.12
24	400	400	6.25	1.20	4.12
25	400	400	5.91	1.21	4.01
26	400	400	6.18	1.06	4.06
27	400	400	5.51	1.09	3.96
28	400	400	5.44	1.09	3.78
29	400	400	5.68	1.12	3.65
30	400	400	6.43	1.19	3.79
31	400	400	6.24	1.05	3.92
32	400	400	6.04	0.98	3.98
33	400	400	6.22	0.97	3.99
34	400	400	5.90	1.23	3.95
35	400	400	5.35	1.55	3.95
36	400	400	5.53	1.40	4.00
37	400	400	6.64	1.11	4.03
38	400	400	6.94	1.08	3.98
39	400	400	6.35	1.18	3.99
40	400	400	6.42	1.13	3.99
41	400	400	6.23	1.05	3.98
42	400	400	6.24	1.03	3.97
43	400	400	6.37	1.21	3.96
Average	**400**	**400**	**6.1**	**1.1**	**4.0**
Standard Dev			**0.5**	**0.1**	**0.1**

Figure 4.6 is a Mine 2-4D® illustration of the geological block model in 2-D. Figure 4.7 and Figure 4.8 show an overlay of the block and wireframe models in 2-D and 3-D, respectively. The total surface area covered by the block model is about 6.88 million m^2 derived from 43 blocks each measuring 400m x 400m.

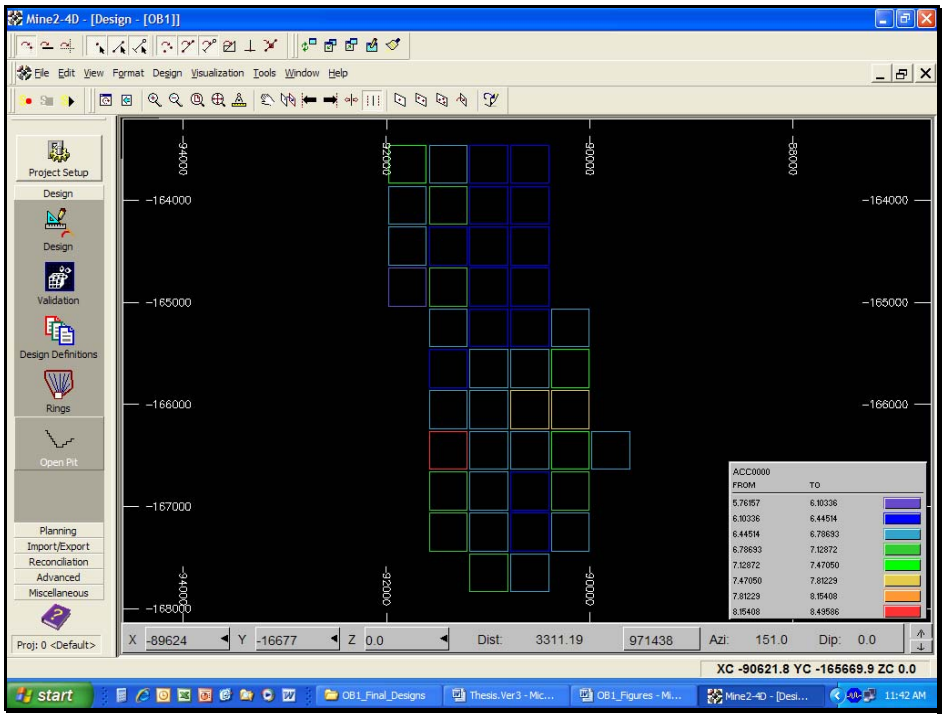


Figure 4.6: A Mine 2-4D® illustration of the OB1 geological block model in 2-D

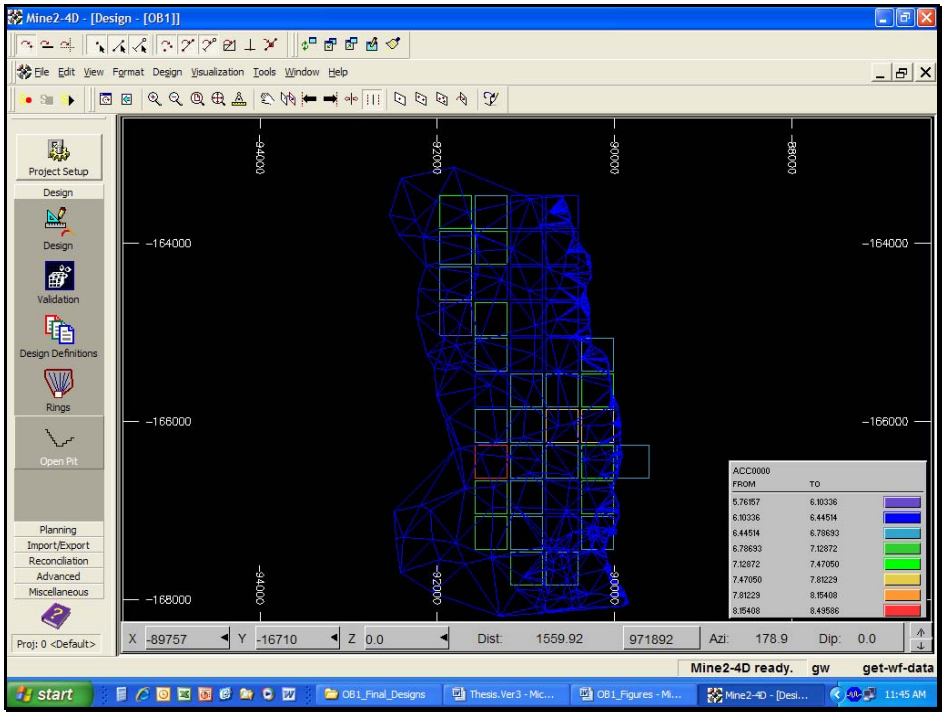


Figure 4.7: A Mine 2-4D® illustration of overlay of OB1 block and wireframe models in 2-D

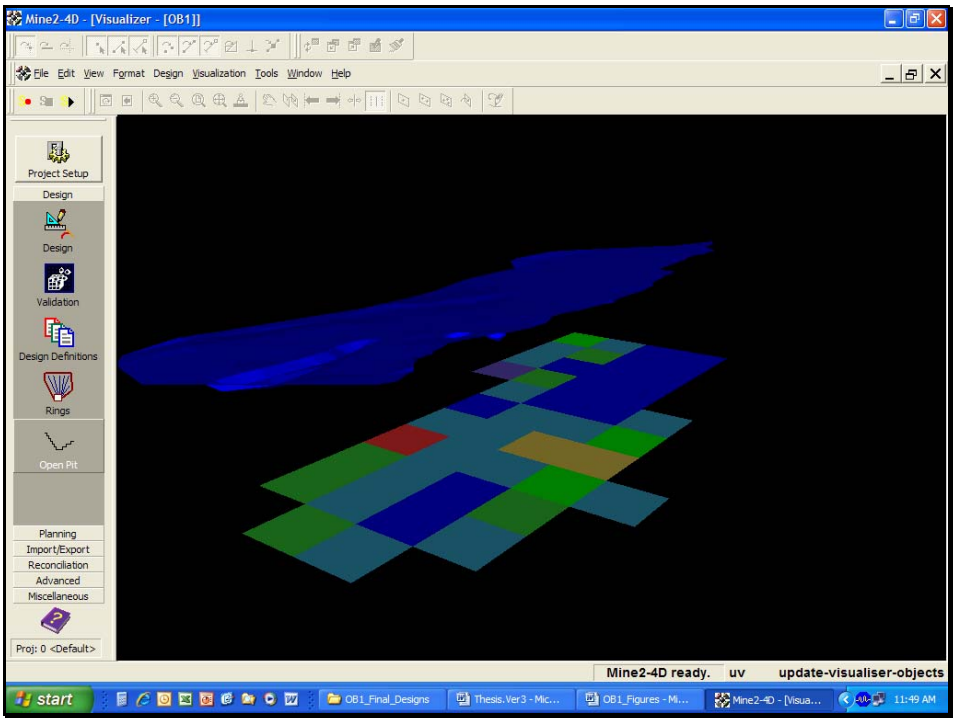


Figure 4.8: A Mine 2-4D® illustration of overlay of OB1 block and wireframe models in 3-D

Figure 4.8 shows the block model as a thin flat sheet of paper and the wireframe model as a slightly deformed inclined thin sheet of paper caused by geological disturbances. This is expected because the orebody, which is about 4km along strike, 2.5km along dip and only 1m thick when viewed in 3-D, will appear as a thin sheet of paper. This is the reason why exploration geologists familiar with the Bushveld complex geology simply report the block model as a 2-D model since the orebody is so thin that it can be approximated by a 2-D representation. The colour coding on the block model is to illustrate the grade ranges. As can be seen from Figure 4.8 the wireframe model on which the mining layouts were designed does not intersect the block model since it lies above the block model. In order to for the stopes designed on the wireframe model to extract values associated with their relative locations on the block model, the block model is stretched by incrementing the ZINC(N) value to overlap the wireframe elevation as shown in Figure 4.9. In this case the ZINC(N) was increased from 1m to 5,000m and the new file saved as the working block model file, *workmod*, which was subsequently used as the block model file during the stage when the design is evaluated against the block model in order to extract geological attributes such as grade, density and reef thickness for export to EPS® scheduler (Figure 4.10). The Datamine® file *workmod* is included on the USB memory stick.

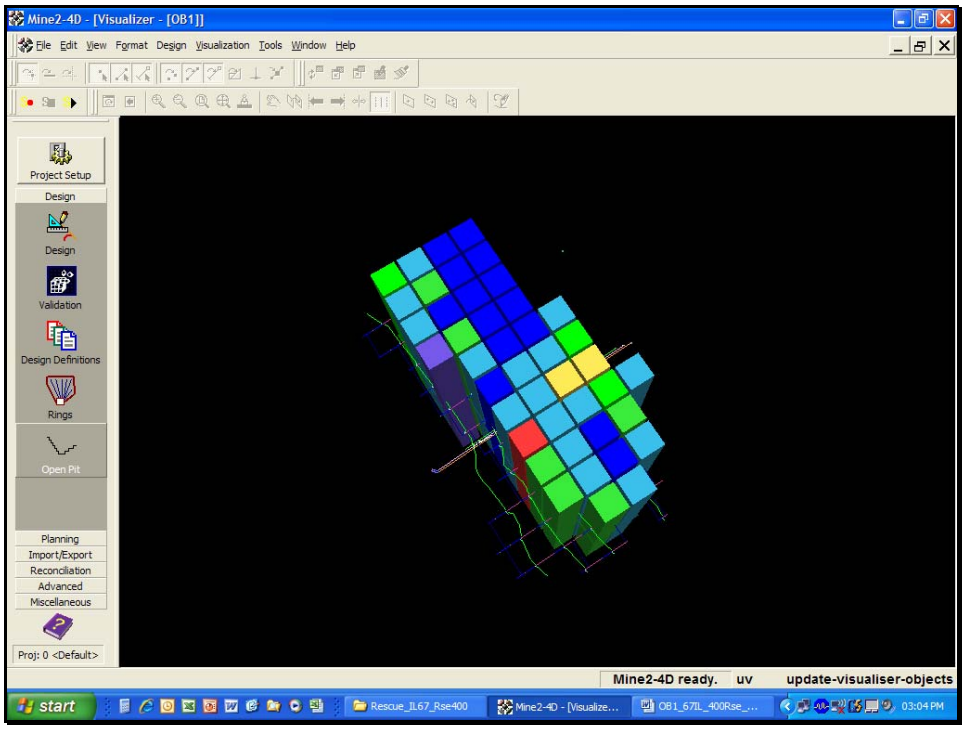


Figure 4.9: A Mine 2-4D® illustration of OB1 block model stretched to intersect the mine design on the wireframe model

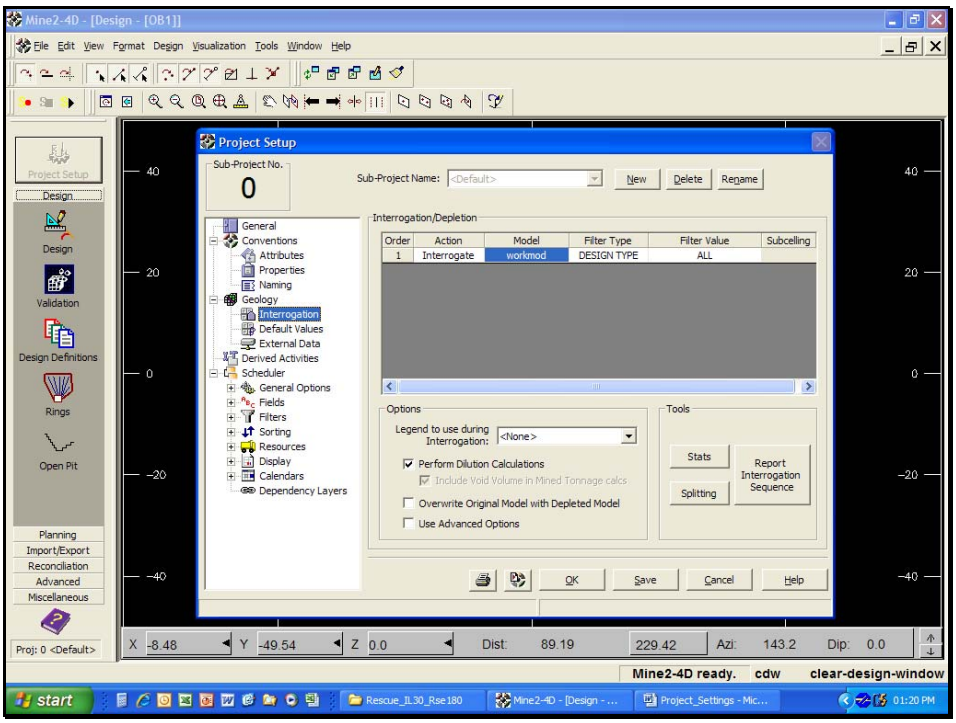


Figure 4.10: Mine 2-4D® project setup showing *workmod* as the default geological block model to be used interrogated when evaluating designs

4.7 OB1 grade-tonnage curve

Mine 2-4D® has a functionality to perform grade-tonnage curve calculations on a block model. Figure 4.11 is a grade-tonnage curve for OB1 that was generated in Mine2-4D® and subsequently exported to Microsoft Excel®.

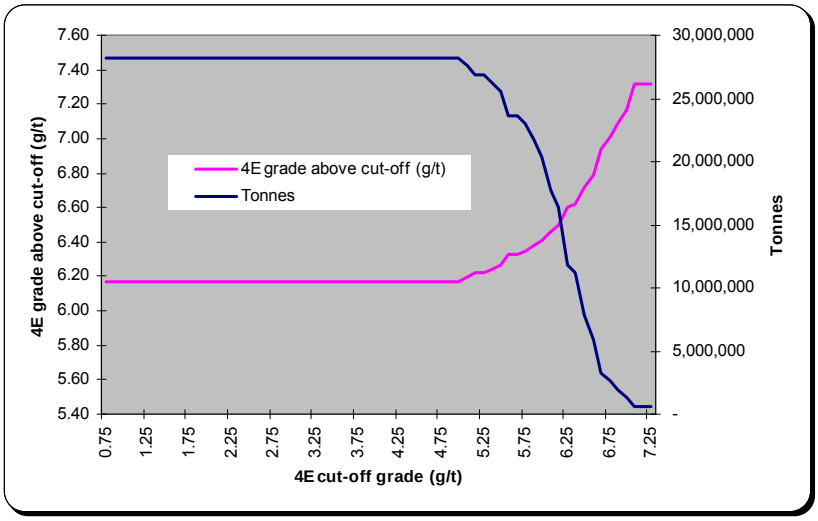


Figure 4.11: Grade-tonnage curve for OB1

The grade-tonnage curve calculations are contained in the Microsoft Excel® file *Grade_Tonnage_Curve_OB1* included in the USB memory stick. The OB1 *in-situ* mineral resource was estimated to be 28.208 million tonnes at the lowest 4E cut-off grade of about 0.75g/t over a mining cut of 1m exclusive of the triplet package since it was on average about 4.35m above the hangingwall of the UG2 chromitite reef. The grade-tonnage curve indicates that the average 4E grade above cut-off is not very sensitive to change in cut-off grade between 0.75g/t to about 4.95g/t. Therefore selective mining is not mandatory for OB1, except for negotiating or circumventing geological structural features such as potholes, dykes and IRUPs. This is consistent with the non-selective mining approaches used on the Bushveld Complex where selectivity is done mainly to negotiate or circumvent geological structures.

4.8 Summary of OB1 wireframe properties

Mine 2-4D® has functionality for evaluating wireframe properties. This functionality was used to determine the OB1 wireframe properties (excluding the oxidised zone) as illustrated by Figure 4.12.

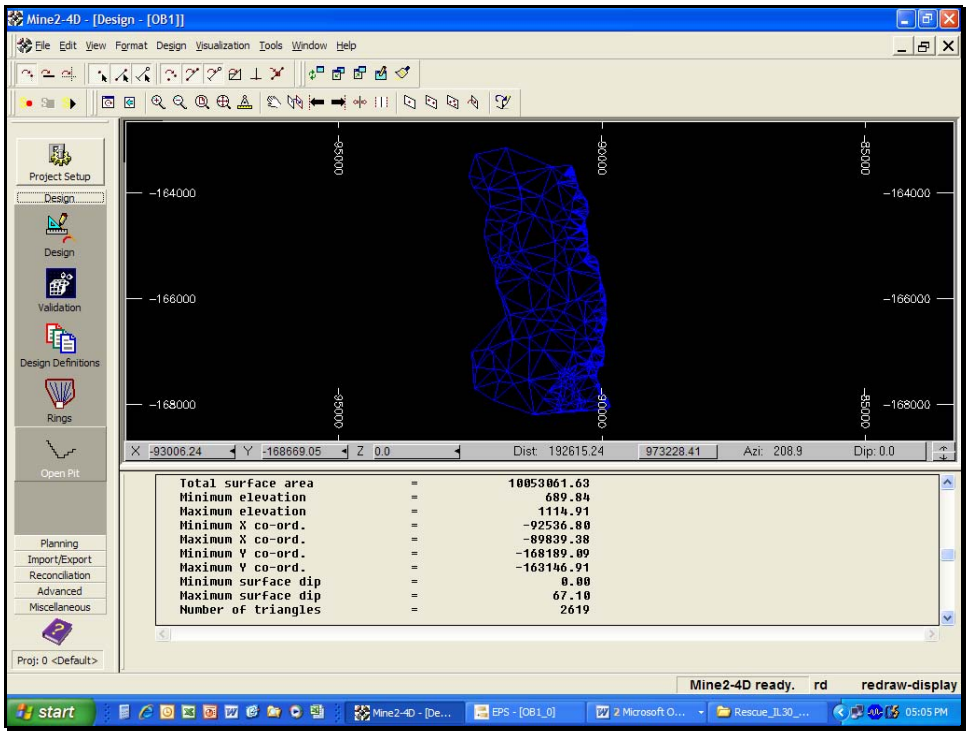


Figure 4.12: Summary of OB1 wireframe properties as evaluated in Mine 2-4D®

The inclined surface area of OB1 is about 10.053 million m^2 which is equivalent to 9,897,847.58m^2 when projected onto the flat surface defined by the assumed flat surface wireframe model *topcut*. The topmost part of OB1 boundary is at an elevation of 1,130masl and the lowest part is at 689masl. OB1 strikes for about 4km, with an inclined length of about 2.5km and has an average dip of 9.6º. It can be seen that the block model *modpge7c* only covered about 68% of the total OB1 wireframe surface area. Therefore some portions of designs made on the wireframe surface would fall outside the area covered by the block model and during an evaluation would therefore have to take on default values. Based on Table 4.3, an assumption was therefore made that the average values for grade (6.1g/t), thickness (1.0m) and density (4.0t/m^3) would be used as default values in Mine 2-4D® and EPS® evaluation of the layouts as shown in Figure 4.13. A default thickness of 1.0m was assumed although the average thickness is 1.1m, in order to minimise dilution and keep stoping height to 1.0m.

If the default values are applied to the OB1 wireframe model, then the *in-situ* tonnage (excluding the oxidised zone) is estimated to be 40.212 million tonnes at an average *in-situ* 4E grade of 6.1g/t.

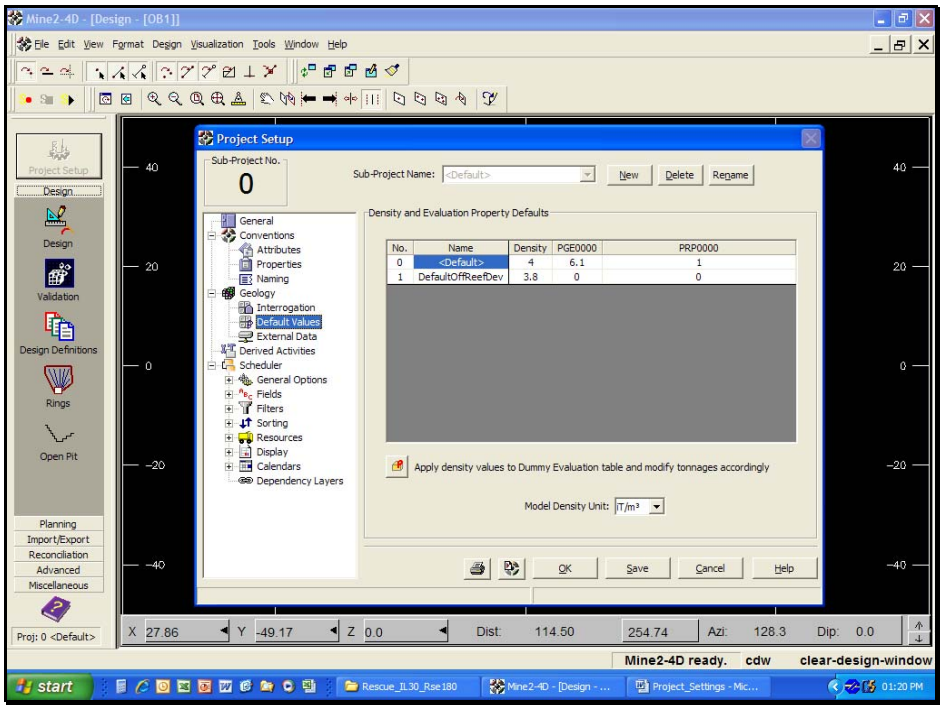


Figure 4.13: Default values assumed for OB1

4.9 OB1 Measured Resource (exclusive of geological losses and oxidised zone)

Table 4.4 summarises the key characteristics of OB1 as a Measured Resource excluding the oxidised zone and geological losses due to dykes, faults and potholes. The geological loss for potholes, faults and dykes based on the geological structural models supplied was about 7.81% as indicated in the Microsoft Excel® file *TechnoEconAnalysis* included on the USB memory stick.

Table 4.4: Summary of the geological characteristics of OB1

Reef type	*In-situ* tonnage	Average *In-situ* 4E grade	4E cut-off grade	Average reef thickness	Average mining height
UG2 out-cropping	40.212mt	6.1g/t	0.75g/t	1.1m	1m

4.10 Summary

This chapter has described the characteristics of OB1 which are consistent with typical geological features of UG2 Bushveld Complex type reefs that were described in Section 1.5. The next chapter discusses the techno-economic assumptions that were applied to the orebody, OB1.

5 DESIGN AND SCHEDULING TECHNO-ECONOMIC ASSUMPTIONS

5.1 Introduction

As indicated in Chapter 4, OB1 was classified as a Measured Mineral Resource according to the SAMREC code. The SAMREC code also allows a pre-feasibility level of study to be done on a Measured Mineral Resource, but the overall confidence of the study must be stated, which should be at a lower level of confidence than that of a feasibility study. Johnson and McCarthy (2001) indicated that typical levels of accuracy in estimates for a bankable feasibility study should be about ±10%-20%, while Pincock (2004) mentioned that typical levels of accuracy for a bankable feasibility study should be about ±15%. It was appropriate therefore, to assume a level of accuracy of about ±25% for this research study since it falls into the pre-feasibility category. This is the reason why some of the design, scheduling and financial assumptions presented in this chapter include Control Budget Estimates (CBE) and Engineer, Procure, Construct and Manage (EPCM) contractor cost estimates.

This chapter explains the engineering basis for the key inputs for the design and scheduling process followed for the 15 laybye access conventional breast mining layouts that were developed. The reasons for choosing the laybye access layout were given in Section 1.7.3. This chapter firstly discusses the design process used, followed by the mining, metallurgical and economic (or financial) assumptions made in developing the OB1 layouts and schedules. The layouts for OB1 were designed in Mine2-4D® and subsequently scheduled in Enhanced Production Scheduler® (EPS®). The final techno-economic analysis on the design and scheduling data generated was then performed in Microsoft Excel®. The results of these analyses are presented in Chapter 6.

5.2 Choice of Mine2-4D® and EPS® software and overall design process

This section explains why Mine2-4D and EPS software were chosen for executing the designs for this research study. It also explains briefly the design process followed in carrying out the designs, schedules and further analysis done to derive the optimal range of level and raise spacing in conventional breast mining layouts. These two aspects are discussed in the next two sub-sections.

5.2.1 Choice of Mine2-4D® and EPS® software

As discussed previously in Chapters 1 and 2, the optimisation of level and raise spacing is a long-term strategic mine planning exercise. Therefore, the designs and schedules had to be done at Life-of-Mine (LOM) resolution. This meant that extreme details such as refuge bays

and explosive cluster drum cubbies along haulages would not be captured in the designs. However, details such as Advanced Strike Gullies (ASGs) which are not reflected in the designs were catered for in EPS® using mathematical relationships between centares mined and corresponding ASG metres developed, as illustrated in the Microsoft Excel file *OB1_Grade_Equation_Model*. The main reasons for choosing Mine2-4D® and EPS® as the software for doing the designs and schedules are as follows:

- The Mine2-4D® and EPS® software suite is the standard software used by platinum mining companies operating on the Bushveld Complex and to some extent by gold mining companies on the Witwatersrand basin for long-term mine planning. Additionally, consulting companies such as GijimaAST Mining Solutions, Ukwazi Mining, Snowden Group and TWP Consulting that undertake feasibility studies for platinum mining companies also use this software for designing and scheduling. Short-term mine planning is typically executed using CADSMine®.
- GijimaAST, the South African software agent for Mine2-4D® and EPS® were willing provide educational training licences and competency training for the software over the entire duration of the research study.
- Mine2-4D is able to read in data files that are in Datamine file format. As indicated in chapter 4, the geological data that was provided for this study was in Datamine file format.
- Since leach layout represented a different mining scenario, it was necessary to use software that can easily allow different scenarios to be evaluated using the same initial raw data. Mine2-4D and EPS have the capability of allowing different scenarios to be evaluated though using template files and swap files.
- EPS is seamlessly integrated with Mine2-4D to allow schedules to be generated from designs and planning parameters set in Mine2-4D. EPS also provides further flexibility in that these parameters can be altered within EPS and the changes exported back to Mine2-4 for synchronisation.

5.2.2 Overview of design and scheduling process and files

The design process followed was the engineering circle or wheel of design developed by Stacey (2006) and Stacey *et al* (2007) but adapted to suit the requirements of this study as shown in Figure 5.1, since this study had no implementation stage. This design process is reflected by some of the industry comments contained in Appendix 10.3 such as (Anglo Platinum Review Committee, 2009), *"The stakeholders were consulted well"*, and, *"The different analysis techniques were well sourced"*.



Figure 5.1: The engineering circle or wheel of design
(Adapted from Stacey, Terbrugge and Wesseloo, 2007)

This section gives an overview of how the design and scheduling files are linked and process followed in applying the design and scheduling parameters explained in preceding sections to the geological data set described in Chapter 4 to produce the files. The directories and files referred to in this section are contained on the USB memory stick.

The files containing the geological data set that were obtained from industry are in the directory *OB1*. Table 5.1 indicates what each of the files contains.

Table 5.1: Geological data files obtained from industry

File	Description
modpge7c	Datamine file for the geological block model data showing the 43 blocks with the geological parameters as described in Table 4.2. The key geological parameters are summarised in the Microsoft Excel® file *OB1_Summary_Statistics*.
ug2dykes	Datamine string file containing dykes together with 2m bracket pillars abutting the dykes.
ug2faults	Datamine string file containing faults.
ug2potholes	Datamine string file containing potholes.
ug2oxidblue	Datamine wireframe file containing the reef horizon excluding the oxidised zone.
ug2oxid	Datamine wireframe file containing the reef horizon including the oxidised zone.
workomd	Datamine file obtained when modpge7c was stretched so that the ZINC(N) value was 5,000m in order to intersect the mine design string file.
OB1_Grade_Tonnage_Curve	Microsoft Excel® file containing the grade-tonnage calculations that were done in Mine2-4D and exported to Microsoft Excel®.
Directory 200_200	Directory containing all the designs and schedules done on OB1 for a level and raise spacing grid of 200m x 200m for purposes of calculating technical operating flexibility.

The directory *OB1_Final_Designs* contains 15 directories for each of the 15 layouts. The naming convention used for the directories is *Rescue_LevelSpacing_RaiseSpacing*. For example the layout at vertical level spacing of 30m and raise spacing of 180m was named *Rescue_IL30_Rse180*. Within each directory are Mine2-4D®, EPS® and Microsoft Excel® files. The OB1 structural geology file *proj_geology* in each of the 15 directories, was obtained by overlaying the files *ug2dykes*, *ug2faults* and *ug2potholes*. The fixed cross-sectional designs for development are contained in the Mine2-4D® file *design1_bh_rses*. The outlines design files for stopes, laybyes and pillars are contained in the Mine2-4D® file *outlines*. The design files were taken through the evaluation process in Mine2-4D and interrogated against the geological model *workmod* and subsequently sequenced before exporting to EPS. The EPS files obtained from this process are named *OB1_0* by the system. After the first design schedule was completed, the activities in *OB1_0* EPS® file were deleted and the file saved as the EPS® file template. The *template* file was then set up as the default file for exporting the remaining designs into EPS® to obtain the *OB1_0* files for the rest of the designs. Other files shown are either intermediate files that were saved by the author during the design and scheduling process or they are system-generated files. The EPS® *OB1_0* files were subsequently exported to Microsoft Excel® for final analysis in the files *TechnoEconAnalysis*. The output from each layout *TechnoEconAnalysis* file were aggregated and further analysed in the Microsoft Excel® file *Final_Criteria_Summary* contained in the directory *Ref_Docs*. Interpolations were made to fill data gaps since only 15 designs could be done within the

research study time. These interpolations and cross-interpolations are contained in the file *Final_Criteria_Summary* and explained in detail in Chapter 6. Finally the efficiency data for the 15 layouts were analysed using AHP methodology. The AHP analysis is contained in the directory *AHP_Survey_Analysis*.

5.3 Production rate and ramp-up profiles

Smith (1997), Bullock (2001) and McCarthy (2006) noted that the selection of the optimal production rates for most mining projects has relied on the rule-of-thumb by Taylor (1977) and Taylor (1986), which is known as Taylor's Law. Taylor's Law states that the life of a deposit is proportional to the fourth root of the expected ore tonnage. The transposed equivalent form of the law states that the optimum annual production rate in tonnes per year is proportional to the three-quarters power of that estimated ore tonnage (Taylor, 1986). Taylor (1986) gave the mathematical form of the law and its transposed equivalent as shown in Equation 5.1 and Equation 5.2.

$$Life\ in\ years = 6.5 \times \left(\frac{Expected\ Ore\ Tonnes}{10^6} \right)^{0.25}$$

Equation 5.1

$$Tonnes\ per\ year\ (350days) = 5.0 \times (Expected\ Ore\ Tonnes)^{0.75}$$

Equation 5.2

Equation 5.2 assumes 350 working days per year and the scaling factor of 5.0 has to be adjusted in proportion to the assumed working days per year for specific working circumstances. The South African mining industry works on a 48-hour week and annual mining production days excluding holidays are about 298 days [=6days/wk*52 weeks – 14 public holidays], say 300 days per year. Therefore, Taylor's Law has to be adjusted to suit the South African working calendar. For example, Vermeulen (2006) observed that Taylor's Law tends to over-estimate the production rate by about 30% for most tabular platinum reef deposits of the Bushveld Complex and adjustments have to be made accordingly. Vermeulen's observation confirms Taylor's comment that the law cannot be directly applied to Witwatersrand type deposits or other inclined tabular or massive deposits that are mined at great depth. The main reason for this exception to the rule is that the predicted production output is less than that observed in reality due to constraints imposed by hoisting capacities and achievable rates of radial or lateral development from the shaft or decline access. Accordingly, Taylor's Law was applied and adjusted down by 30% to estimate the production rate for OB1 to be about 100,00tpm as shown in Table 5.2.

Table 5.2: Application of Taylor's Law to estimate production rate for OB1

In-situ resources	Area extraction	Resources adjusted for area extraction	Local geological & mining losses	Expected ore tonnes	LOM	Taylor's prod rate	Taylor's prod rate	Planned production rate
40.212mil.t as estimated in Chapter 4	81% exclusive of local geological and mining losses as explained in Chapter 6	32.6mil.t	30% as explained in Chapter 6	22.8mil.t after adjusting for local geological and mining losses	14.2 years, obtained using Equation 5.1	1.6mtpa	133,333tpm	93,333tpm (say 100,000tpm), obtained by downward adjustment of Taylor's rate with 30%

The derived production rate for OB1 lies within the range of 70,000tpm-300,000tpm for most Bushveld Complex platinum mines as noted from the Hey (2003) study of some Anglo Platinum mines. The production rate of 100,000tpm is equivalent to a monthly stoping rate of about 25,000m^2 assuming a tonnage factor of 4t/m^2 since the density of OB1 is about 4t/m^3 and the stoping width assumed is 1m. A typical Impala Platinum conventional mining shaft usually has about 14 producing half-levels, and Impala best practice requires each half-level to produce at least 2,000m^2 per month equating to 28,000m^2 for the 14 half-levels Mohloki (2007). At a stoping height of 1m and density of 4t/m^3, the production rate is about 112,000tpm, which is quite comparable to the production rate derived for OB1 above. Additionally, it is also standard practice to plan for waste at ±20% of total material hoisted for UG2 and Merensky reefs (Vermeulen, 2006).

The study by Hey (2003) also analysed production rate build-up (i.e. ramp-up) periods and noted that conventional mining and hybrid ramp-up periods ranged between 19months-43months. The fastest build-up period observed for the Waterval shaft, using mechanised mining, was ascribed to the simultaneous sinking of a trackless two-decline cluster combined with the use of trackless room-and-pillar.

5.4 Access development

The primary functions of development in underground mines are to (Fleming, 2002):

- Provide access to the orebody.
- Delineate the orebody into manageable sections.
- Generate additional geological information for evaluating the orebody.
- Prepare the orebody for subsequent extraction (i.e. stoping or production).
- Provide a network of arteries for the transport (of ore, waste and material) and movement of services (e.g. water drainage; compressed air; service water for machines, drinking or for chilling; electricity; ventilation; and backfilling), into and out of the mine.

Typically development excavations cost more per m^3 of rock excavated compared to stoping excavations because in stoping there is a free breaking face, while in development the free breaking face has to be created by initially blasting out a cut or drilling a large diameter relief hole in nearly the centre of the excavation. This is the reason why it is necessary optimise development planning by minimising the amount of development. Generally, on narrow tabular reef mines using conventional mining methods, every 1,000m^2 of stoping requires about 20m-50m of development (Fleming, 2002), implying that replacement factors can be expected to range between 20m^2/m-50m^2/m. Development access to the orebody is generally divided into capital development, off-reef primary development, and on-reef secondary development. Planning parameters for the three categories of development are discussed in the next sub-sections.

5.4.1 Capital development

Capital development is excavated mainly in the footwall waste and occasionally in the hangingwall waste, to provide initial or primary access to the orebody. Capital development excavations are usually planned to last for the Life-of-Mine (LOM). Capital development design should always consider factoring-in initial over-capacity if no significant additional expenses can be incurred, so that the capacity is never less than that needed for optimum operation and can cope with any possible future increases in the scale of operations (Fleming, 2002). This philosophy is consistent with Macfarlane's (2005:187) argument that an optimal ore extraction path can be created by removing operating constraints at mine design and planning stage because, *"where flexibility to deal with changing economic cycles has not been created, (as a value-adding decision) reactive planning has to be undertaken, which is value-destroying"*. Examples of capital development include shafts and declines, and their ancillary infrastructure.

Chapter 4 noted that the elevation of OB1 ranged between 1,130masl-689masl to give a total vertical depth of about 441m below surface. It was therefore appropriate in this research study to select decline access in preference to vertical shaft access because the economically optimal change-over depth to switch from decline access to vertical shafts is about 1,000m (McCarthy and Livingstone, 1993; Elevli, Demirci and Dayi, 2002). This is mainly due to the fact that declines provide quicker and early access to the reef horizon resulting in early generation of project cashflows compared to vertical shaft systems. In addition, primary access for some of the most recent platinum projects has been by decline clusters. For example Jagger (2006) reported that the Impala Platinum No. 20 Shaft project was planned on three declines; one as a chairlift decline for man transportation, the second one as a material decline equipped with a monorail for material handling and the third as a

conveyor decline for ore transportation. It was therefore decided that OB1 would be accessed by a three-decline cluster (material decline, conveyor decline, and chairlift decline) since there would be no dedicated intake or return airway (Figure 5.2). A chairlift system is illustrated in Figure 5.3.

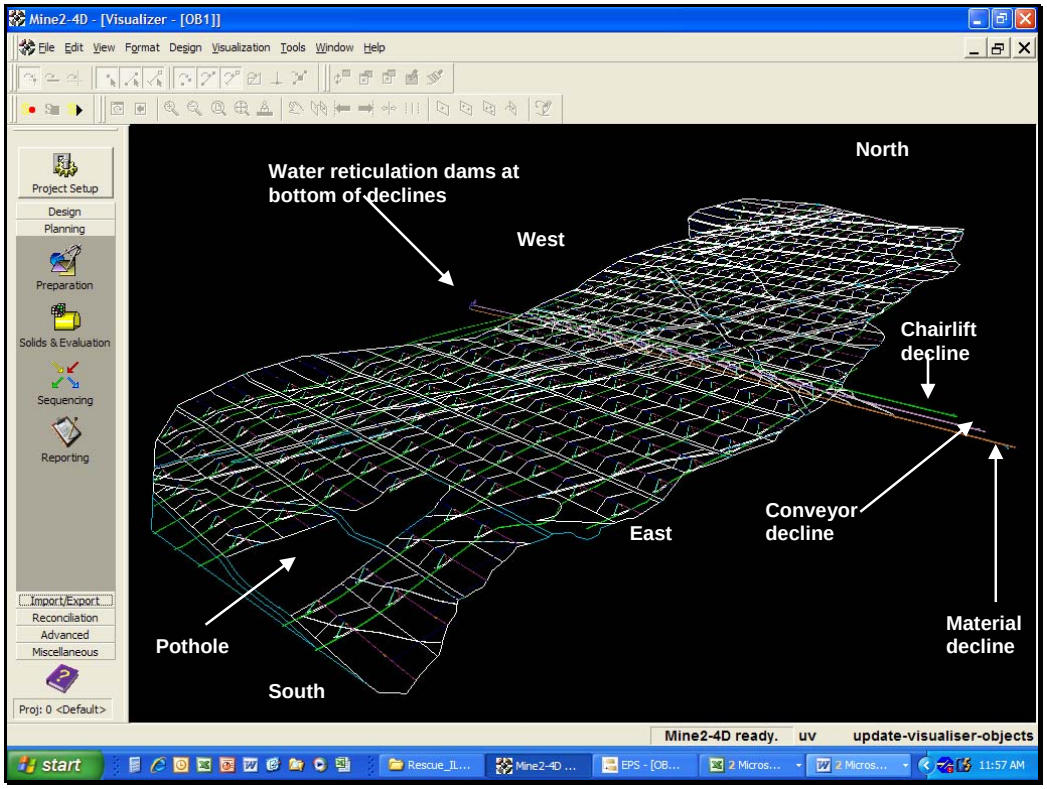


Figure 5.2: Relative positions of the three-cluster decline system for OB1 in Mine2-4D®

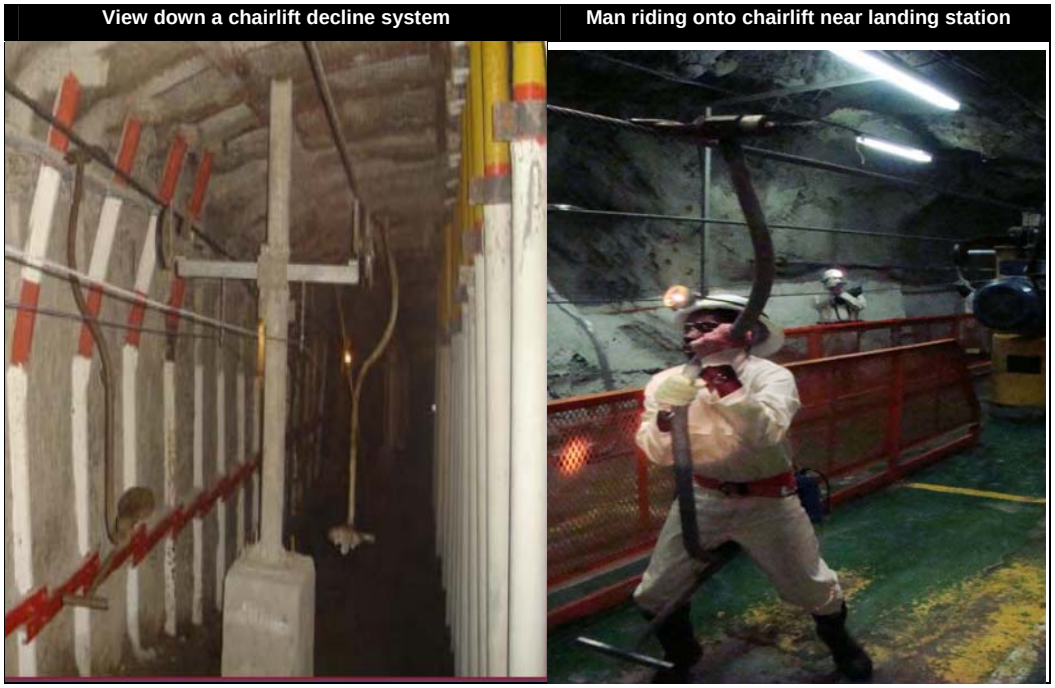


Figure 5.3: Photos taken underground illustrating a chairlift decline

Jager and Ryder (1999) proposed that footwall development in hard rock narrow reef mines should be located not closer than 30m below the reef horizon to avoid interaction with stope stress fracturing and where the development is closer than 30m, support design must cater for increased stress fracturing. It was therefore decided that the decline cluster would be carried on true dip but offset into the footwall by at least 40m for rock engineering stability. Budavari (1983), and Jager and Ryder (1999) observed that the localised stress concentration around an excavation rises sharply and peaks immediately adjacent to the excavation but decays to the field stress at a distance that is approximately two diameters from the centre of an excavation. Jager and Ryder (1999) further noted that in order to avoid stress interactions between adjacent excavations from reaching damaging levels, it is recommended that the excavations are sited no closer than twice their diameters apart when measuring the centre-to-centre distance. In this study it was therefore decided to space the declines at 24m centres both on dip and strike since the design environment is expected to be a low-stress environment due to the shallow mining depths.

In order to minimise haulage costs and quickly expose production faces to increase operating flexibility, it is preferable to locate the primary access as close to the centre of gravity of the deposit as possible. Zambó (1968) used a cost function to investigate the optimal shaft location for a steeply dipping reef and observed that the optimal location was along a line through the centre of gravity of the orebody but offset some distance from the

centre of gravity. McIntosh Engineering (2000:82) noted a similar rule-of-thumb that, *"the normal location of the shaft hoisting ore (production shaft) is near the center of gravity of the shape of the orebody (in plan view), but offset by 200 feet or more"*. Some of the platinum mines have also used this rule-of-thumb as shown by the location of declines at Boschfontein shaft (Figure 5.4). It was therefore decided to locate the OB1 decline clusters close to the centre of gravity of the orebody (Figure 5.5). The centre of gravity of OB1 shown in Figure 5.5 was determined using a Mine2-4D® function that calculates the centre of gravity. Figure 5.6 illustrates the water reticulation infrastructure that was designed for at the bottom of OB1.

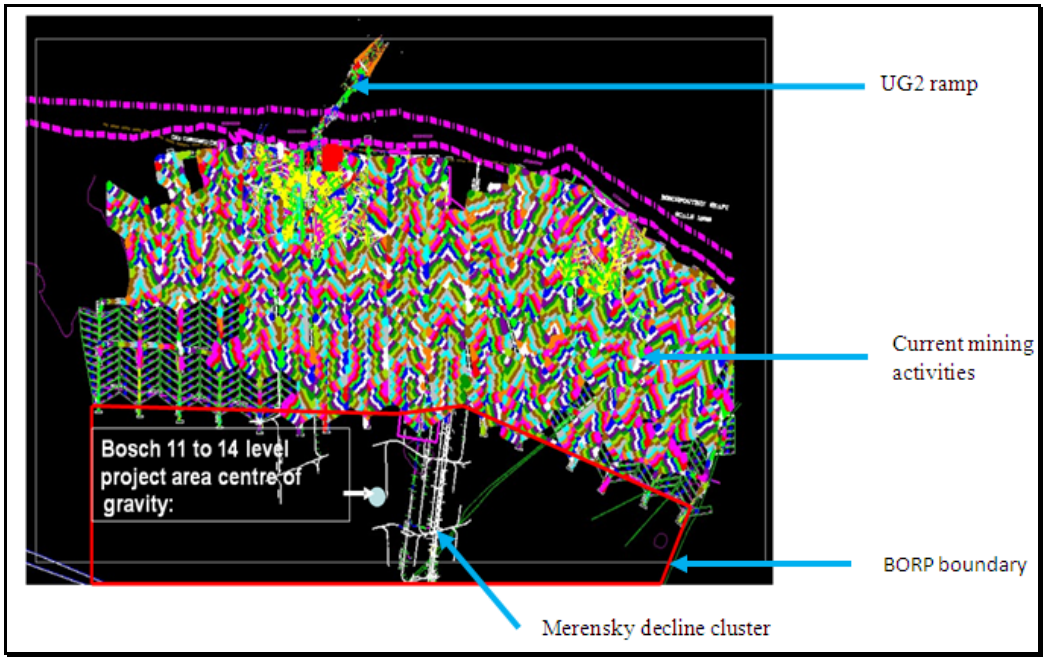


Figure 5.4: Location of Merensky reef decline cluster at Boschfontein shaft illustrated in Mine2-4D®
(Courtesy of Anglo Platinum, Boschfontein Shaft, Geology Department, 2004)

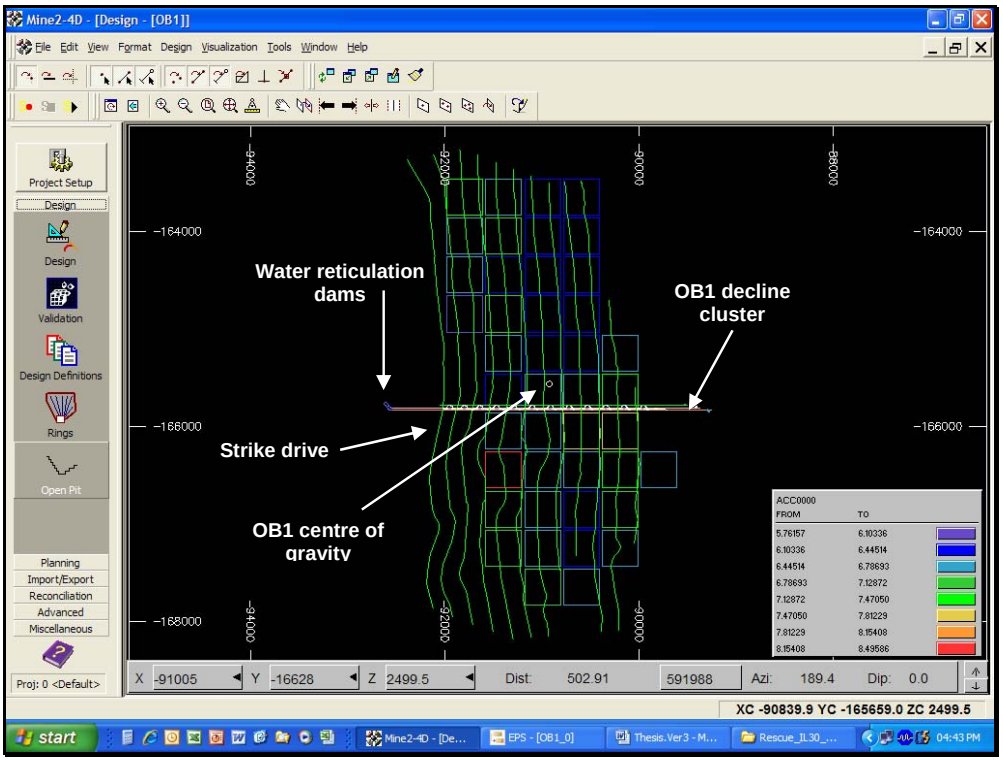


Figure 5.5: Location of OB1 decline cluster relative to centre of gravity in Mine2-4D®

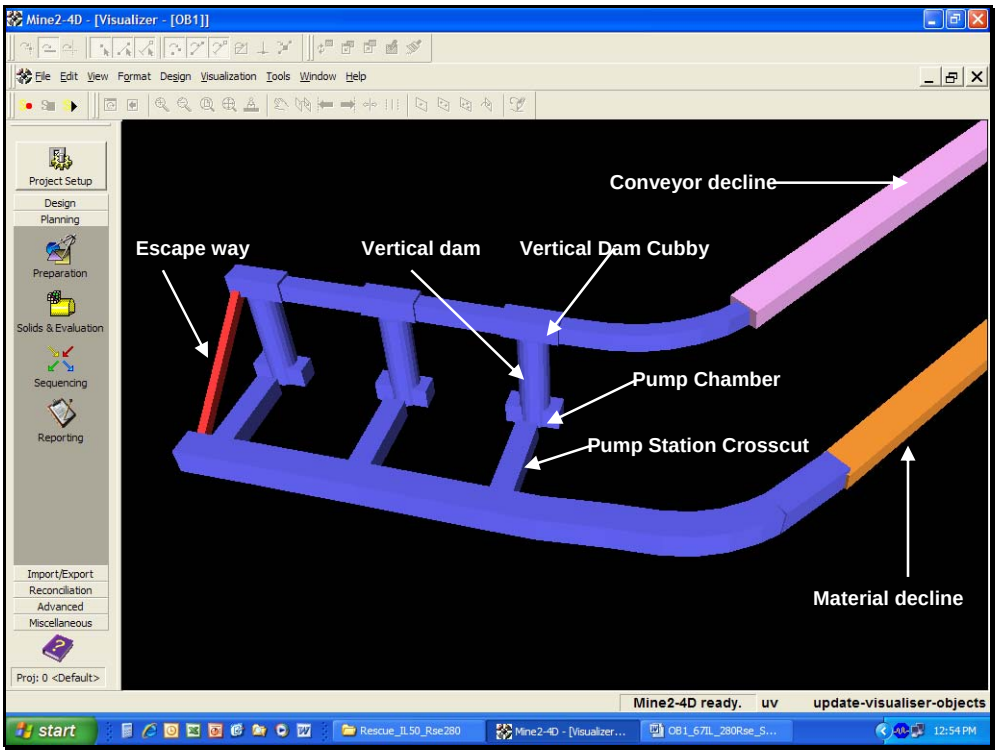


Figure 5.6: Mine2-4D® illustration of water reticulation infrastructure at bottom of OB1

Declines on the Bushveld Complex are typically 5m-6m wide by 3m-5m high. The height is dictated by headroom requirements for trackless equipment used for sinking the declines and the width by geotechnical considerations since wider declines are more difficult to support and maintain. Jager and Ryder (1999) state that as a general rule, service excavations should be made as small as possible but still small enough to cater for anticipated operational requirements since larger excavations expose more volume of rock mass to induced stress and structural discontinuities resulting in thicker unstable rock mass surrounding the excavation. For example, the Impala Platinum (2007a) standards stipulate that excavations larger than 6.0m wide by 4.5m high should not be developed underground without written permission of the Rock Engineering Manager and General Manager Mining, who must first assess the excavation's stability. It was decided to use the wider dimensions of 6m x 4m since the OB1 declines are planned to also serve as ventilation intake and exhaust airways. In addition each decline would have a split near surface (Figure 5.7) for use as intake or exhaust shafts fitted with ventilation fans.

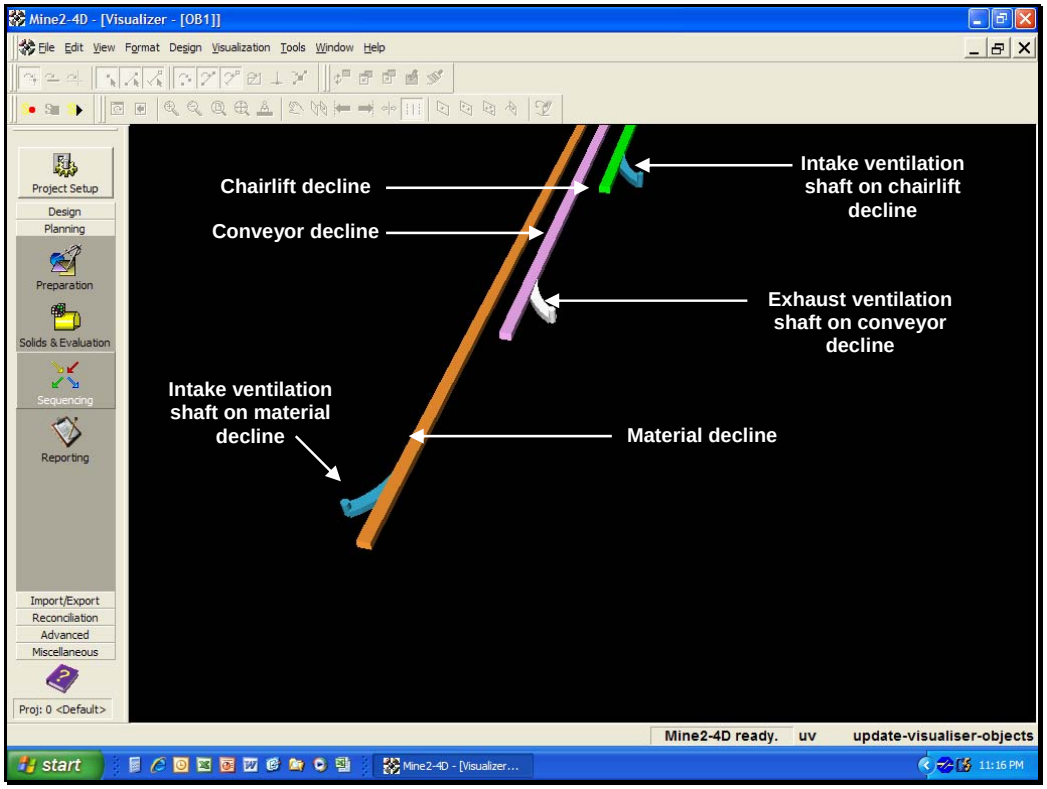


Figure 5.7: Mine2-4D illustration of intake and exhaust shaft splits on declines near surface

Rupprecht (2006) benchmarked decline development rates based on actual projects in the South African mining industry and noted that when the drilling, blasting, cleaning and other associated activities are optimised, a rate of 80m/month was appropriate and achievable, particularly when using high speed mechanised development methods. This is the decline

development rate used in this study. It was also assumed that the development of declines would be out-sourced to Engineer, Procure, Construct and Manage (EPCM) contractors; therefore manpower planning excluded manpower for decline development. Figure 5.8 is a summary of the typical industry dimensions and mining rates assumed for OB1 development excavations.

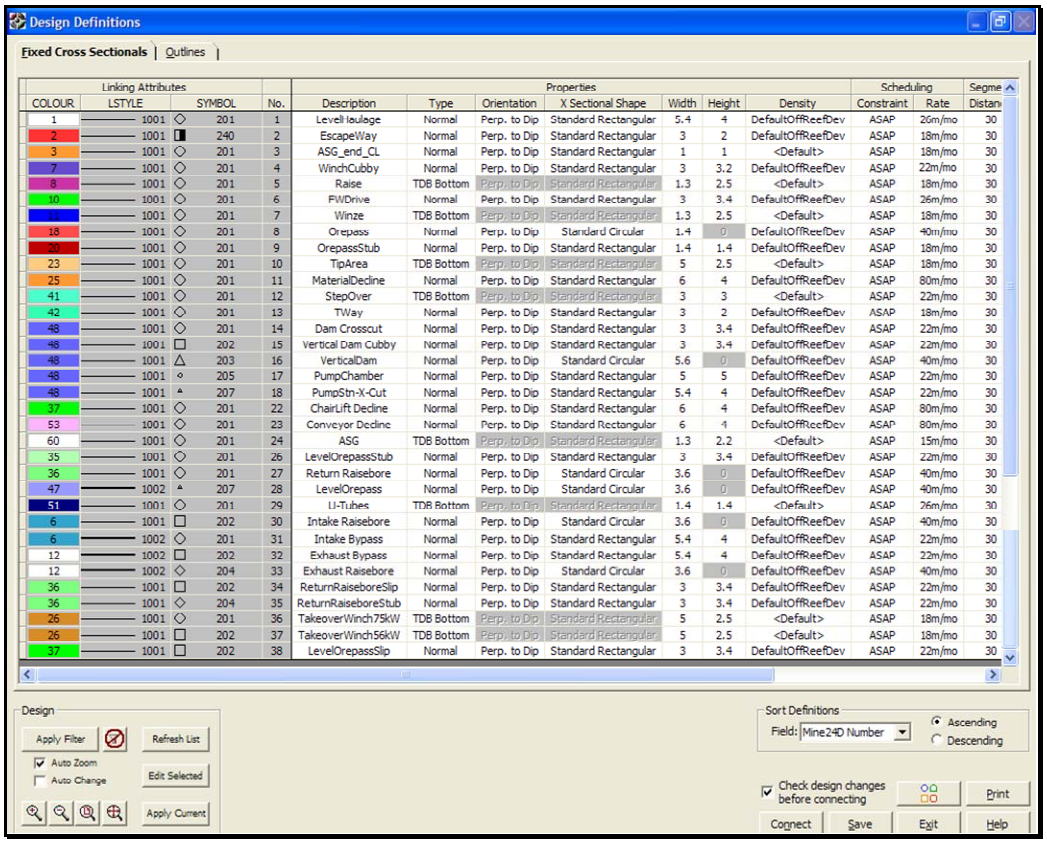


Figure 5.8: Mine2-4D® illustration of design definitions of fixed cross-sectionals assumed for OB1

5.4.2 Off-reef development

Off-reef primary development is typically excavated at a gradient of 1:200, in the footwall waste to provide access to the reef horizon. Figure 5.8 illustrates that all off-reef development has density denoted as *DefaultOffReefDev* which was assigned the value of 3.8 in Figure 4.13. The location of strike haulages relative to the expected elevation of the reef horizon is a major planning decision for a number of reasons. Firstly, strike haulages should be sited in geo-technically competent ground to minimise support requirements for long-term stability. Secondly, strike haulages should be sited such that stress fracturing by over-stoping the haulage on the reef horizon is minimised. The middling between the reef horizon and footwall haulages generally increases with increasing depth of mining because

stress levels increase with depth. Thirdly, it must avoid having longer travelling ways that are difficult for men and material movement to the reef horizon. Lastly, the strike haulage position must allow sufficient length of boxholes to create adequate capacity to handle ore from the stopes. Examples of off-reef development include haulage levels or shaft crosscuts, footwall strike drives, laybyes, travelling ways and boxholes.

The off-reef development consisted of haulage levels branching away from the declines towards the reef horizon, footwall strike drives branching away from haulage levels and developed following reef strike (Figure 5.9); and laybyes, travelling ways (TWays) and boxholes that branch away from the footwall strike drives to the reef horizon (Figure 1.25). The dimensions for these types of development and their associated mining rates vary from one company to another and even within the same company; they can vary from one shaft to the other. The dimensions and average mining rates for off-reef development used in this study were illustrated in Figure 5.8.

Table 5.3 indicates how the development ends were grouped together and assigned common mining rates per group as captured earlier on in Figure 5.8.

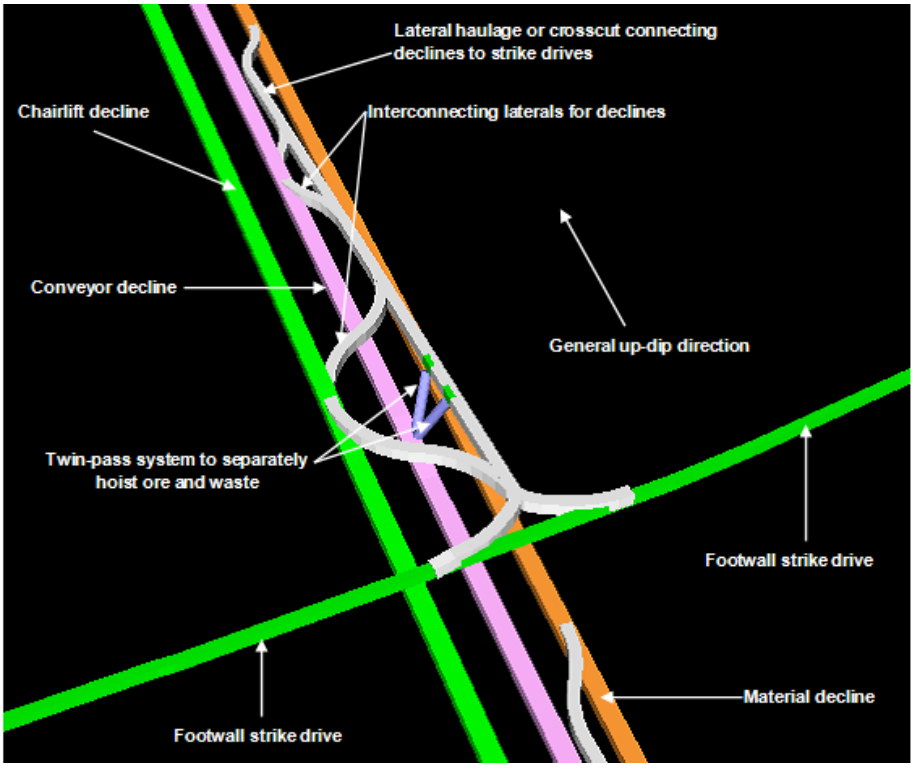


Figure 5.9: 3-D illustration of off-reef development breaking away from declines

Table 5.3: Mining rates of development ends by category

Category	Development ends	Advance rate	Comments
Lateral haulages/drives	LevelHaulage; FWDrive; U-Tubes	26m/momnth	Typically advance at 1.2m per round for an effective 23 shifts/month.
Inclined ends	EscapeWay; Raise; Winze; OrepassStub; TipArea; TWay; TakeoverWinch Chambers;	18m/month	Advance much slower than lateral ends because of more difficult cleaning conditions on inclined pathways.
Lateral chambers	WinchCubby; StepOver; Dam Crosscut; Vertical Dam Cubby; PumpChamber; PumpStn-X-Cut; LevelOrepassStub; Intake Bypass; Exhaust Bypass; ReturnRaisboreSlip; ReturnRaisboreStub; LvelOrepassSlip.	22m/month	Advance slower than lateral haulages because more volume is excavated per metre advanced.
Raise-bored inclined ends	Exhaust raise-bore,	40m/month	Raise-boring much faster than conventional drilling and blasting and is contracted out to EPCM contractors.
Declines	Chairlift decline; Conveyor decline; Material decline.	80m/month	High-speed trackless mechanised development contracted out to EPCM contractors.
In-stope inclined ends	ASGs	15m/month	Advance at same rate as panel faces.

5.4.3 On-reef secondary development

On-reef secondary development is excavated within the reef horizon to delineate and prepare stopes or mining blocks for eventual extraction. There are some factors that must be considered in determining the size and location of on-reef secondary development. Firstly, the spacing of the development must provide adequate sampling density for ore reserve demarcation and evaluation because raises and winzes are sampled regularly for grade estimation and control. Secondly a balance must be made between mining raises and winzes deep into footwall waste to create adequate temporary storage capacity for ore before it is finally transferred to the boxholes and the dilution that the footwall waste brings. Examples of on-reef secondary development include step-overs, raises, winzes, ASGs and ventilation U-tubes for ventilating stopes at the orebody boundaries and these have densities denoted by <*default*> under the density column in Figure 5.8.

5.5 Stoping layout

The stoping layout for a conventional breast mining method includes the planning of pillars, panels, stope shape and size, and on-reef development. These aspects are discussed individually in the next sub-sections. Where a fault was in close proximity to a dyke, the middling between the fault and dyke including the dyke, and dyke bracket pillars was left as an irregular pillar. Additional support would also be used in ground where dykes and/or faults criss-crossed to produce blocky ground.

5.5.1 Regional and boundary pillars

Pillars fulfil the two main support functions of providing regional support and in-stope (alternatively called in-panel) support for underground tabular reef mines. In-stope pillars are designed to either carry the full weight of overburden rock to surface or to just carry a portion of that load. On the other hand regional pillars (which include barrier pillars, boundary pillars and water-barrier pillars) are designed to carry the full overburden weight. Regional pillars additionally serve the function of compartmentalising the mine so that fires, water inflows and in-stope pillar failures can be confined to a localised scale and prevented from spreading to a mine-wide scale. Regional pillars can also be used to demarcate mine boundaries, in which case they are referred to as boundary pillars.

The Department of Minerals and Energy (2001) requires boundary pillars planned to abut adjacent mines to have a total width of not less than 9m on metalliferrous mines. York (1999) investigated the design of hard rock pillars on tabular reef mines and noted that Merensky reef pillars at a width: height (w/h) ratio equal to 10:1 had actually tested to destruction in the laboratory. The laboratory tests invalidated the common belief that regional pillars with a w/h ratio of 10:1 were indestructible. York (1999) therefore recommended that regional pillars for the tabular reef platinum mining environment be designed at w/h ratios greater than 10:1, meaning that the minimum width for a barrier pillar should be 10m for stoping heights (alternatively called stoping width) of 1m. In a separate study of rock engineering practices on narrow reef tabular hard rock gold and platinum mines in South Africa, Jager and Ryder (1999) recommended that barrier pillars should be designed at w/h ratios of not less than 15:1 and that 20m was a typical width for barrier pillars in this design environment. In this study 20m wide boundary pillars were left at the north and south boundaries of OB1 since the average design stoping width is 1m. The outcrop portion of OB1 did not need any boundary pillars since the mine is starting from surface and additionally a 30m portion of the weathered zone of the reef is left unmined as a crown pillar.

When regional pillars have their long axis running along dip direction, they are called down-dip regional pillars and when their long axis runs along strike direction, they are called strike regional pillars. Down-dip regional pillars are preferable to strike regional pillars because they are less likely to fail under shear stress acting on the pillar footwall and hangingwall. This research study therefore assumed the regional pillars to be of the down-dip type. The spacing of the regional pillars is governed by the long-term stability of the stope arch created during stoping operations. Jager and Ryder (1999) indicated that from practice the maximum stable span that can be safely supported is limited to 400m for the hard rock tabular reef mining environment. Jagger (2006) also indicated that stopes for the 16 shaft and 20 shaft projects were designed to have maximum strike spans of 400m. In this study the maximum stope span was taken as 400m. Figure 5.10 shows a typical regional pillar layout for OB1.

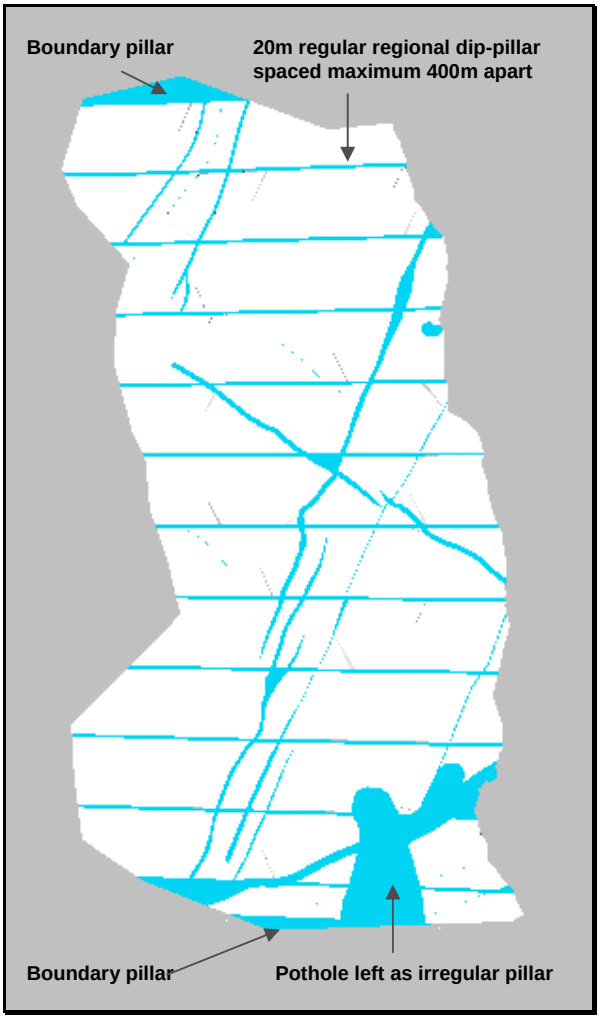


Figure 5.10: A Mine2-4D® illustration of the 67_400 layout showing stoping areas (in white), regular pillars and geological structures left as irregular pillars (in blue).

5.5.2 In-stope (in-panel) pillars

Watson *et al* (2008a) noted that in-stope pillars were introduced into breast mining layouts on the Bushveld Complex as far back as 1978 in order to reduce the occurrence of serious stope collapses colloquially known as 'back breaks'. Without in-stope pillars, back breaks frequently occurred when the unsupported span had advanced to a point 30m to 40m on both sides of the centre gulley (Watson *et al*, 2008a and Watson *et al*, 2008b). Current mining practice across the platinum industry is marked by the widespread use of small in-stope chain pillars orientated either on strike for breast mining or on dip for up-dip or down-dip mining as was illustrated earlier on in Figure 1.22.

For in-stope pillars on gold and platinum mines, York (1999) noted that generally these should have a width of more than 2m and a w/h ratio of at least 3:1 in order to efficiently support the panels. At stoping widths of 1m in-stope pillars should therefore be at least 3m wide. Current practice is for the in-stope pillars to be of length between 4m and 6m and width between 3m and 4m with ventilation holings of 2m between pillars. For example, in the two recent 16 Shaft and 20 Shaft projects at Impala Platinum, in-stope pillars were planned at 6m x 4m with 2m wide ventilation holings (Jagger, 2006). Since this study used a stoping width of 1m, in-stope pillars it was decided to plan in-stope pillars dimensions as 6m x 4m with 2m ventilation holings separating them.

5.5.3 Optimal stope shape (Lagrange multiplier method)

As previously mentioned in Chapter 1, a conventional breast mining stope is defined by the raiseline spacing and backlength. When the backlength is not equal to the raiseline spacing as currently practised on most platinum mines, the stope shape is rectangular. Considering the fact that the RF, which was discussed in Chapter 2, is an important factor in measuring the efficiency of conventional breast mining layouts, it is necessary to establish a stope shape that results in an optimal RF. An optimal RF is obtained when a fixed perimeter defined by a raiseline spacing and backlength produces a shape of maximum area. Such a problem is typically solved using the Lagrange multiplier method. Consider a stope shape bounded by two consecutive raiselines each of length B and at a raiseline spacing of R as shown by the longitudinal section in Figure 5.11.

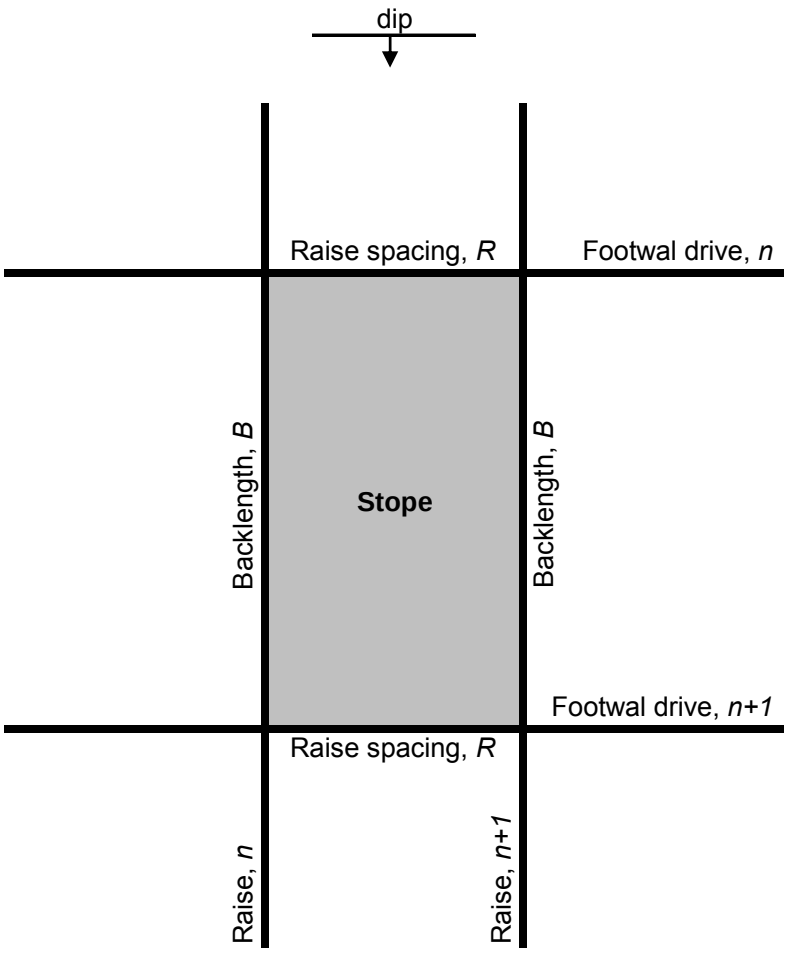


Figure 5.11: Stope blocked out by two consecutive raiselines and two consecutive levels

The problem can be summarised as a function of two variables *R* and *B* that requires maximising the area of the stope, *A*, given by Equation 5.3.

$$A = RB$$

Equation 5.3

subject to the perimeter constraint given by Equation 5.4.

$$P = 2R + 2B$$

Equation 5.4

where *P* is the fixed perimeter of the stope. The Lagrangian, *L*, for this system of equations is given by Equation 5.5.

$$L = RB + \lambda(P - 2R - 2B)$$

Equation 5.5

where λ is the Lagrange multiplier.

By taking the partial derivatives for R, B, and λ and setting them to zero to determine the saddle points the transform equations shown by Equation 5.6, Equation 5.7 and Equation 5.8 are obtained.

$$\frac{\partial L}{\partial R} = B - 2\lambda = 0$$

Equation 5.6

$$\frac{\partial L}{\partial B} = R - 2\lambda = 0$$

Equation 5.7

$$\frac{\partial L}{\partial \lambda} = P - 2R - 2B = 0$$

Equation 5.8

The solution of the two transform equations, Equation 5.6 and Equation 5.7, is given by Equation 5.9.

$$\frac{R}{2} = \frac{B}{2} = \lambda$$

Equation 5.9

When Equation 5.9 is back-substituted into the transform Equation 5.8, the following equations are obtained:

$$R = B = \frac{P}{4}$$

Equation 5.10

$$\lambda = \frac{P}{8}$$

Equation 5.11

It can be concluded from Equation 5.9 and Equation 5.10 that the maximum RF is obtained when the raiseline spacing is equal to the backlength, that is, the optimal stope shape is a square. Therefore maximum values of RF are obtained when stope shapes are nearly square in shape when raise spacing is almost equal to level spacing. The value of the Lagrange multiplier, λ, given by Equation 5.11 means that a unit increase in perimeter will result in a one-eighth increase in area provided the change in perimeter is kept small enough. For example at a raiseline spacing of 180m and backlength of 180m, R = B = 180m, implying that P = 720m and A = 32,400m^2. If P increases to 721m, then R = B = 180.25m and A = 32,490.0625m^2. Therefore by comparing the two calculated areas, $\Delta A \approx$ 32,490-32,400 = 90. Using the expression for λ, $\Delta A = 90 \approx \frac{720}{8}$ (i.e. $\approx \frac{P}{8}$).

5.5.4 Panel length

From a rock engineering perspective, a stope panel collapse will occur due to excessive panel length or stope span which is often dictated by equipment in use and previous experience under similar conditions (Swart and Handley, 2005). In their study, Swart and Handley (2005) used four different rock mass classification systems for a chromitite seam in a shallow mining environment on the Bushveld Complex and established that the length of stable stope spans lies in the range 12m-50m. Jager and Ryder (1999) noted that 28m is a typical panel length in breast mining layouts in many hard rock situations. Impala Platinum (2007) standards use a 30m panel length when considering manpower allocations. As mentioned earlier on in Chapter 2, the study by Brassell (1964) established that the optimal panel length for breast mining was between 30m-36m. Based on these studies, it was therefore decided that a panel length of 30m was appropriate for this study.

5.5.5 Take-over scraper winch concept

The maximum scraping distance for a 75kW gulley scraper is approximately 200m when scraping down-dip and approximately 160m when scraping up-dip. The difference in the scraping distances is due to gravity aiding down-dip scraping in rock movement but limits up-dip scraping because gravity works against up-dip scraping. The maximum scraping distance of 200m is also dictated by the amount of rope that a scraper drum can take. Similarly, the maximum pull for a 56kW scraper winch is about 100m. Therefore in this study, at raiseline spacing exceeding 200m, 56kW ASG scraper winches were substituted with 75kW scraper winches. This is reflected by the scraper winch cost prices captured in the Microsoft Excel files *TechnoEconAnalysis* contained on the USB memory stick. In some cases, due to irregular surface undulations of OB1, backlengths could exceed 300m. Impala Platinum also sometimes experiences such cases and use take-over winches to scrape the distances in excess of 200m up-dip or 100m down-dip. This is the reason why in this research study the maximum raise spacing was set at 400m and backlength at 400m (≈67m at 9.6º).

5.6 Basic Grade Equation (BGE)

The BGE is a term used in the South African platinum mining industry to describe the process of determining the mined ore grade after factoring the effects of dilution, mining cut, over-break and under-break, extraction percentage and waste handling. Detailed calculations on this section are contained in the Microsoft Excel® file *OB_Grade_Equation_Model* included on the USB memory stick. These concepts are individually discussed in the next sub-sections.

5.6.1 On-reef development dilution

As explained in Section 5.4.3, on-reef development is typically mined to include a waste portion, thereby bringing in dilution. For example, consider a raise or winze with dimensions given earlier on in Figure 5.8 as 1.3m wide by 2.5m high, being mined on OB1 with reef that is 1m thick as shown in Figure 5.12 and an average PGE grade of 6.1g/t as given in Chapter 4. The bottom 1.5m of the raise or winze will be in waste and it can be conservatively assumed that the waste is mined at a grade of 0g/t. From the Microsoft Excel® file *OB_Grade_Equation_Model* it can be seen that ore coming from raises and winzes will be at a grade that is 41.24% of *in-situ* grade. The *in-situ* grades for raises were read into EPS® from Mine2-4D® and subsequently exported to the Microsoft Excel® file *TechnoEconAnalysis*, were adjusted by this factor to get final grade attributable to each raise or winze. This process was done for all on-reef development.

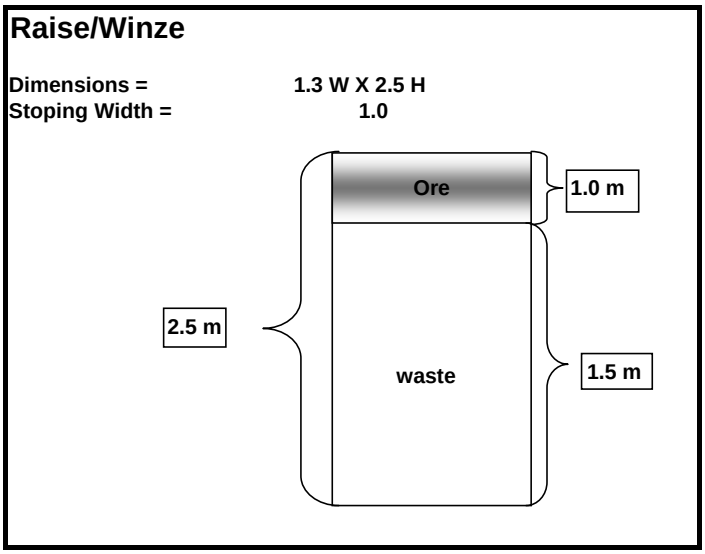


Figure 5.12: Cross-sectional area of a raise or winze showing the ore and waste portions

5.6.2 Optimal resource mining cut

The optimal resource mining cut location and size for platinum reefs is determined mainly by the PGE value distribution over the mineralised width, the base metals (Cu and Ni) value distribution over the mineralised width, the existence of any recognisable geological reference datum such a reef contact or where this is not visually possible the reference datum is determined by sampling, geo-technical constraints and mining technology used for the ore extraction. The size of the optimal mining cut should be such that equipment and employees can fit into the stope width but dilution is minimised so that the highest possible

PGE and base metal grades are obtained. Anglo Platinum Projects Division (2007), and Lionnet and Lomberg (2006) provide examples of how the above factors have been used determine the practical optimal mining cuts for Unki platinum mine and Bafokeng Rasimone platinum mine (BRPM), respectively. Figure 5.13 shows a typical PGE value distribution for the UG2 reef and its associated geological facies. The mining cut should ideally not go beyond the UG2 reef footwall contact because then it encroaches into barren pegmatoids. Most operations on the Bushveld Complex assume a minimum mining cut of 0.9m and a maximum of 1.8m when using trackless mechanised mining methods, in order to minimise dilution while at the same time providing a working height in which employees can endure over a full working shift. The average reef thickness for OB1 was 1.1m and a mining cut of 1.0m was therefore assumed.

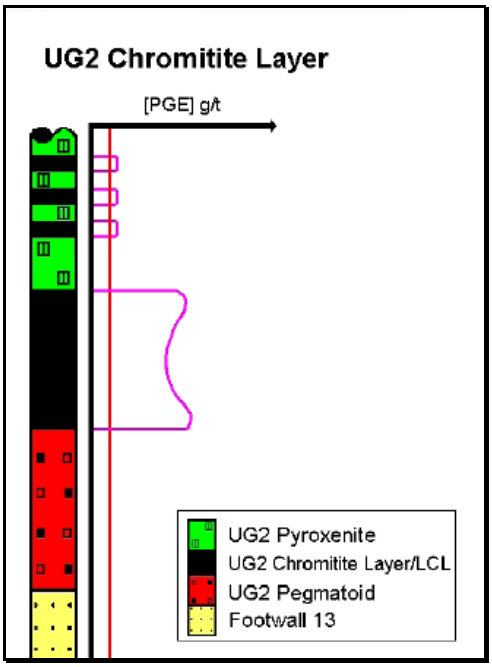


Figure 5.13: Typical PGE value distribution over reef thickness in UG2 reef
(Impala Platinum Mining Projects, 2003)

5.6.3 Over-break and under-break

Poor geo-technical ground conditions and drilling errors which are encountered during mining on the platinum reefs are the major contributors of unplanned dilution. The rolling reef geological feature described in Chapter 1, particularly when it occurs with a high amplitude combined with a short wavelength, tends to lead to higher unplanned dilution although this is usually minimised by mining ASGs at 20º-25º above strike. An over-break is undesirable because it leads to loss of grade by mining more waste. Under-break on the other hand is mainly due to drilling errors and uneven footwall. In trying to avoid mining pegmatoid footwall

waste, drilling and blasting of the toe holes can leave part of the reef in the footwall. This is called an under-break. An under-break is undesirable because it leads to loss of grade by excluding part of the reef which was supposed to have been mined. However, experience d drillers are able to minimise over-breaks and under-breaks. Therefore, conservative figures of 5% over-break and 5% under-break were assumed for OB1, assuming that experienced operators will be employed.

5.6.4 Area extraction percentage per panel

The calculation of the area extraction percentage assumed that there were no local geological losses or mining losses. The calculation was therefore based on a 30m stope panel, with 6m long by 4m wide in-stope pillars, separated by 2m wide ventilation holings to give an area extraction of 91.5%. This calculation is contained in the Microsoft Excel® file OB1_*Grade_Equation_Model*. This factor was used in EPS calculations to determine the *in-situ* stoping centares mined after geological losses from dykes, faults and potholes had been accounted for.

5.6.5 Waste handling

As shown earlier by Figure 5.9 a twin-pass system was designed for each level to allow for separate handling of ore and waste. All development waste will be treated as waste and tipped into the waste pass while all ore from stoping will be tipped into the ore pass.

5.7 Basic mining equation (BME)

The BME is a term used in the South African platinum mining industry to describe the calculation procedure of the number of production units required to meet a planned production level by considering constraints to production. A typical panel production crew can produce between 400m^2-500m^2 of stoping per month based on Rogers (2005) and the Impala Platinum best practices. Ideally about 3 production crews should be allocated per raiseline although a maximum of 5 production crews can be allocated to a single raiseline. If more than 5 crews are allocated to a raiseline, the number of people working in the stope becomes too large for comfortable and safe working conditions. Additionally, the logistics of services required become more difficult resulting in more lost blasts and slower face advance rates. Therefore for this study it was assumed that 5 production crews would be allocated to a raiseline but the scheduling in EPS® was such that it could allow the 5 crews to be split so that some crews could be allocated to an adjacent raiseline. This would also ensure better flexibility to meet planned production.

Typical production call for a half-level is at least 2,000m^2 per month with the production coming from one raiseline or in some cases two raiselines per half-level. This production constraint is dictated by boxhole capacity per raiseline and capacity of level tramming system, which is usually done using 10t battery locos pulling a train of 8 by 6t hoppers or 10 by 4.5t hoppers. For this study, 5 crews can produce about 2,250m^2 per month assuming each crew can do about 450m^2 per month. To get the equivalent *in-situ* production rate required this figure is adjusted by the area extraction percentage of 91.5% to get about 2,460 *in-situ* m^2 per month. This is the EPS® stoping rate applied to OB1 as shown in Figure 5.14. Based on the production rate of 100,000tpm derived in Section 5.3, at a tonnage factor of 4t/m^2, as derived in an earlier part of the thesis, this production rate is equivalent to 25,000m^2 of stoping per month. This production figure was adjusted with estimated local geological, regional pillar and mining losses of 30% in addition to the total loss of 7.81% attributed ground left as part of potholes in order to carry out Flexibility Index calculations shown in the Microsoft Excel® files *TechnoEconAnalysis*. The production rate of 100,000tpm can be achieved by mining approximately 12 half-levels. This figure compares well with the 16 Merensky half-levels and 16 UG2 half-levels for a combined production rate of 225,000tpm for Impala No. 17 Shaft project (Zindi, 2008).

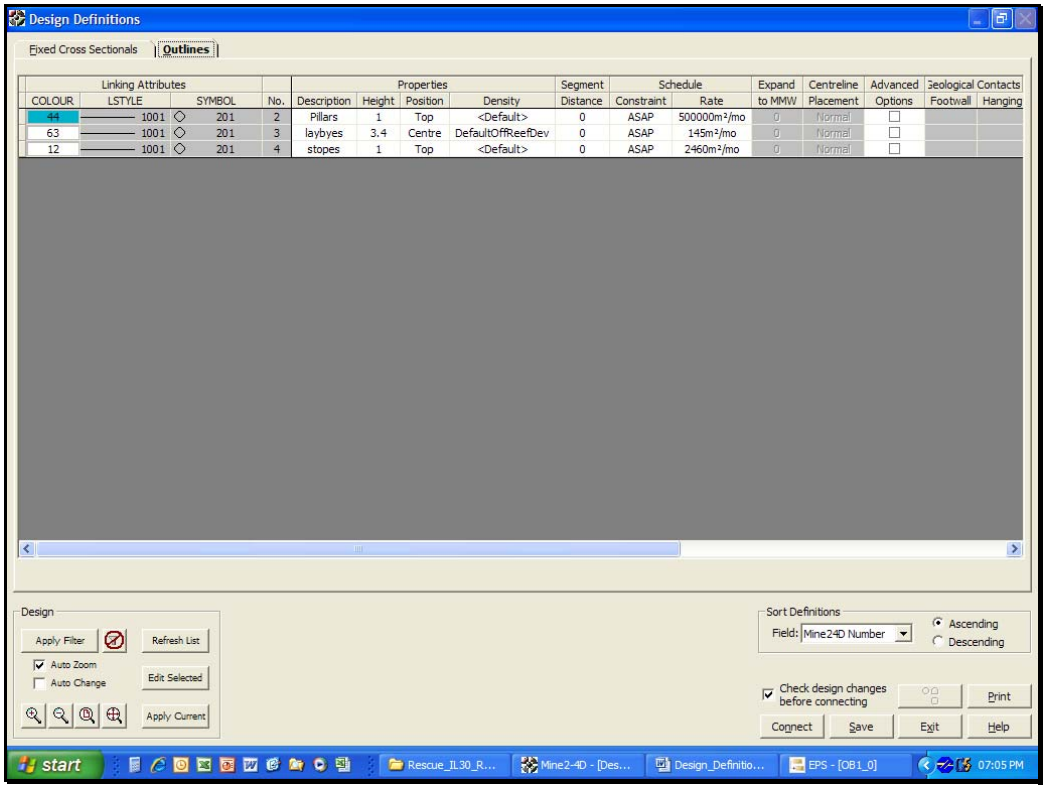


Figure 5.14: Mine2-4D® illustration of design definitions of outlines assumed for OB1

Figure 5.14 also shows that laybyes which are part of off-reef development were classified outlines design strings not as fixed cross-sectionals because their cross sectional area increases from 3m wide by 3.4m high when they break away from footwall strike drives to 5m wide by 3.4m high and end in a refuge bay that is 3m wide by 3.4m high. Since their cross-sectional area is variable, they were handled as outline design strings. The average area of a laybye as calculated in Mine2-4D® is about 290m^2 and Mohloki (2007) indicated that it takes about 2months to completely mine a laybye. This is why the scheduling rate used in EPS® is 145m^2/mo. Figure 5.14 also shows that pillars have an excessively high mining rate of 500,000m^2/month. This was done deliberately, so that EPS® could calculate the pillar areas and tonnages and report them in the production schedule at the start of the scheduling process in the year 2010 as indicated in the Microsoft Excel® files *TechnoEconAnalysis*.

5.8 Ventilation planning

The minimum ventilation requirements are the mining ventilation requirements. Ackerman and Jameson (2001) indicated that the air required for mining on the Bushveld Complex is approximately 3m^3/s per kt per month for both reef and waste rock broken. Jagger (2006) gave air factors for the Bushvled Complex between 3.2kg/s – 3.6kg/s per ktpm of both reef and waste rock broken. Fleming (2002) gave total air requirements for narrow reef tabular gold mining as 3kg/s per ktpm. For this study an air factor of 3.5kg/s per ktpm was therefore used for ventilation planning. For a production rate of 100,000tpm, the air requirement would be 350kg/s. A figure of 400kg/s was then used to estimate the capital cost for the ventilation system as this additional capacity of 50kg/s would cater for extra heat load pick-up due to use of men and machinery that generate heat. It was decided that the chairlift decline and material would serve as the main air intakes while the conveyor decline would serve as the main exhaust to ensure that dust is blown out of the mine and not into the mine along the conveyor carrying blasted rock material. Some of the main levels would be used as dedicated intake airways while others would be used as dedicated exhaust airways following the ventilation design proposed by Jagger (2006) for Impala 16 shaft project.

5.9 Manpower planning

Decline and ancillary development, level haulage and raise-boring would be contracted out. EPCM contractors indicated that development rates of 40m/month are achievable for raise boring if the boxhole stub and tip area are already prepared by the mine. Labour would be required for off-reef strike development, secondary on-reef development, production stoping, and ancillary services such as winch moves, half-level maintenance, tramming, and construction, pre-development and re-development, centre gulley cleaning. The labour team

complement setting was done using guidance from the Code of Practice described in Impala Platinum (2007) and summarised in Table 5.4 for convenience. The assumption is that the project will operate on a double-shift system whereby drilling and blasting are performed by the morning shift teams, while night shift teams execute all cleaning and support activities. Pre-development and re-development teams work morning shifts only. The centre gulley winch is operated on both shifts. Panel establishment and support teams work night shifts only. Maintenance, tramming, and construction teams are allocated on a per half-level basis; specifically equipment helpers for haulage maintenance are allocated on the basis of 1 helper per 500m of half-level haulage distance. OB1 strike distance is about 4km implying that each half-level will require about 4 helpers. Once a working half-level is completely mined, the labour teams will leapfrog to the next replacement half-level. Labour will be hired and laid off to suit the production and development profile. A labour contingency of 5% was incorporated for to cater for absenteeism, sickness and disablements. A further assumption was that mining labour accounted for 50% of total mine labour.

Table 5.4: Labour complement setting for OB1

Team Category	Shift Labour Distribution	Total Allocation
Main Teams		
Main Development	Morning (6) and Night (2)	8
Secondary Reef Dev	Morning (3) and Night (5)	8
Stoping (per panel)	Morning (4) and Night (8)	12
Centre Gulley (per stoping team)	Morning (1) and Night (1)	2
Ancillary Teams		
Panel Establishment / Support (per stoping team)	Night only (4)	4
Pre- and Re- Development (for raise spacing<=100m)	Morning only (2)	2
Pre- and Re- Development (for raise spacing>100m)	Morning only (6)	6
Takeover Winch	Morning (4) and Night (4)	8
Per Half-Level Teams		
Winch Moves (stoping only)	Morning only (4)	4
Maintenance, Tramming, Construction	Morning (11) and Night (3)	14

In EPS® manpower planning was captured as a resource that is assigned to development and stoping activities. Each working half-level was assigned a main development team. This team would move to a replacement half-level when main development on the working half-level was completed. This team mines FW Drives at 26m/mo, laybyes at 145m^2/mo, WinchCubbies at 22m/mo through the use of production look-up tables in EPS®. There were two secondary development teams; one for raising and the other for winzing. Secondary development teams would also mine TipAreas, U-Tubes, TakeoverWinch Cubbies, StepOvers, whenever these arise, again though using the production look-up tables. Stoping teams mine at a rate of 450m^2/mo per team which is equivalent to 491.803m^2/mo of *in-situ* centares after accounting for panel extraction of 91.5%. Figures Figure 5.15, Figure 5.16 and

Figure 5.17 are examples of how manpower planning was captured in EPS® as a resource. Once resources were assigned to activities, the resources were levelled in order to smooth out production. The process of levelling allows EPS to compare available resource capacity against capacity required by activities and then do an assignment of the proportion of the resource to meet the activities' capacity. For example if only one secondary development end is available, but there are two secondary development teams, then EPS will assign only one team to the development end since each development end can only be mined by one team. However, because there is a resource over-capacity, EPS® will highlight this resource assignment in red. Once schedules were levelled, they were subsequently exported to the Microsoft Excel® files *TechnoEconAnalysis* for further analysis.

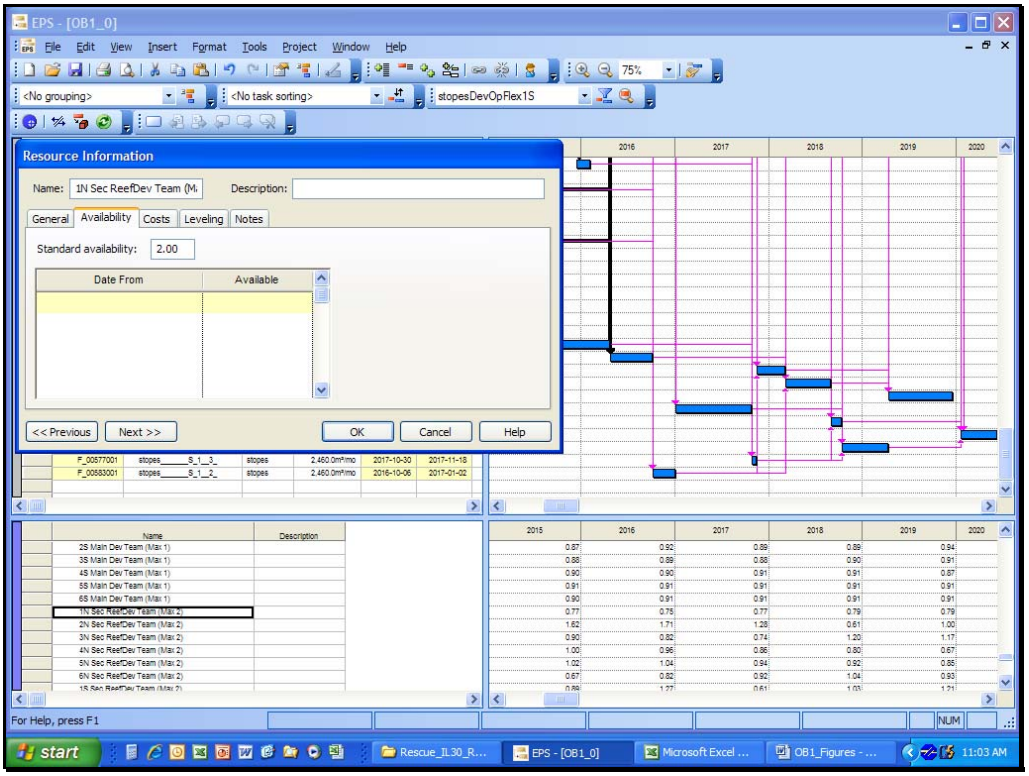


Figure 5.15: An EPS® illustration of a maximum of two teams for secondary on-reef development

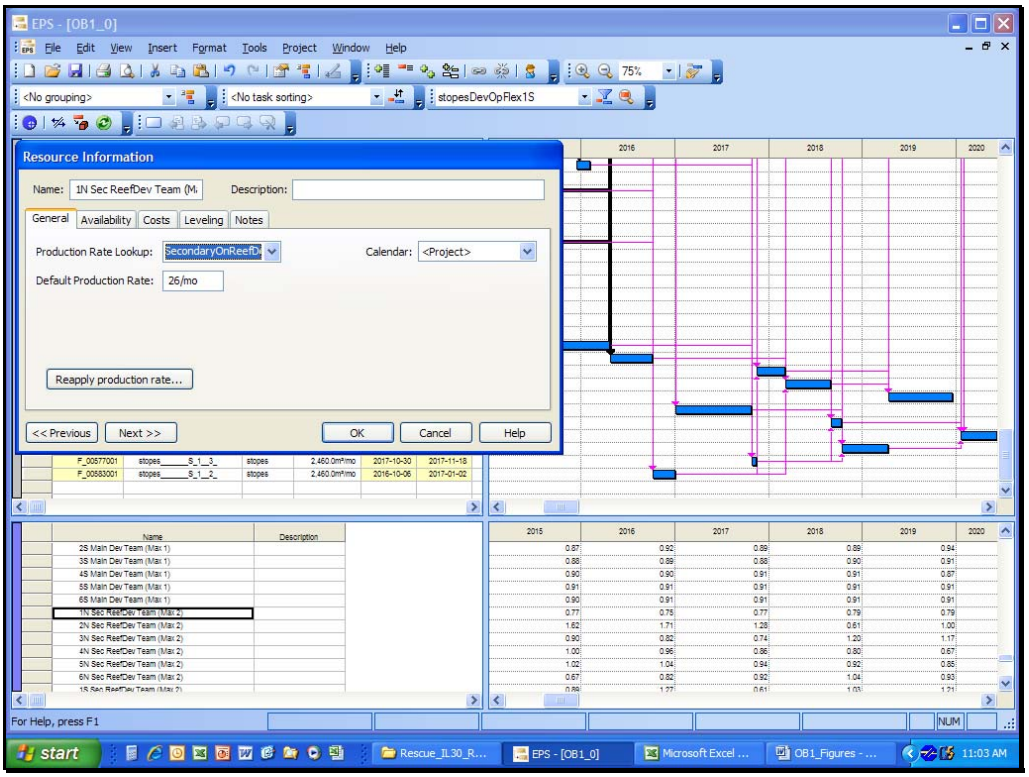


Figure 5.16: An EPS® illustration of a secondary on-reef development team using the *SecondaryOnReefDev* look-up table

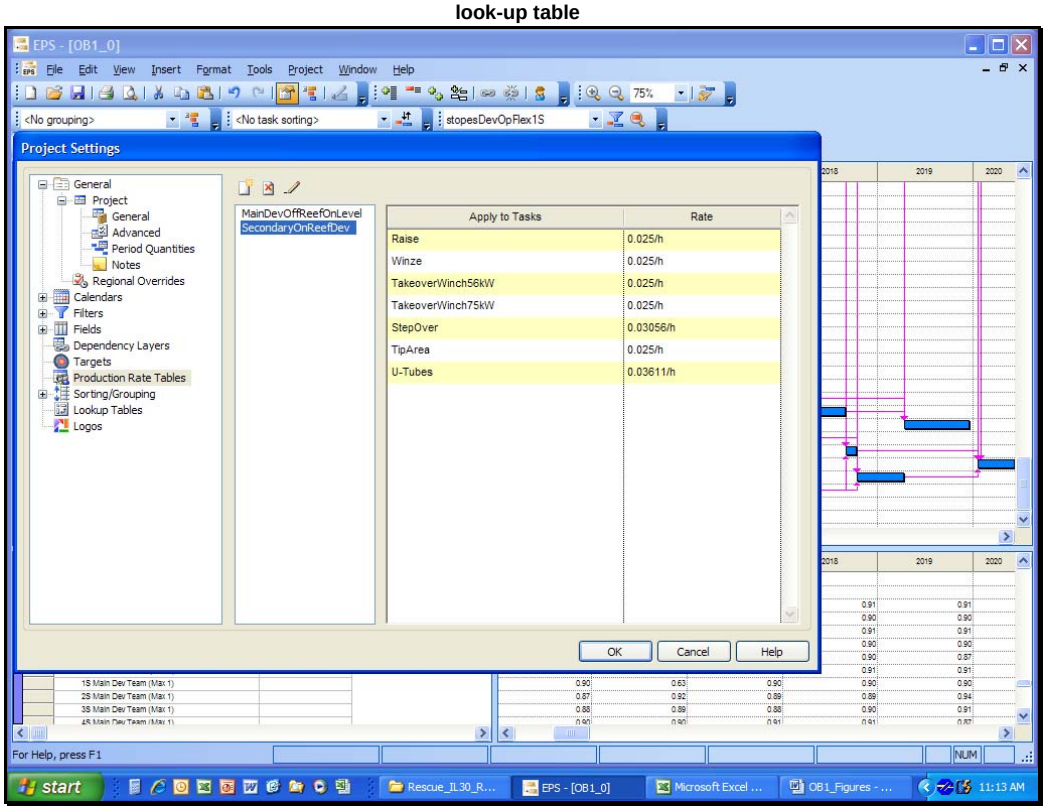


Figure 5.17: An EPS® illustration of a production look-up table showing variable mining rates for different development ends

5.10 Tramming equipment planning

It was assumed that each half-level would be equipped with a battery loco and ancillary equipment such as LM loaders, hoppers, material cars, explosives car, jumper car, engineering car, drill carriages and cable car. The cost of a battery loco and its two batteries are captured separately while the cost of ancillary equipment is captured as rolling stock in the Microsoft Excel® files *TechnoEconAnalysis*.

5.11 Metallurgical Assumptions

UG2 ores have traditionally been difficult to concentrate and smelt. The small PGM grain sizes make it difficult to float the PGM values into a concentrate while the low Cu/Ni sulphides coupled with high chromite content in the UG2 ores make it difficult to treat the PGM concentrate (Deeplaul and Bryson, 2004). Pincock (2008) quote a chromite content of between 60%-90% for UG2 ores compared to 3%-5% for Merensky reef ores. Northam (2009) indicated that chromite content in UG2 ore ranged between 25%-30%, while chromite content in Merensky reef ranged from 1%-2% at Northam mine. The difficulty with chromite (Cr_2O_3) in the ore is that it contains chromium which has a limited solubility in the furnace slag thereby forming chromite spinels (Coetzee, 2006). The chromite spinels have a high melting point and are very dense and therefore precipitate and settle out at the bottom of the furnace resulting in clogging of the slag tapping system and concomitantly, electrodes 'lifting out' of the furnace bath with subsequent electrode arcing (Coetzee, 2006; Northam, 2009). These problems also lead to low PGM and base metal recovery. These challenges have been driving the research for improvements in mineral processing technology of UG2 ores in the recent decade. The UG2 ore is currently treated in Mill-Float-Mill-Float (MF2) circuit plants (Hay and Rule, 2003; Deeplaul and Bryson, 2004; Hay and Schroeder, 2005; Coetzee, 2006; Pincock, 2008; Northam, 2009). The Mill-Float (MF1) plants typically that are used to treat Merensky ore were initially used to process UG2 ores but due to low recoveries obtained, these were subsequently upgraded to MF2 processing plants.

In the MF2 processing strategy the UG2 ore is coarsely milled and floated and the flotation tailings are further milled to a finer size in order to liberate the small PGM grains which are re-floated. This technology has been able to produce concentrator recoveries ranging between 80%-90%, concentration factors ranging between 30-80 to give a PGE concentrate grade of 100g/t-600g/t at a Mass Pull factor of grater than 1% (Pincock, 2008; Northam, 2009; Hay, 2009). The concentrator Mass Pull is the ratio of the concentrate tonnage to the initial ROM tonnage, expressed as a percentage. The beneficiation assumptions shown in Table 5.5 were based on Hay (2009) estimates were therefore used and gave average PGE

concentrate grade of about 250g/t as shown in the Microsoft Excel® files *TechnoEconAnalysis*.

Table 5.5: Beneficiation Assumptions

Concentrator Recoveries	**%**
Pt	**78.12%**
Pd	**84.84%**
Rh	**78.12%**
Au	**96.60%**
Overall Concentrator Recovery	**84.00%**
Concentrator Mass Pull	**1.65%**

The concentrate would be toll-smelted based a typical toll-smelting contract whereby the project would receive 78% of the PGE value in the concentrate based on a 90-day payment pipeline and associated penalties such exceeding chromite or moisture content limits. For this research study it was assumed that the 90-day pipeline could be ignored in order to simplify calculations and also because roll-over payments would equally affect all the 15 layout schedules.

5.12 Financial modelling assumptions

The author is associated with Venmyn in an associate consulting role. Venmyn is a South African registered consulting company whose core business is to undertake independent techno-economic assessments and valuation of mineral assets for possible transactions or listing on the Johannesburg Stock Exchange (JSE) or other international Stock Exchanges. Details of techno-economic assessments and valuation of platinum (or PGM) mineral assets undertaken to date by the company can be viewed at their website: http://www.venmyn.co.za. Most of the financial modelling assumptions were obtained from Venmyn's internal company database and modified where necessary to suit the level of accuracy that was required for this study.

Venmyn regularly receives consensus forecasts for mineral prices, inflation and exchange rates from reputable analysts for use in mineral asset valuations. These forecasts are typically given for a five-year period after which the forecasts are held constant as long-term (LT) forecasts. The most recent forecasts that were available at the time when the financial analysis of the 15 layout options was conducted are outlined in the next sub-sections. The financial models are contained in the Microsoft Excel files *TechnoEconAnalysis* included on the USB memory stick and assume 2010 as the base year for all calculations. The assumptions made for the key financial inputs are discussed in the following sub-sections.

5.12.1 Forecasts of commodity prices, inflation and exchange rates

The long-term US inflation was estimated to be 2.5% while the long-term South African inflation was estimated to be 8.85%. For exchange rates, Venmyn often uses an in-house calculation model based on the relative Purchasing Power Parity (PPP) theory for log-term exchange rate forecasts and this calculation model was adopted in this study. The relative PPP theory assumes that exchange rates change over time according to the inflation differentials between the two countries under consideration, based on the long-term inflation forecasts. Since commodity prices are quoted in US$ the PPP model for the US$/ZAR exchange rate developed in Microsoft Excel® was used as shown in the file *TechnoEconAnalysis*. A summary of the 4E commodity price forecasts which were used in this study are shown in Table 5.6.

Table 5.6: 4E Commodity price forecasts in US$/oz using 2009 as the base year

PGE	Real Terms base 2009	2010	2011	2012	2013	2014	LT
Pt	**1,040**	1,145	1,237	1,303	1,303	1,305	1,305
Pd	**212**	253	320	359	363	361	361
Rh	**1,324**	1,742	2,159	2,549	2,579	2,582	2,582
Au	**917**	917	906	867	860	802	802

5.12.2 Royalty rate

The South African Mineral and Petroleum Royalty Bill that was first released in 2003, was enacted in 2008 into the Mineral and Petroleum Resources Royalty Act, after going through three revisions. May 2009 was the date set for its implementation but this has had to be postponed to March 2010 by government in an effort to mitigate job losses that could have arisen if the Act had been implemented under the effect of the current global economic crisis (Research Channel Africa, 2009). The royalty calculation guidelines contained in the Royalty Act that was proposed for implementation from 1st May 2009 were adopted for the financial model. These guidelines suggest that royalty should be paid on a sliding scale depending on profitability with a minimum threshold of 0.5% meant to protect the integrity of the Bill (Research Channel Africa, 2009). The quantum of the royalty revenue payable on all minerals is dependent on the profitability of the company based on the following formula given by Equation 5.12.

$$Royalty\ Rate = \frac{Earnings\ before\ Interest, Taxes, Depreciation\ and\ Amortisation}{Aggregate\ Gross\ Sales\ x\ 12.5}\ x\ \frac{10}{1}$$

Equation 5.12

5.12.3 Discount rate

The Capital Asset Pricing Model (CAPM) was used to derive the project discount rate based on a South African risk-free rate because the project's cash flows will be ZAR denominated, since OB1 is a South African deposit. The CAPM is used in financial modelling to calculate the cost of equity using the formula given by Equation 5.13. In this research study the cost of equity derived using the CAPM is equal to the weighted average cost of capital (WACC) since it is assumed that the project will be wholly funded by equity.

$$Rp = Rf + \beta(Rm)$$

Equation 5.13

where *Rp* is the project discount rate or expected return on investment, *Rf* is the risk-free rate, β is the a measure of the relative volatility of the project stock if it were a listed entity, and *Rm* ifs the market risk premium. When using the CAPM, Venmyn prefers to use a mining project risk, which varies with the level of techno-economic knowledge on the project. The project risk ranges between 2%-10% and is weighted according to nineteen critical project development factors (Figure 5.18). Since OB1 is from the Bushveld Complex whose geological prospectivity is well known and that it is has been categorised as a Measured Resource in Chapter 4, and it is further assumed that the deposit will be mined by an existing large platinum mining company with experience and knowledge of mining on the Bushveld Complex, it was prudent to assume the lower limit of 2% for mining project risk (Table 5.7). A risk premium of 3% was assumed since South Africa is considered to be an emerging market. Since OB1 is a typical Bushveld Complex deposit that will be mined by a stable, large platinum mining company, it was appropriate to assume a beta (β) equal to one, as the company would be expected to move with the market. Typically in South Africa, the R153 and R157 are used as indicators of risk-free rates. The maturity of a R153 bond is typically 3 years while that of a R157 bond is typically 5 years. It was therefore more appropriate to use the risk free rate for the R157 bond since the layouts were designed and scheduled at LOM resolution. The R157 bond rate as of 13th June 2009 obtained from the Standard Bank website was 8.29% (Table 5.8) and this is the risk-free rate used in the CAPM calculations. As Table 5.7 shows, the project discounted rate was estimated to be 13.29% but since this study is at a pre-feasibility level of study it was approximated to 13%. The detailed calculation of the discount arte is contained in the Microsoft Excel® file *Calculation_of_Discount_Rate*.

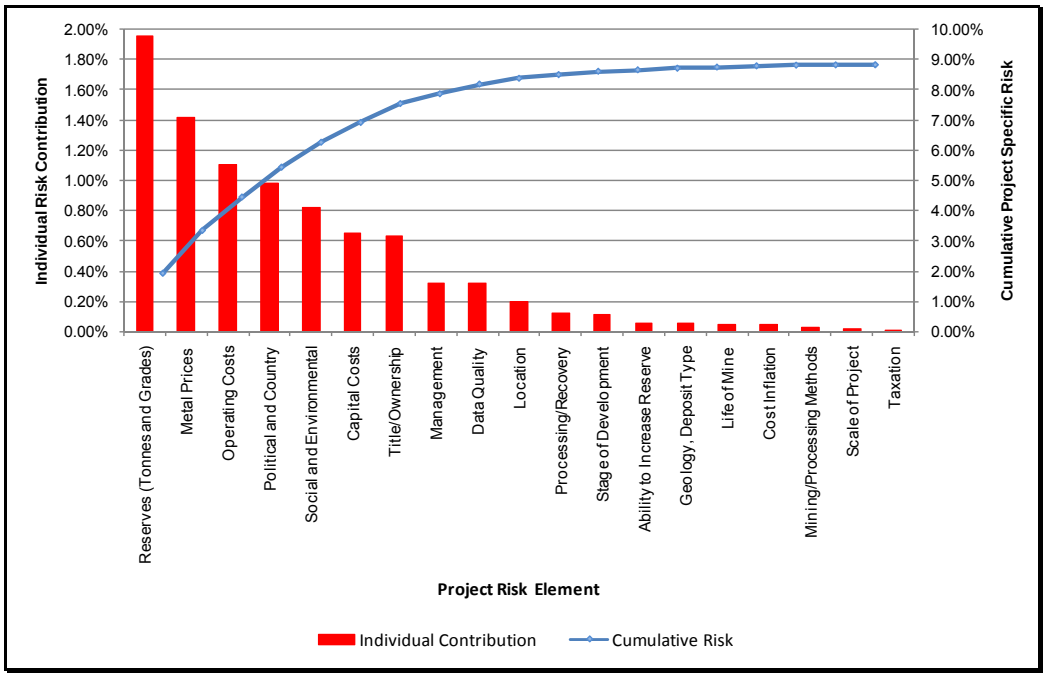


Figure 5.18: Individual contribution to project risk by 19 project development risk components
(Courtesy of Venmyn)

Table 5.7: Project discount rate estimation for OB1 using CAPM

CAPM item	Estimate
R157 Bond Rate	8.29%
SA emerging market	3%
Assumed Beta	1
CAPM	11%
Project-specific risk	2.00%
Project discount rate	**13.29%**

Table 5.8: R153 and R157 bond rates as of 13th June 2009

Source: http://ws9.standardbank.co.za/marketrates/#

Market rates & tools

Bonds

Bond	Bid	Offer	Time	Date
R 153	6.84	6.74	20:58	13-Jun
R 157	8.29	8.26	20:58	13-Jun

5.12.4 Corporate Tax Rate

The South African corporate tax rate for mining companies is 28% and is charged on taxable income from mining (Deloitte Touche Tohmatsu, 2009). This is the tax rate that was used in this research study as can be seen in the Microsoft Excel® files *TechnoEconAnalysis*. The taxable income is derived from accounting profits adjusted by certain allowances as provided for in the Income Tax Act. Tax losses arising in any one accounting period may be carried forward. In addition to these deductions, a mining company is allowed to capitalise all expenditures made during development.

5.12.5 Capital costs

It is normal practice for platinum mining companies capitalise all development expenditure up to the second raiseline for mining projects based on conventional mining layouts (Zindi, 2008; Rogers, 2009). However, for this project the capital expenditure was restricted to capital expenditure (capex) for declines and associated infrastructure only since these were the primary means of accessing the orebody was the same for all layouts. The major capital costs were assumed to be made up of capital costs for the decline cluster, ventilation infrastructure, processing plant, locos and associated rolling stock, ancillary surface infrastructure and tailings impoundment dam. These are briefly described below.

The capex for declines and associated infrastructure was based on typical industry average EPCM cost estimates per metre of equipped development that were obtained from EPCM contractors though the offices of Impala Platinum Projects. The PV of the costs was estimated to be ZAR796.03mil for all layouts. The unit costs are captured in the Microsoft Excel files *TechnoEconAnalysis*.

Capital costs and unit operating costs for ventilation were solicited from a reputable ventilation consulting company through the offices of Impala Platinum Projects. The production rate, production profiles and layouts designs had to be submitted to the company to enable them to draw realistic estimates. The ventilation system was designed for a capacity of 400kg/s that was derived in Section 5.8. A summary of the estimate costs are shown in Table 5.9.

Table 5.9: Summary calculation of ventilation capital and unit operating costs

VENTILATION OPERATING COSTS EXAMPLE			
Primary vent flow	400	kg/s	
Main fan power	2400	kW	
Auxiliary fan power	1920	kW	80% of primary
Total fan power	4320	kW	
Cost of power	0.24	R per kWhr	
Cost of fan power per annum	R 9,082,368		
Auxiliary fan replacement per annum	20	% of inventory	
Auxiliary fan replacement per annum	384	kW	
Cost of auxiliary fan replacement per annum	R 1,344,000	3500	R per kW with duct systems
Total above operating costs	R 10,426,368		
Additional provision	R 2,085,274	20	% staffing, vent doors, etc
Total cost consideration	R 12,511,642		
Total above operating costs per kg/s	**R 31,279**		
VENTILATION CAPITAL FOR 400 kg/s MINE			
Primary vent flow	400	kg/s	
Main fan power	2400	kW	
Capital cost of main fan station	R 20,400,000	8500	R per kWabs
Auxiliary fan power	1920	kW	80% of primary
Capital cost of auxilary fan duct systems	R 6,720,000		
Total above vent capital	**R 27,120,000**		

The capex estimates for the processing plant were obtained from an independent metallurgical consultant, Martin Hay, through the offices of Venmyn as indicated by Hay (2009). Martin Hay has extensive experience in designing and optimising MF1 and MF2 processing plants as evidenced by the publications Hay and Rule (2003), and Hay and Schroeder (2005). The capex estimate for a 100,000tpm UG2 MF2 processing was ZAR520mil (Hay, 2009).

The capital cost estimates of ZAR468,000 in 2009 money terms for locos and associated rolling stock, and ZAR54mil for surface infrastructure and tailings dam impoundment, were based on platinum projects that have recently been evaluated by Venmyn.

5.12.6 Operating costs

The operating costs captured in the discounted cashflow model in this study as reflected in the Microsoft Excel files *TechnoEconAnalysis*, are typical industry average figures based on

CBE and EPCM estimates that were sourced through the offices of Impala Platinum Projects. The labour costs were based on the average annual wages published by the Chamber of Mines, South Africa and these were extrapolated using regression analysis as shown in Figure 5.19 to obtain the labour cost per employee for the base year 2010 which was ZAR104,778.30. The detailed calculation is contained in the Microsoft Excel® file *Wage_Stats_Extrapolation*. The assumption here is that the industry average cost per employee irrespective of employee grade is more representative at a pre-feasibility level of study because the actual wage or salary for each employee tends to be individually negotiated and varies according to employee qualifications, experience and grade.

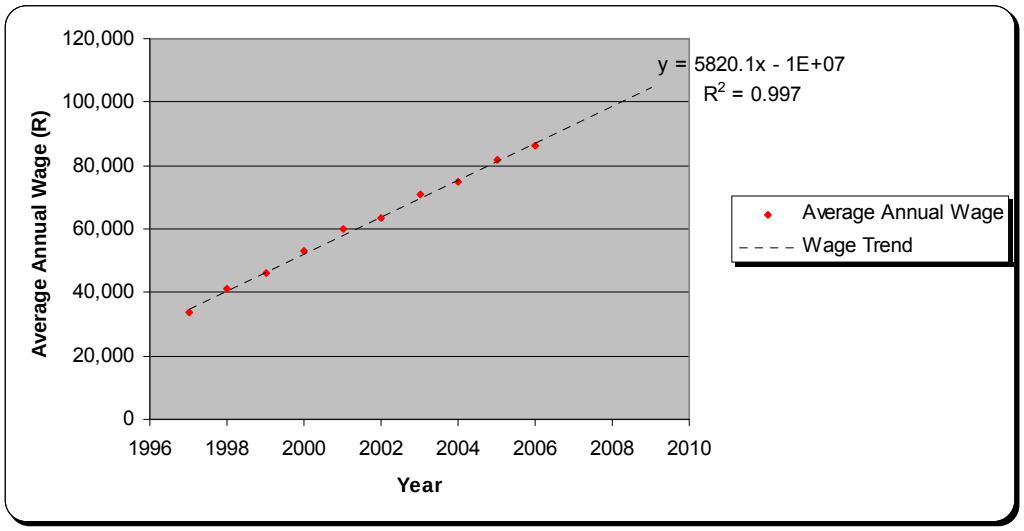


Figure 5.19: Average annual wages published by the Chamber of Mines
(Chamber of Mines, 2007)

5.12.7 Environmental rehabilitation costs

In estimating the environmental rehabilitation costs, guidance was sought from the guidelines by the Department of Minerals and Energy (2004a) and Department of Minerals and Energy (2004b). The guidelines clarify that area covered by the 'flat rate' per hectare is for every hectare that the mineral property covers, not just the portion of the mineral property that will be disturbed by the mining and mineral processing activities. Therefore the surface area covered by OB1 as determined in Chapter 4 to be 9,897,847.58m^2 was taken as the area to which the flat rate would be applied. This area is equal to about 989.78ha (Table 5.10). The flat is further adjusted to account for nature of terrain and proximity to urban area. The adjustment factors assumed for terrain and proximity to urban area are 1.0 and 1.1 respectively because the terrain for OB1 is a flat terrain defined by the wireframe model *topcut* as mentioned in Chapter 4 and OB1 is remotely located relative to nearest developed urban area.

Table 5.10: Calculation of environmental rehabilitation costs for OB1

Environmental Rehabilitation Costs	Estimate
Unit Rehabilitation cost per ha in ZAR	50,000
Terrain factor	1.0
Proximity Factor	1.1
Total projected property area in ha	989.78
PV of Rehab cost (PVA) in ZARmil	54.44

Table 5.11: Weighting factors applied to 'flat rate'
(Source: Department of Minerals and Energy, 2004b)

Terrain	Flat	Undulating	Rugged
Factor	1.00	1.10	1.20
Proximity	**Urban**	**Peri-urban**	**Remote**
Factor	1.00	1.05	1.10

The Department of Minerals and Energy (2004a) and Department of Minerals and Energy (2004b) propose that the total quantum for environmental rehabilitation can be provided for by making approved contributions to a dedicated environmental rehabilitation trust fund. The annual contributions to be made were calculated using a sinking fund annuity formula as shown in Equation 5.14.

$$PVA = A\left[\frac{(1+r)^n - 1}{r(1+r)^n}\right]$$

Equation 5.14

where *PVA* is the present value of the environmental rehabilitation quantum that must be provided, which in this case is ZAR54.44mil for all layouts, *A* is the annual payment into the sinking fund, *r* is the rate at which the fund must grow and *n* is the project LOM. It was appropriate to use r that is equal to the South African LT inflation rate of 8.85% so that the fund would grow at a rate that matches inflation. Solution of Equation 5.14 gave the required annual payment for each layout schedule since the layout schedules had different LOM. The annuity calculation for each layout is contained in the Microsoft Excel® files *TechnoEconAnalysis*.

5.13 Summary

This chapter has discussed the techno-economic assumptions that were made in order to undertake designs and schedules at a pre-feasibility level of study using Mine2-4D® and EPS® as software of choice for the study. The results obtained from the design and scheduling process are described in detail in Chapter 6.

6 TECHNO-ECONOMIC RESULTS AND VALIDATION CHECKS

6.1 Introduction

The planning parameters established in Chapter 5 were applied to OB1, the UG2 orebody that was described in Chapter 4, to develop the 15 different layouts at variable level and raise spacing. The layouts were designed in Mine2-4D® and scheduled in EPS®. The designs and schedules are contained in the directory *OB1_Final_Designs* included on the USB memory stick. Each layout took about 8 weeks to completely design and schedule. The time taken appears reasonable because typical designs with 8 half-levels and 12 raiselines when contracted out to consulting companies by mining companies can take anywhere between 3weeks-6months to complete depending on the level of detail required by the mining company and the geological complexity of the orebody (Mohloki, 2007). The scheduling results were subsequently exported and analysed in Microsoft Excel®. The analysis was done at Life of Mine (LOM) resolution for the reason mentioned in Chapter 5. Some of the hypotheses noted in Chapters 2 and 3 were tested and confirmed by the analyses done in this chapter.

This chapter initially looks at the validity of the results obtained from the design and scheduling process by performing reasonableness checks. The reasonableness checks confirm that the results were within the ±25% level of accuracy required for a pre-feasibility level of study. Since it was necessary to study the behaviour of each optimisation criterion over the range of level and raise spacing limits set in Chapter 5, yet only 15 data points were available for each criterion, it was necessary to perform interpolations and cross-interpolations using curve fitting techniques complemented by the expected behaviour of the criteria with respect to level and raise spacing, in order to fill-in gaps in the data between the set limits. The relationship that was considered most appropriate was the one where the set of curves obtained from curve fitting produced was the set having mostly highest values of the R^2 statistic. Most of the figures and underlying calculations that were done are contained in the Microsoft Excel® file *Final_Criteria_Summary*. This file is also contained on the USB memory stick. The Microsoft Excel® files *TechnoEconAnalysis* and *Average_Backlength* for each layout are also contained on the USB memory stick.

6.2 Reasonableness checks

It was necessary to undertake reasonableness checks in order to establish confidence to proceed to analyse the results obtained from the design and scheduling process. Three sets of checks were performed namely, centares discrepancy, tonnage discrepancy and valuation checks. Digitising errors sometimes occur during the design process leading to overlap of

boundaries of excavations or having boundaries that are common to two excavations but are separated by gaps between them in some sections of the common boundary. This can lead to over-estimating or under-estimating centares and tonnages. If the discrepancy in tonnages exceeds 10%, designs usually have to be re-checked for such errors. This is despite the fact that there is an in-built function in Mine2-4D® for checking overlaps and crossovers, but the software can sometimes miss small overlaps and minor crossovers, or adjust strings and points in a design to avoid data corruption arising from such errors. Therefore, it is not unusual for final designs to still have discrepancies in centares and tonnages when compared with *in-situ* estimates.

6.2.1 Centares discrepancy

In order to check for centares discrepancy, the total *in-situ* stope centares were added to the sum of areas left *in-situ* as regional pillars and geological losses, and then compared to the original *in-situ* centares for the OB1 wireframe. The original *in-situ* centares for OB1 were estimated in Chapter 4 to be 10,053,061m^2. The centares discrepancy was then calculated using Equation 6.1.

$$CentaresDiscrepancy(\%) = \frac{originalinsitucentares - (stopeinsitucentares + regionalpillarsarea + geologicallossesarea)}{originalinsitucentares} * 100$$

Equation 6.1

Centares discrepancy calculations are contained in the Microsoft Excel® files *TechnoEconAnalysis*. Table 6.1 shows the centares discrepancies obtained for each layout. Since all the discrepancies were within ±25%, the results could be accepted as reliable for further analyses.

Table 6.1: Summary centares discrepancy for the 15 layouts

		Raise Spacing (m)				
		180	200	280	360	400
Level Spacing (m)	30	0.07%	0.10%	0.03%	0.03%	0.04%
	50	0.24%	-0.02%	0.07%	-0.99%	0.05%
	67	0.06%	0.01%	0.06%	0.12%	-0.23%

6.2.2 Tonnage discrepancy

In order to check for tonnage discrepancy, the total *in-situ* stope tonnes were added to the sum of tonnes left *in-situ* as regional pillars and geological losses, and then compared to the original *in-situ* tonnes for the OB1 wireframe. The original *in-situ* tonnes for OB1 were

estimated in Chapter 4 to be 40,212,244t. The discrepancy was then calculated using Equation 6.2.

$$Tonnes\ Discrepancy(\%) = \frac{original\ insitu\ tonnes - (stope\ insitu\ tonnes + regional\ pillar\ tonnes + geological\ loss\ tonnes)}{original\ insitu\ tonnes} * 100$$

Equation 6.2

Tonnage discrepancy calculations are also contained in the Microsoft Excel® files *TechnoEconAnalysis*. Table 6.2 shows the tonnage discrepancies obtained for each layout. Since all the discrepancies were within ±25%, the results could be accepted as reliable for further analyses. A comparison of Table 6.1 and Table 6.2 shows that on average the tonnage discrepancies are about 4 times the centares discrepancies, confirming the tonnage factor of 4t/m^2 mentioned in Chapters 4 and 5.

Table 6.2: Summary tonnage discrepancy for the 15 layouts

		Raise Spacing (m)				
		180	200	280	360	400
Level Spacing (m)	30	0.28%	0.30%	0.24%	0.23%	0.24%
	50	0.44%	0.19%	0.27%	-0.79%	0.26%
	67	0.27%	0.22%	0.26%	0.33%	-0.02%

6.2.3 Valuation check

Venmyn has over the past five years, developed a platinum valuation curve for use when valuing platinum mineral assets. The platinum curve was developed based on the market approach principle of 'willing buyer, willing seller' and requires that the amount obtainable from the sale of an asset should be determined as if the transaction was an arm's length transaction. The curve is compiled and regularly updated from a comprehensive database of relatively recent transactions of platinum mineral assets and current market capitalisation of PGM projects available from the public domain or from transactions in which Venmyn or their associate companies would have participated in. The platinum mineral assets are valued in monetary value per unit of resource ounce (US$/oz) and sorted according to the development stage of the project which can be the inferred, indicated or measured mineral resource categories or, probable or proven mineral reserve categories. The mineral assets are further sorted according to their location on the Bushveld Complex. The transactions used to construct the valuation curve occurred at specific points in time and therefore at specific PGE basket prices and US$/ZAR exchange rates, which are then adjusted to current PGE prices and exchange rates.

A difficulty of this approach in the mining industry is that there are no true comparable transactions to arrive at a Fair Market Value (FMV), unlike in Real Estate, Oil, or Gas sectors where many comparable transactions exist, because each mineral asset is unique with respect to key factors such as location, geology, mineralisation and reef type (e.g. UG2 or Merensky), exploration costs incurred, stage of development and infrastructure already in place. Snowden (2009) highlight this same challenge that is faced by competent valuators when deriving a FMV for a mineral property. These factors contribute to produce the generalised valuation curve which is a nearly lognormal band of US$/oz across all resource and reserve categories as illustrated in Figure 6.1. This curve, when applied to Bushveld Complex transactions provides general guidance in terms of a range of transaction values that can be considered. The Net Present Values (NPVs) from the discounted cash flow (DCF) valuation of each of the 15 layouts were then normalised to US$/oz and plotted on the valuation curve as shown in Figure 6.1. The calculations of the US$/oz values are contained in the Microsoft Excel® files, *TechnoEconAnalysis* for each layout. These values plotted on the lower band of the Measured Resource category which is typical of UG2 properties on the Bushveld Complex in this resource category, because as indicated in Chapters 4 and 5, a typical UG2 prill split contains less platinum than a Merensky prill split. Therefore, the economic and financial assumptions used in this thesis for the valuation process could be considered reliable enough to proceed with further analyses.

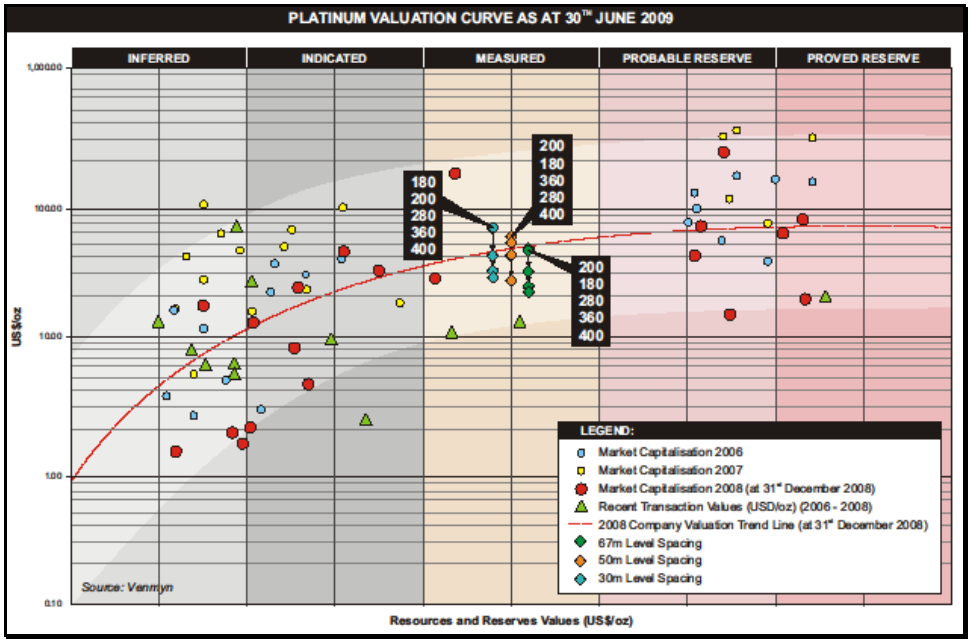


Figure 6.1: Relative positions of the 15 layouts on the platinum valuation curve (Courtesy of Venmyn)

It is discernible from Figure 6.1 that the layouts at 180m and 200m raise spacing plot on nearly the same point indicating that there is no significant change in resource value when

raise spacing is altered by increments of up to 20m. Therefore when considering raise spacing changes, it makes economic sense to change the spacing in increments of at least 20m. This fact was considered in doing the 3-D surface contour plots of the final AHP priority scores in Chapter 7. Figure 6.1 also shows that the mineral asset value decreases as level spacing is increased, suggesting that there is no economic merit in terms of value creation when level spacing is increased. This could be as a result of the slow build-up to full production at longer level spacing that consequently leads to lower and delayed returns.

6.3 General results

The previous section indicated that the results from the design and scheduling process were reliable enough to be used for further analyses. This section and subsequent sections, present the results of other analyses done on the data in order to get further insights into the problem of optimising level and raise spacing. Some of the results obtained from the design and scheduling process which are not part of the criteria used for optimisation but help to understand some of the hypotheses discussed in earlier chapters are analysed in the next sub-sections.

6.3.1 Backlengths at fixed vertical level spacing

The lengths of individual backlengths on each raiseline for each of the 15 layouts were measured using the *query string* function in Mine2-4D®. The Microsoft Excel® files *Average_Backlength* contain the individual backlengths obtained from this process. For each layout, the backlengths obtained were quite variable in size because the surface configuration of the orebody is not regular due to variable geology throughout the entire orebody as can be observed from the 3-D configurations of OB1 that were presented earlier in Chapter 4. This finding confirms the argument in Chapter 2 that it was inappropriate for Lawrence (1984) to assume a fixed backlength because backlength is variable for a fixed level spacing. A summary of the range of backlengths obtained is shown in Table 6.3.

When the average backlengths are compared with the calculated backlength for a constant reef dip of 9.6º at the level spacing of 30m, 50m and 67m that were selected for this study, there is an average deviation of 4.5% from the expected backlength (Table 6.4). Again, this deviation is attributable to the variable geology throughout the entire orebody which increases the dip distance along a raiseline on the actual reef horizon.

Table 6.3: Summary of ranges of backlengths obtained for the 15 layouts

Vertical level spacing (m)	Raise spacing (m)	Minimum backlength (m)	Maximum backlength (m)	Average backlength (m)
67	400	382.25	501.77	418.98
67	360	338.93	537.11	419.06
67	280	344.77	557.82	419.76
67	200	322.47	531.12	418.25
67	180	320.26	558.20	420.07
50	400	277.87	391.82	314.40
50	360	249.46	413.04	314.99
50	280	255.81	457.03	314.10
50	200	233.29	450.54	313.37
50	180	243.81	420.93	310.49
30	400	153.88	268.23	188.19
30	360	133.88	279.37	188.29
30	280	137.15	286.51	187.86
30	200	125.99	285.72	188.04
30	180	122.86	290.26	187.81

Table 6.4: Deviation of actual backlengths from calculated backlengths at constant reef dip of 9.6º

Level spacing (m)	Actual average backlength (m)	Calculated backlength @ 9.6º (m)	Deviation
67	419.23	402	4.3%
50	313.47	300	4.6%
30	188.04	180	4.5%
		Average	**4.5%**

6.3.2 Total development metres in relation to level and raise spacing

When the total development metres were plotted at variable level and raise spacing, the best fit relationship obtained was a power function in which total development metres decrease asymptotically with increasing level and raise spacing (Figure 6.2 and Figure 6.3), confirming the observations made in Chapter 2.

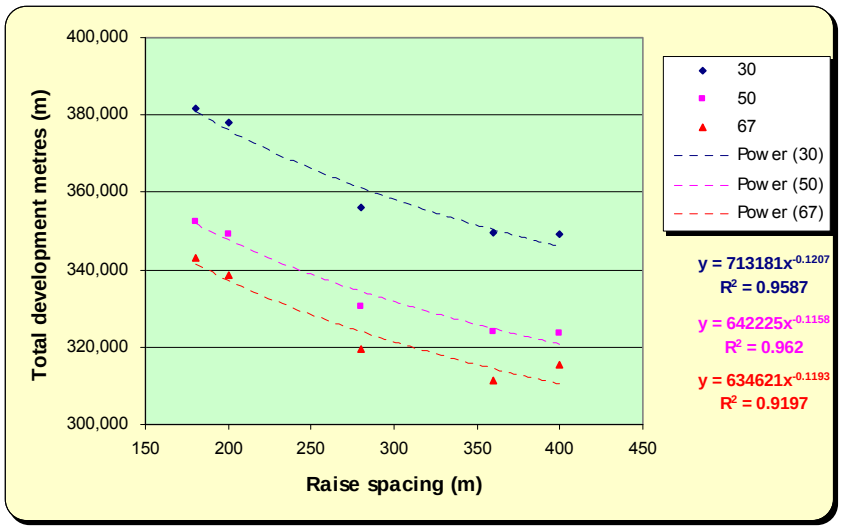


Figure 6.2: Variation of total development metres with raise spacing

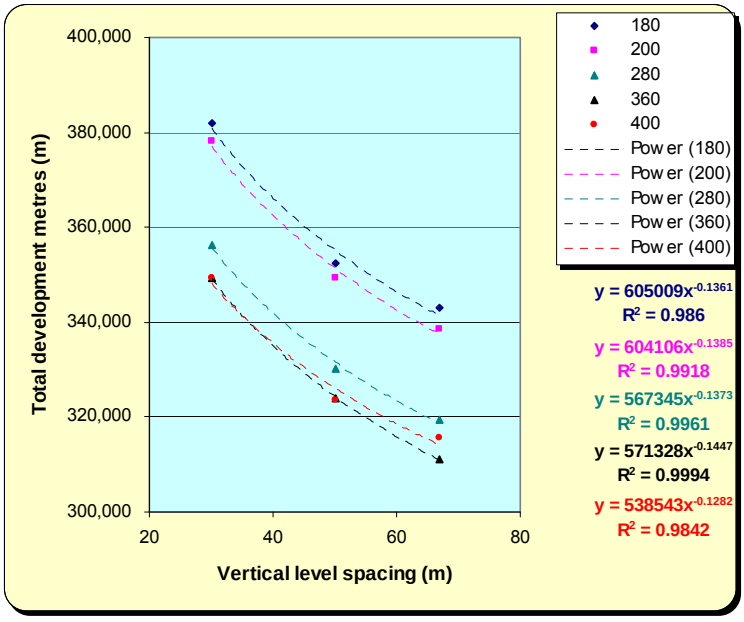


Figure 6.3: Variation of total development metres with level spacing

6.3.3 Number of levels

When the number of levels was plotted against increasing vertical level spacing, the best fit relationship obtained was a power function in which the number of levels decreases asymptotically with increasing vertical level spacing (Figure 6.4), confirming the analogy drawn in Chapter 2, between dividing a line repeatedly into equal parts to dividing dip distance repeatedly into smaller level intervals or dividing strike distance repeatedly into

smaller raise spacing intervals. The number of levels was obtained using the *wireframe multiple slicing* functionality in Mine2-4D®.

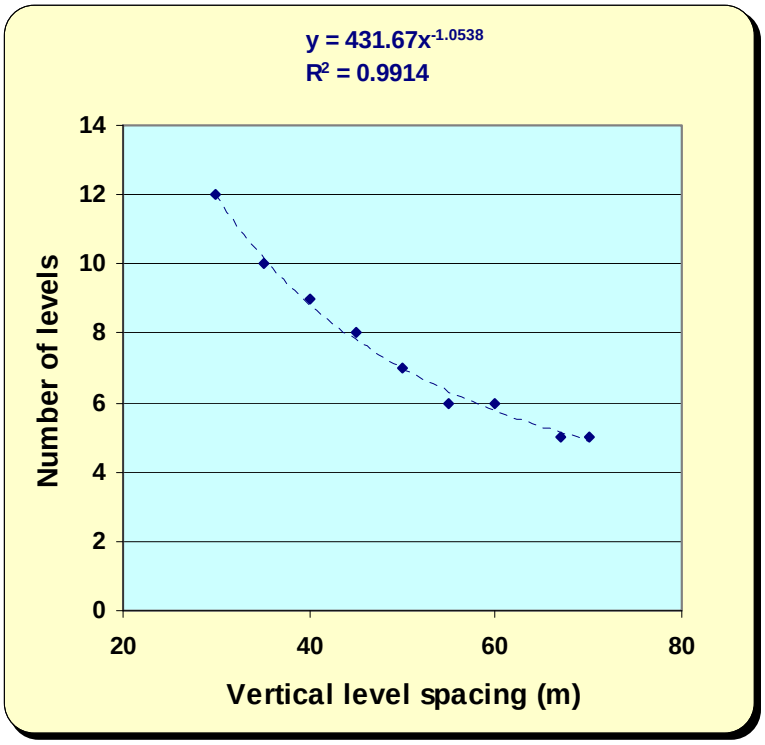


Figure 6.4: Variation of number of levels with increasing vertical level spacing

6.3.4 Number of stopes

The number of laybye connections was used as a proxy for the number of stopes because each laybye connection services a single stope, although a stope may be further divided by geological discontinuities into blocks within the stope (Figure 6.5). As expected, when the number of stopes was plotted against increasing raise spacing, the best fit relationship obtained was a power function in which the number of stopes decreases asymptotically with increasing raise spacing (Figure 6.6), again confirming the observations made in Chapter 2.

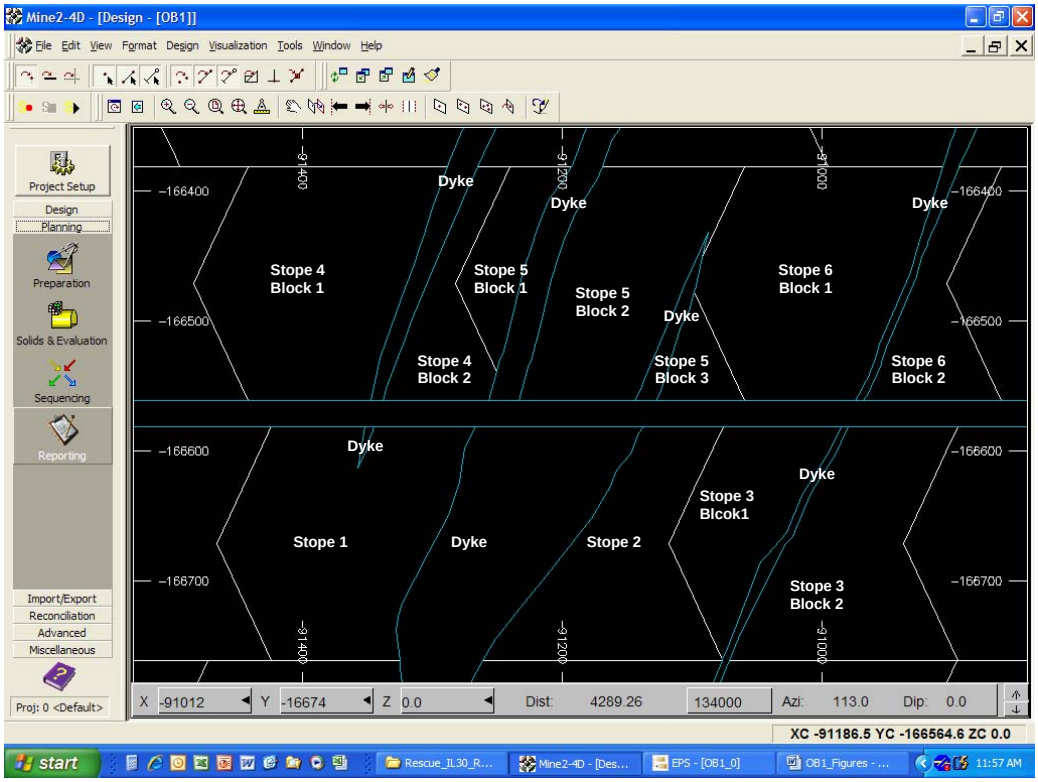


Figure 6.5: A Mine2-4D® illustration of OB1 stopes divided into blocks by dykes

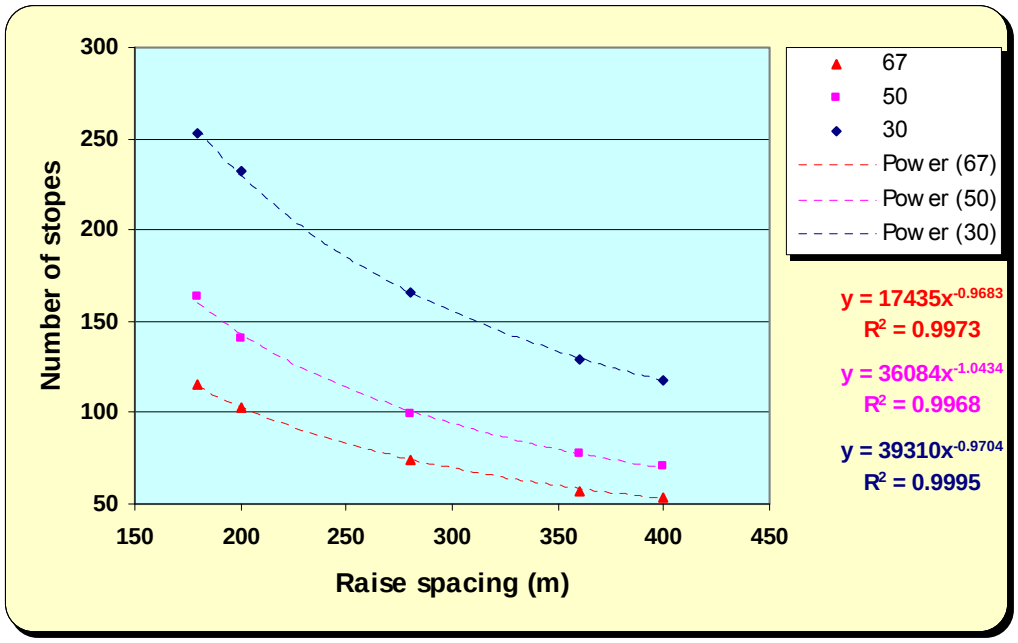


Figure 6.6: Variation of number of stopes with increasing raise spacing

6.3.5 Redundant criteria

It is standard practice in experimental work to have controls set over justifiably chosen factors so that the behaviour of other variables can be studied. For example, Lawrence (1984) fixed backlength at 180m and production rate at 80,000ca per month; Vieira, Diering and Durrheim (2001) fixed production rate at 45,000m^2 of reef per month; and Egerton (2004) fixed production rate at 100,000tpm of ROM ore. In AHP analysis, a criterion that is held constant or is almost constant for all the alternatives under consideration is regarded as a redundant criterion and can be excluded from the final analysis.

Since altering level and raise spacing directly influences productivity and production, it was decided for this study, not to put controls on production rate but rather on other criteria. It was appropriate therefore to put controls on capital development since the primary means of accessing the orebody was the same in all cases as noted in Section 5.12.5 and additionally, most of the mining rate for each type of development is almost constant as is not always directly affected by spacing but by the technology in use. For all layouts the PV of total capital development amounted to ZAR796.03mil as indicated in the Microsoft Excel® files *TechnoEconAnalysis*. The in-panel areal extraction factor, exclusive of geological losses was assumed to be the same for all layouts since the same mining method was used for all 15 layouts. This resulted in overall areal extraction factors that were nearly the same for all layouts at about 81% as shown in Table 6.5. These area extraction factors may appear a bit high when compared to factors being obtained from existing UG2 operations where the average extraction rate is about 44% (Impala Platinum, 2008). This is due to the fact that the area extraction factors derived in this study could not include detailed local geological losses in the form of small faults, potholes and dykes which cannot be picked at the geological exploration stage, as these are picked up during the actual mining of the reef.

Table 6.5: Area extraction percent for all 15 layouts

		Raise spacing (m)				
		180	200	280	360	400
Level spacing (m)	30	82.19%	82.79%	80.60%	81.55%	82.58%
	50	80.93%	82.25%	80.69%	81.56%	82.61%
	67	81.86%	82.36%	80.42%	80.68%	82.88%

Another factor that was not considered in the final analysis was the production tailing-off period since it was not highlighted in the industry survey undertaken as described in Chapter 7, despite a caveat asking respondents include any other criteria they considered important in evaluating mining layouts for narrow tabular reefs. A reason for this could be due to the

fact noted in one of the feedback comments in Appendix 10.3 that the production tail cannot be accurately planned and therefore difficult to determine since it occurs at the end of life of a project (Impala Platinum, 2009). However, if this criterion was included in the analysis, it would have a similar impact as build-up period which favours designs at lower level and raise spacing as established by this study. Table 6.6 and Table 6.7 show similar trends of decreasing time with increasing spacing for both build-up and tail-off periods which could be explained by the fact that at larger level spacing it is not possible to have replacement levels when those currently being mined are exhausted making it difficult to have a rapid tail-off. A similar effect can be expected at wider spacing of raises. The combined effect of wider level and raise spacing can therefore be noticed in the long drawn-out tails as obtained from this study.

Table 6.6: Estimates of build-up period in years for OB1 layouts

		Raise Spacing (m)				
		180	200	280	360	400
Level Spacing (m)	30	8	9	12	11	12
	50	9	12	13	14	15
	67	10	11	11	14	20

Table 6.7: Estimates of production tail-off period in years for OB1 layouts

		Raise spacing (m)				
		180	200	280	360	400
Level spacing (m)	30	10	12	18	25	23
	50	10	18	22	38	33
	67	16	23	23	34	35

Therefore, for this study the capital cost, area extraction percent, and development rates were redundant criteria.

6.4 Trends in optimisation criteria

The optimisation criteria identified in Chapter 2 were analysed to check how they behaved in relation to increasing level and raise spacing. These criteria are PV of development cost per centare mined, project NPV, project payback period, replacement factor (RF), shaft head grade, production rate, productivity, flexibility index, life of raiseline, LOM and build-up period. The results of the relationships obtained are presented in the next sections.

6.4.1 PV of development costs per centare

The development costs that were analysed were exclusive of capital development costs, which cost was constant across all the 15 layouts as indicated in Section 6.3.5. The PV of the development costs was reported in ZAR/m^2. The variation of PV of development costs with level and raise spacing is illustrated by Figure 6.7 and Figure 6.8, respectively.

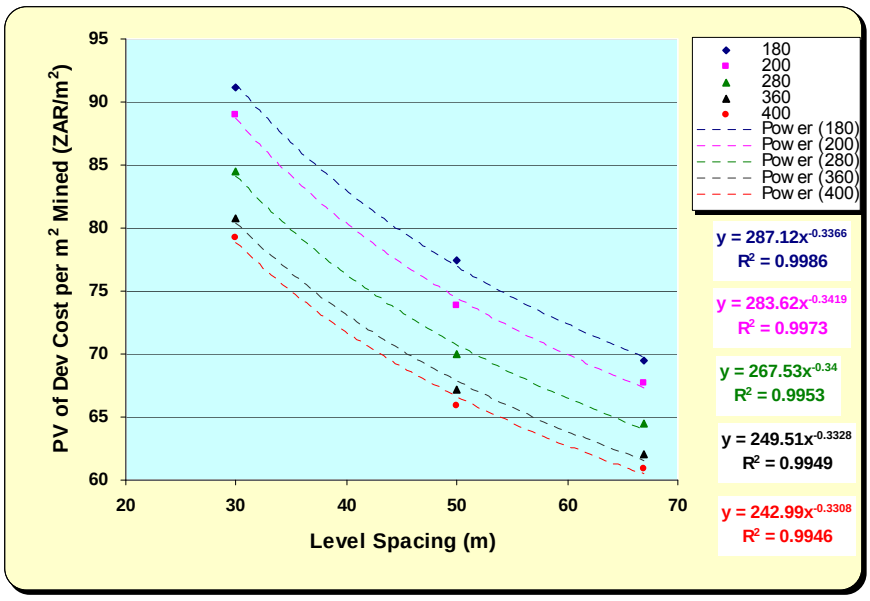


Figure 6.7: Variation of PV of development cost per centare mined in relation to level spacing

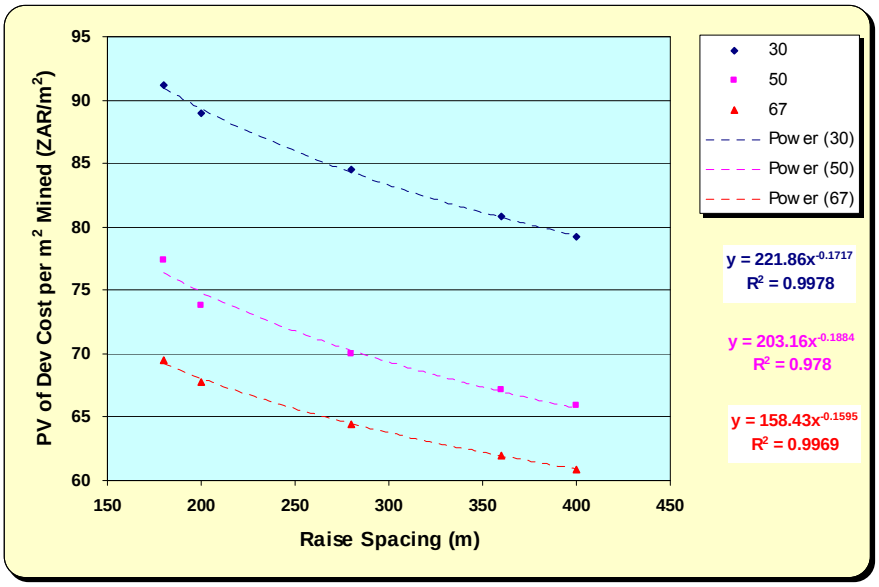


Figure 6.8: Variation of PV of development cost per centare mined in relation to raise spacing

If Figure 6.8 is used to estimate the cost saving associated with the Impala Platinum decision to increase raise spacing to 220m if they were mining OB1 on 50m vertical level spacing and 180m raise spacing (at development costs of ZAR77.40/m^2), then this change could result in development cost savings of about ZAR3.86/m^2 at a present value to ZAR73.54/m^2. In percentage terms this implies that a 22% increase in raise spacing (from 180m to 220m) can lead to about 5% saving in development cost per m^2 mined in present value terms.

6.4.2 Project NPV

The best fit obtained for the relationship between project NPV and increasing level and raise spacing was quadratic as depicted in Figure 6.9 and Figure 6.10. This quadratic trend can be explained using Eaton's (1934) argument that beyond a certain level or raise spacing, the cost saving benefit associated with reducing the amount of development is more than off-set by the cost of mining at longer distances.

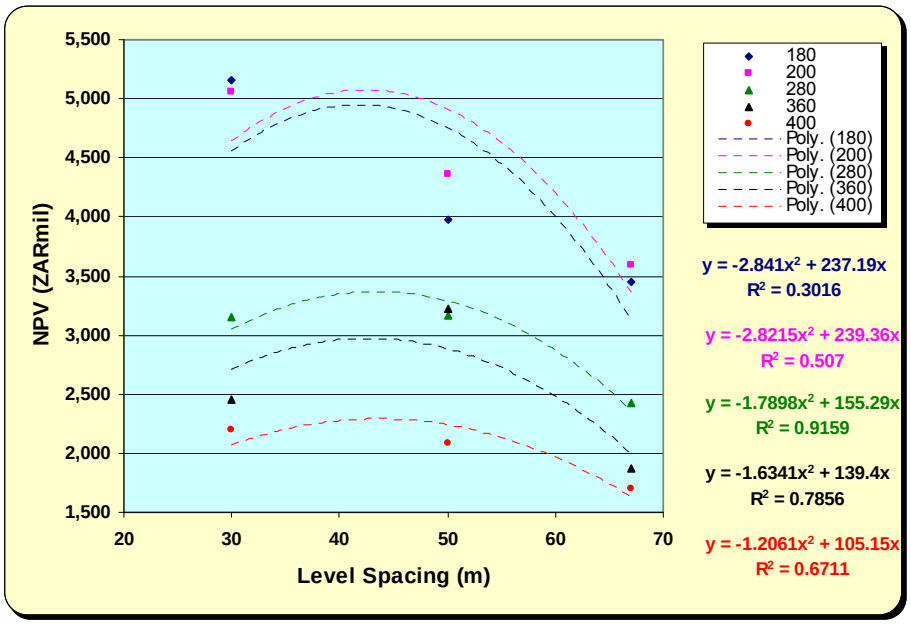


Figure 6.9: Variation of project NPV with increasing level spacing

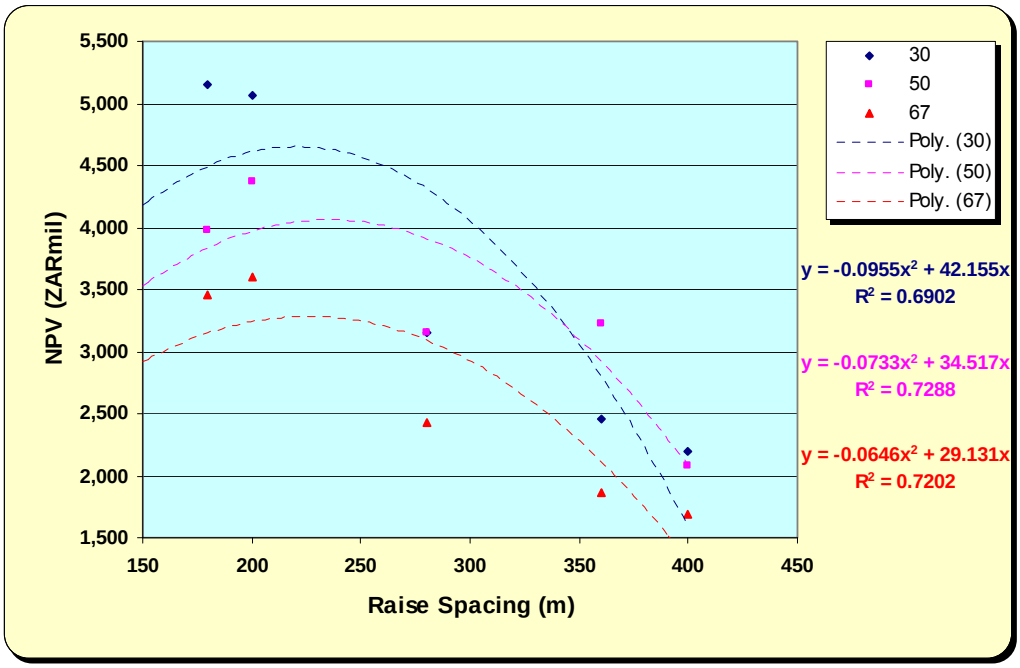


Figure 6.10: Variation of project NPV with increasing raise spacing

It can be noticed from Figure 6.9 that the optimal range of vertical level spacing for OB1 is between 40m-50m. Figure 6.10 shows that the optimal range of raise spacing for OB1 is between 200m-250m, a range which confirms Lawrence's (1984) findings on the economic optimal raise spacing for conventional breast mining as discussed in Chapter 2, if NPV was the sole optimisation criterion.

6.4.3 Payback period

The best fit for the behaviour of the payback period with respect to increasing level and raise spacing was exponential as indicated in Figure 6.11 and Figure 6.12. No feasible explanation could be made at this stage to explain why payback period should have such a relationship with increasing level and raise spacing.

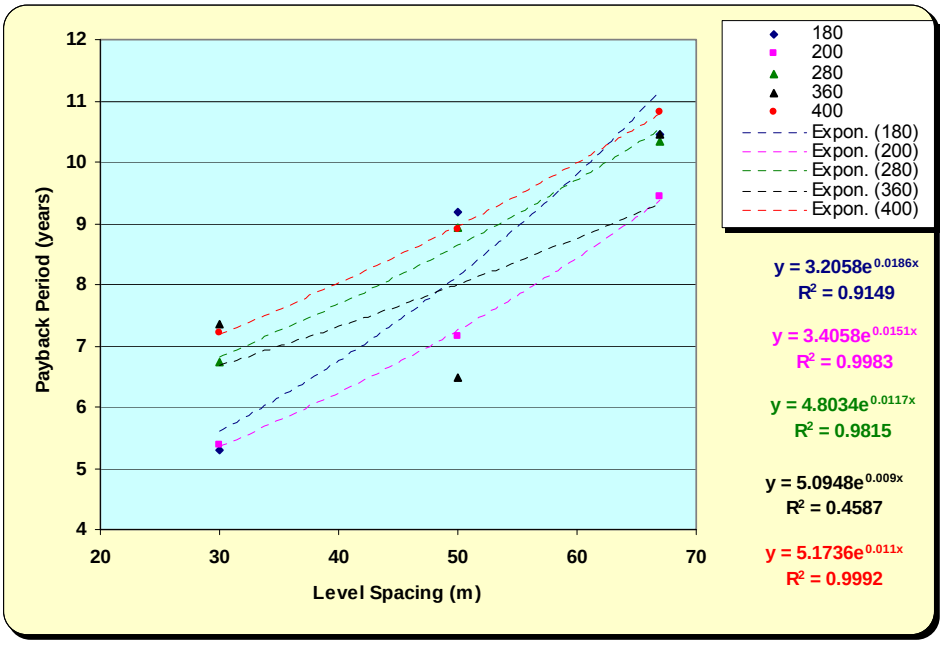


Figure 6.11: Variation of payback period with level spacing

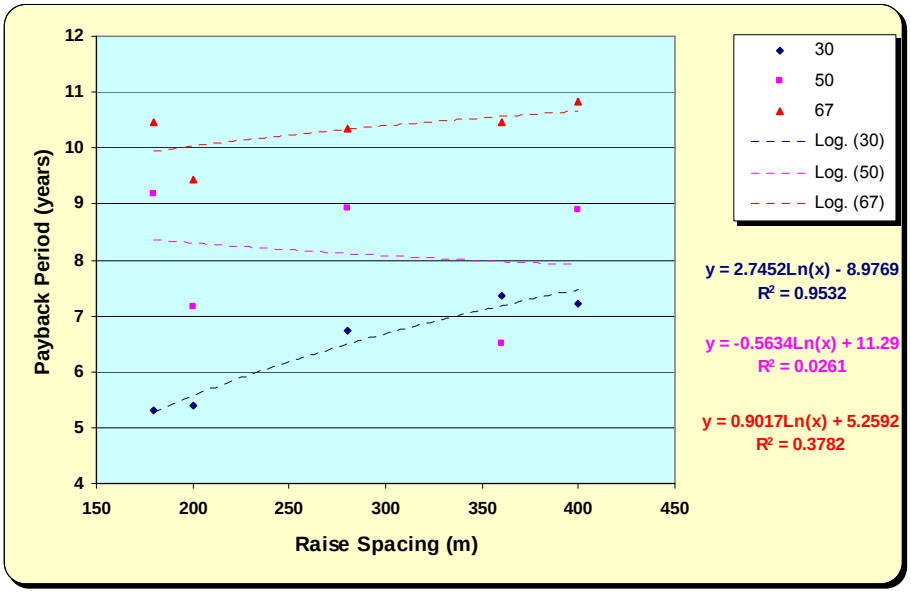


Figure 6.12: Variation of payback period with increasing raise spacing

6.4.4 Replacement Factor (RF)

The RFs obtained ranged between 21.6m^2/m-26.4m^2/m, a range which is consistent with the observations made by Fleming (2002) for conventional mining on narrow tabular reefs, as highlighted earlier on in Section 5.4. As expected the RF increased linearly with increasing level and raise spacing as depicted by Figure 6.13 and Figure 6.14.

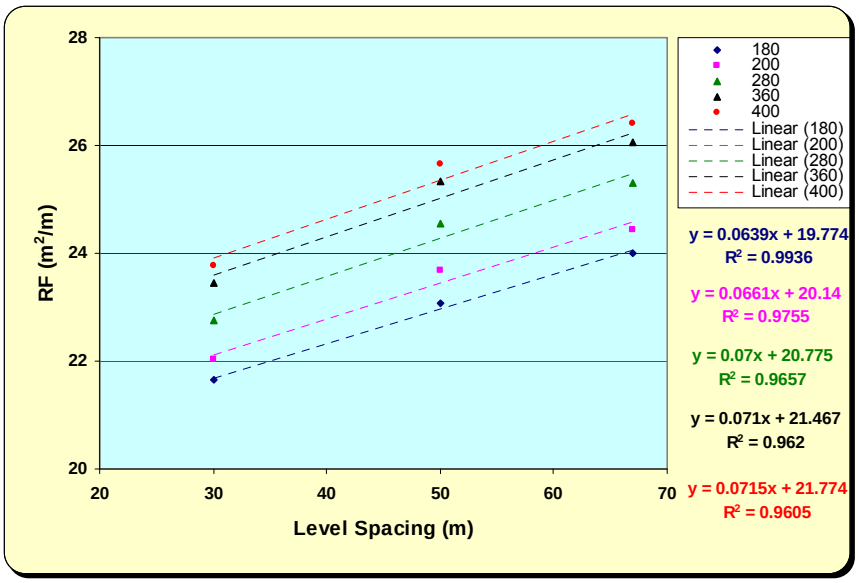


Figure 6.13: Variation of RF with level spacing

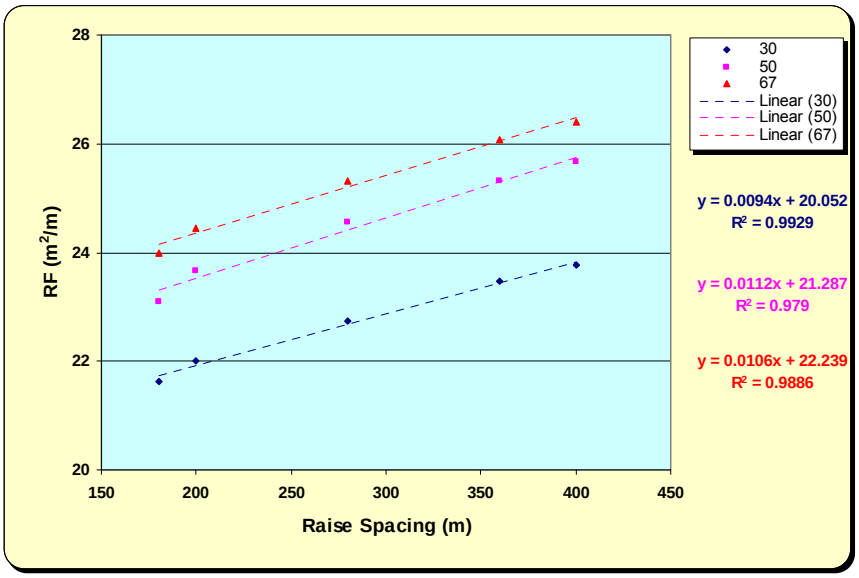


Figure 6.14: Variation of RF with increasing raise spacing

6.4.5 Shaft head grade

Shaft head grade is not affected by level spacing (Table 6.8) because all the material that is blasted from level development and off-reef development is trammed separately as waste. Shaft head grade therefore has a uniform relationship with level spacing. However, shaft head grade is affected by raise and winze spacing because development ore from raises and winzes is scraped together with ore from production panels. The impact of reduced dilution from more spaced out raises is not very significant as Table 6.8 shows that grade

only changes from 5.50g/t to 5.53g/t. Therefore, only the relationship between shaft head grade and raise spacing was analysed. The best fit for the relationship that was obtained was a logarithmic fit as indicated in Figure 6.15. It has not been possible at this stage to understand why the relationship should be logarithmic.

Table 6.8: Shaft head grade constant with respect to increasing level spacing for all 15 layouts

shaft head grade (g/t)		Raise Spacing (m)				
		180	200	280	360	400
Level Spacing (m)	30	5.50	5.50	5.52	5.53	5.53
	50	5.50	5.51	5.52	5.53	5.53
	67	5.50	5.51	5.52	5.53	5.53

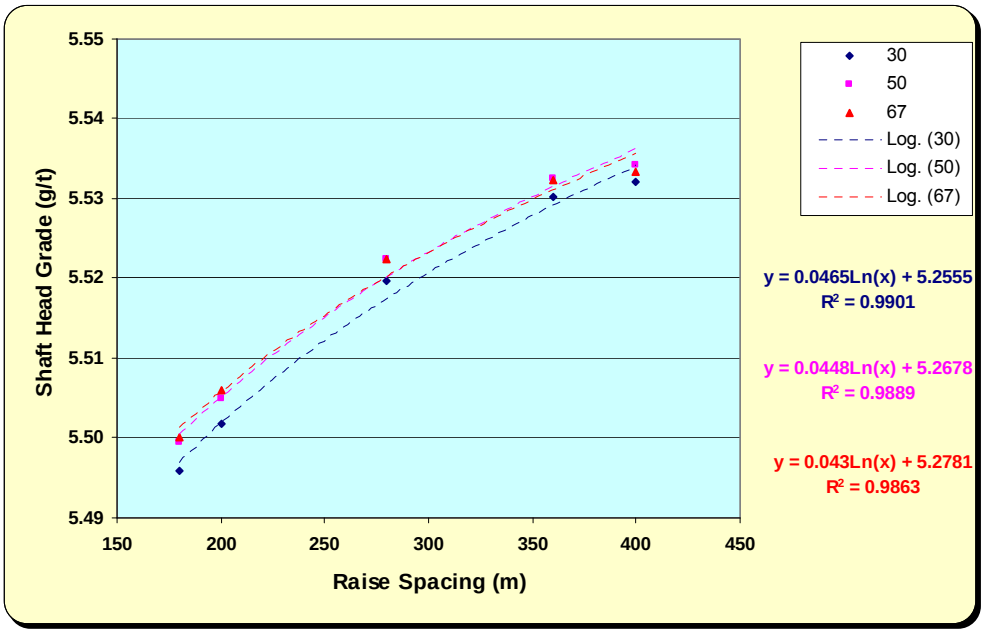


Figure 6.15: Variation of shaft head grade with increasing raise spacing

6.4.6 Productivity

The productivity decreased with increasing level and raise spacing as depicted by Figure 6.16 and Figure 6.17. This trend confirms the work of Brassell (1964) and that of Lawrence (1984). The productivity figures obtained were between 30m^2/stope employee-40m^2/stope employee which agrees closely with productivity figures prevailing on the Bushveld Complex.

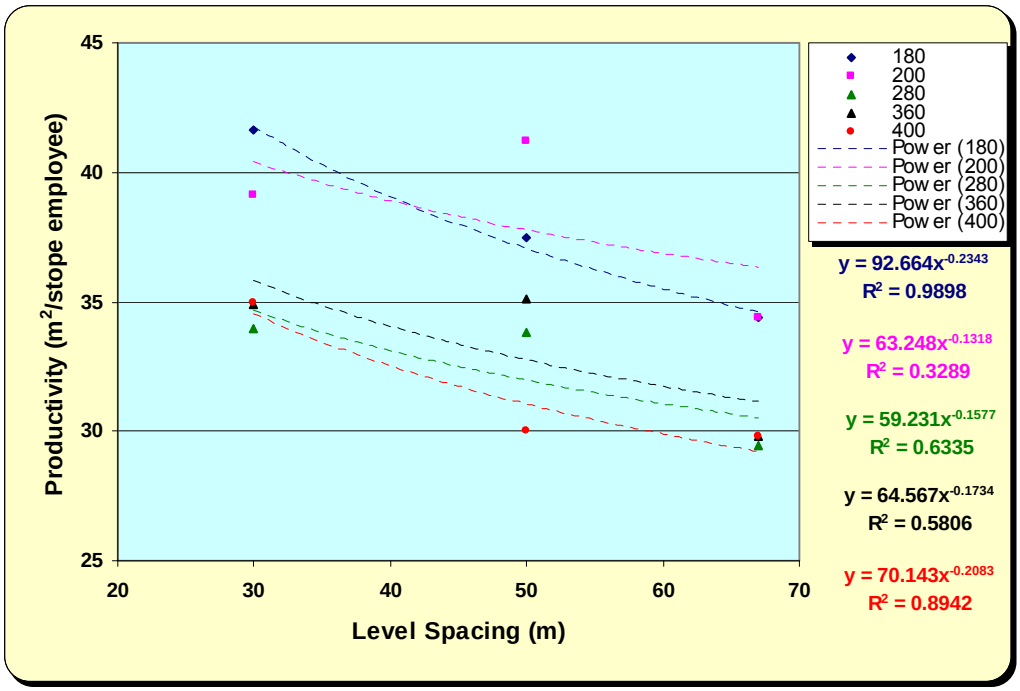


Figure 6.16: Variation of productivity with increasing level spacing

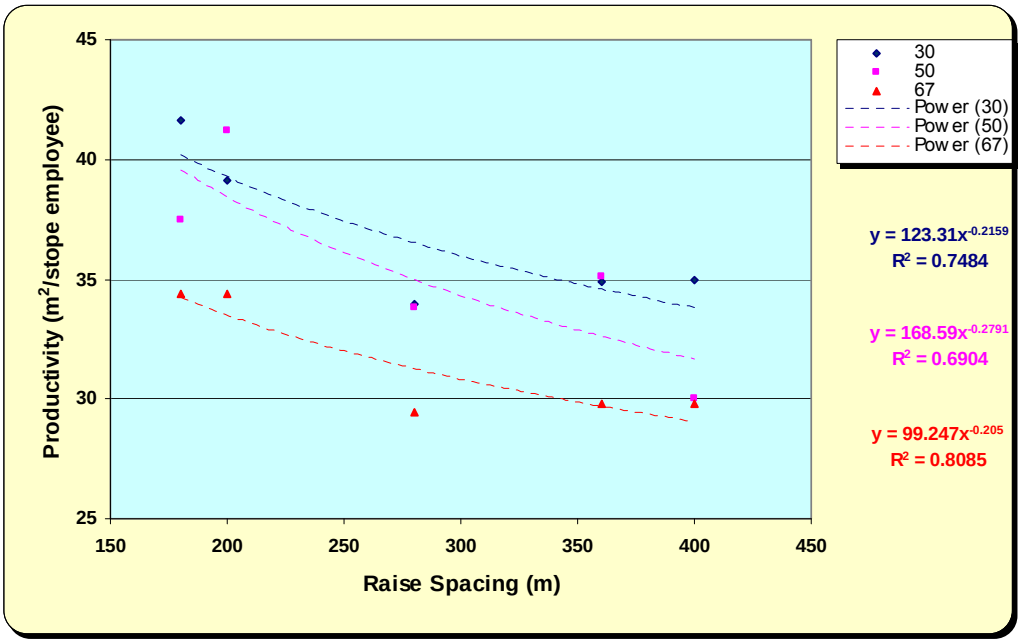


Figure 6.17: Variation of productivity with increasing raise spacing

6.4.7 Production rate

The production rate is a function of productivity and is therefore expected to exhibit a power relationship with increasing level and raise spacing as depicted by Figure 6.18 and Figure 6.19.

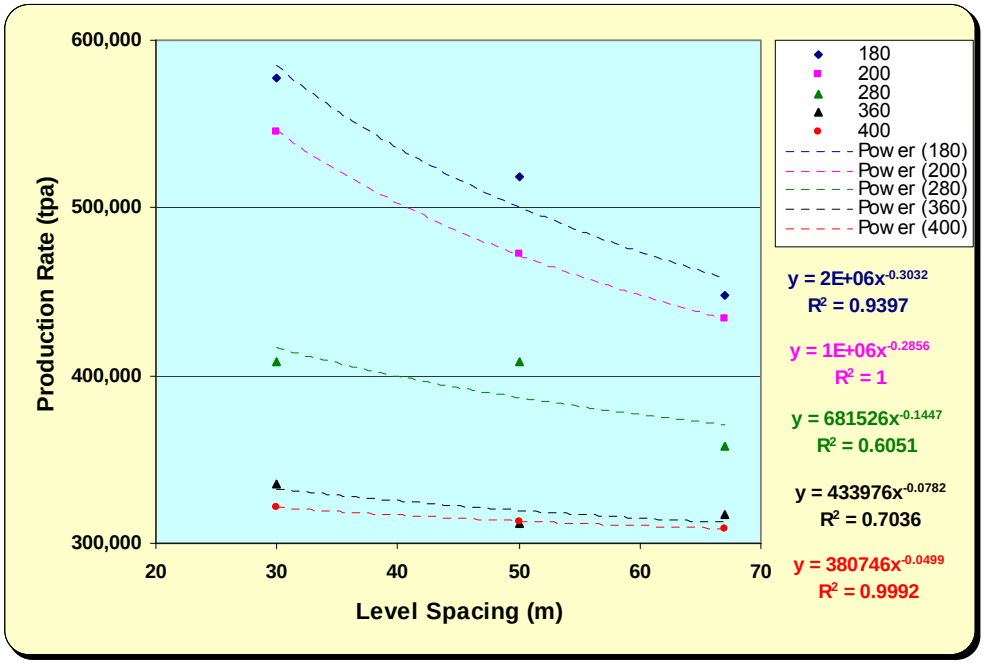


Figure 6.18: Variation of production rate with increasing level spacing

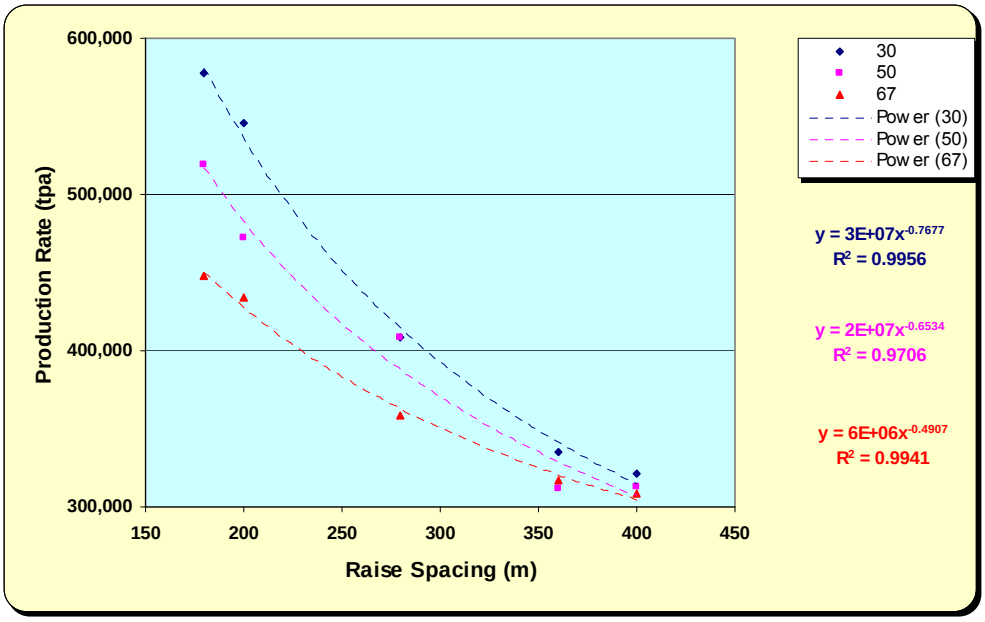


Figure 6.19: Variation of production rate with increasing raise spacing

6.4.8 Flexibility index (FI)

The *FI* is a function of the number of stopes. The number of stopes exhibited a power function relationship with level and raise spacing. Therefore *FI* can be expected to exhibit a power function relationship with level and raise spacing as indicated in Figure 6.20 and Figure 6.21.

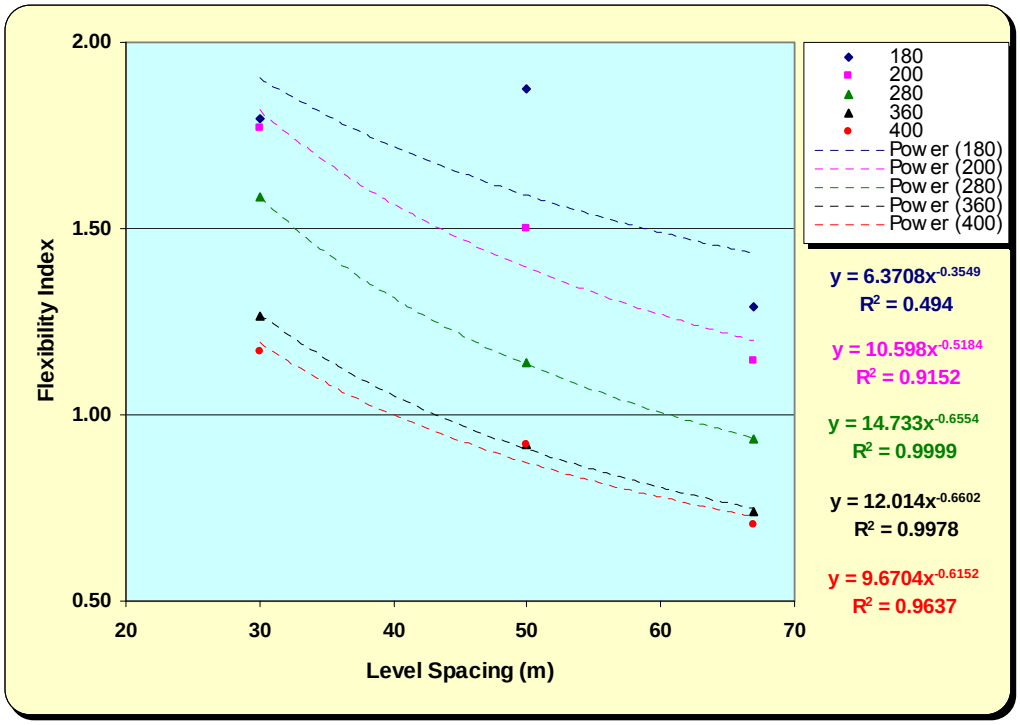


Figure 6.20: Variation of flexibility index with level spacing

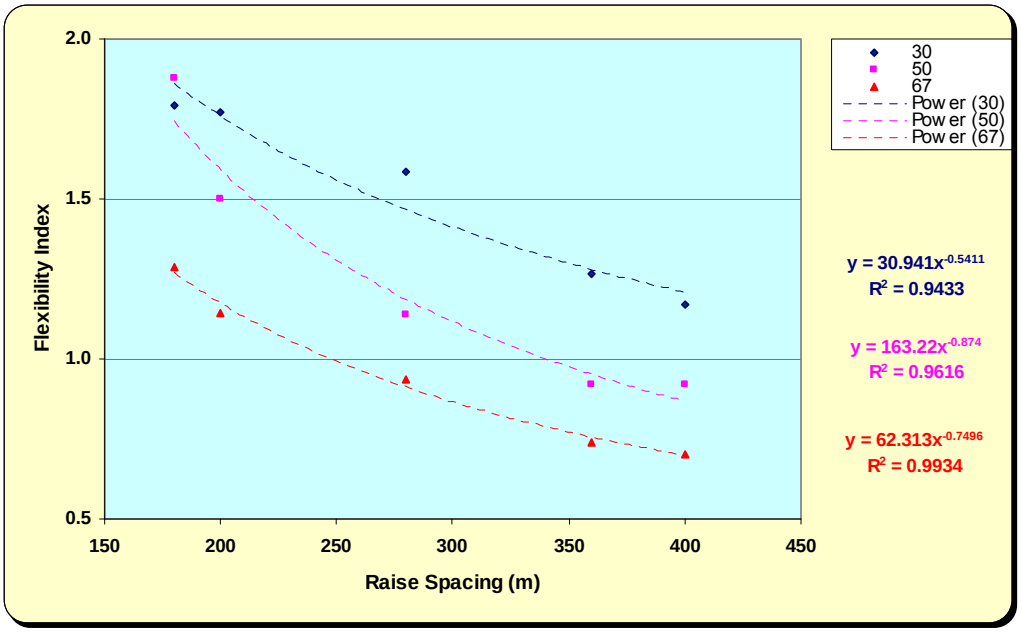


Figure 6.21: Variation of flexibility index with raise spacing

.

6.4.9 Life of raiseline

Life of raiseline had a linear relationship with level and raise spacing as shown by Figure 6.22 and Figure 6.23.

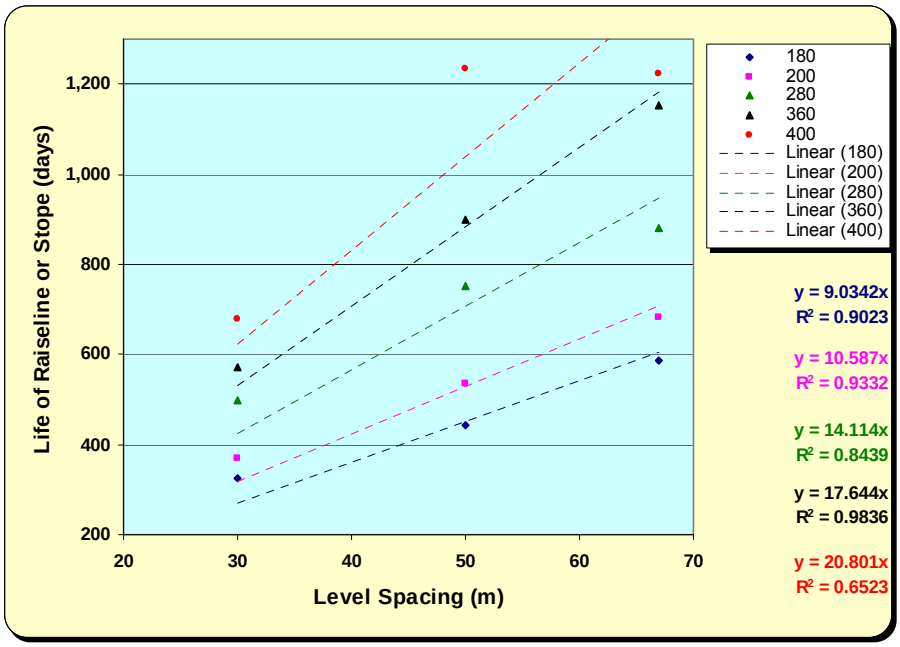


Figure 6.22: Variation of life of raiseline with increasing level spacing

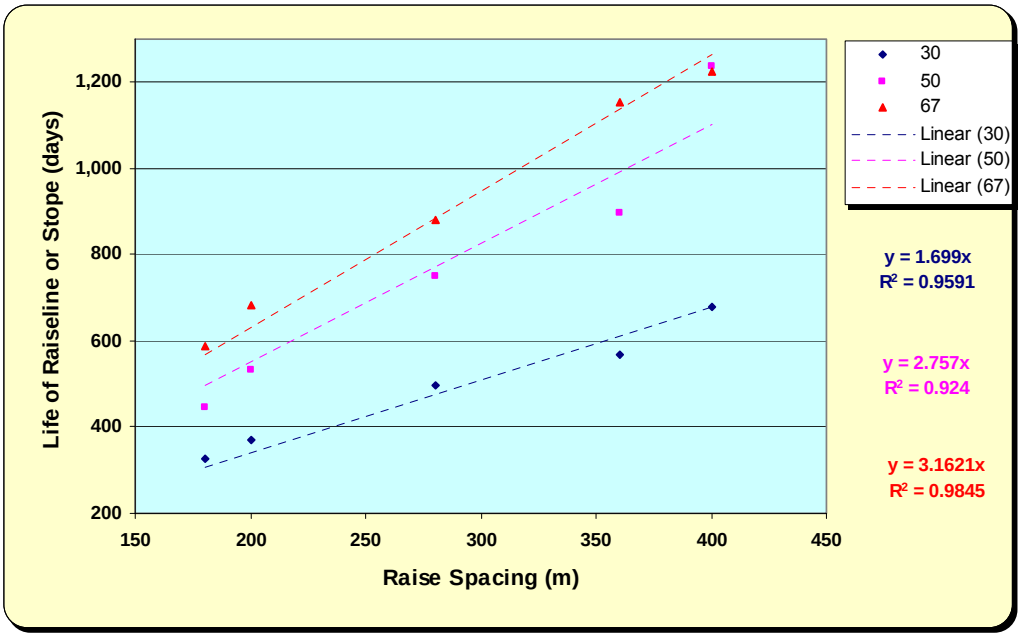


Figure 6.23: Variation of life of raiseline with increasing raise spacing

6.4.10 Life of mine (LOM)

Figure 6.24 and Figure 6.25 reveal that the relationship between LOM and level and raise spacing is linear.

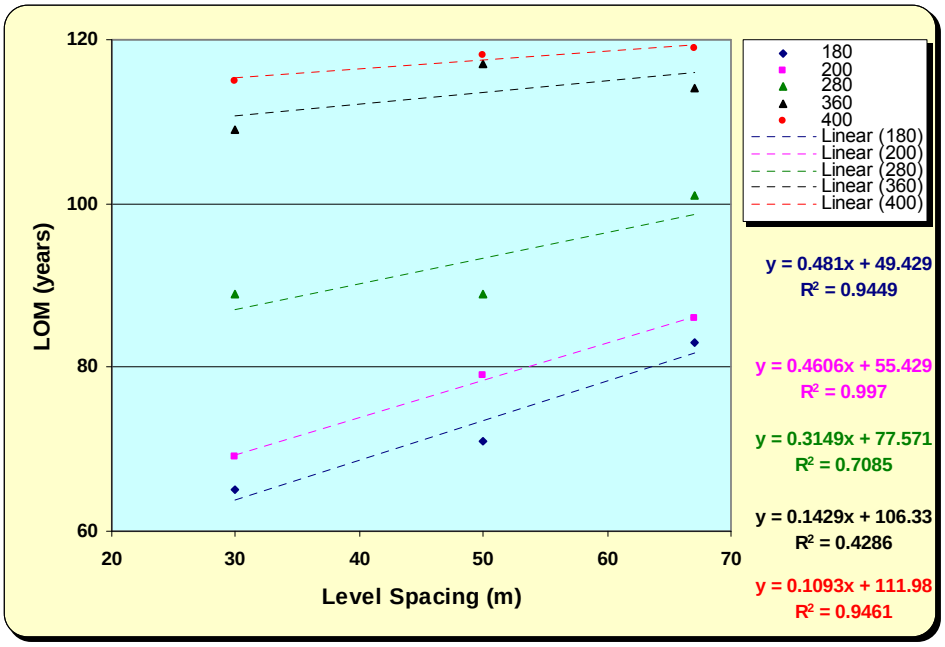


Figure 6.24: Variation of LOM with increasing level spacing

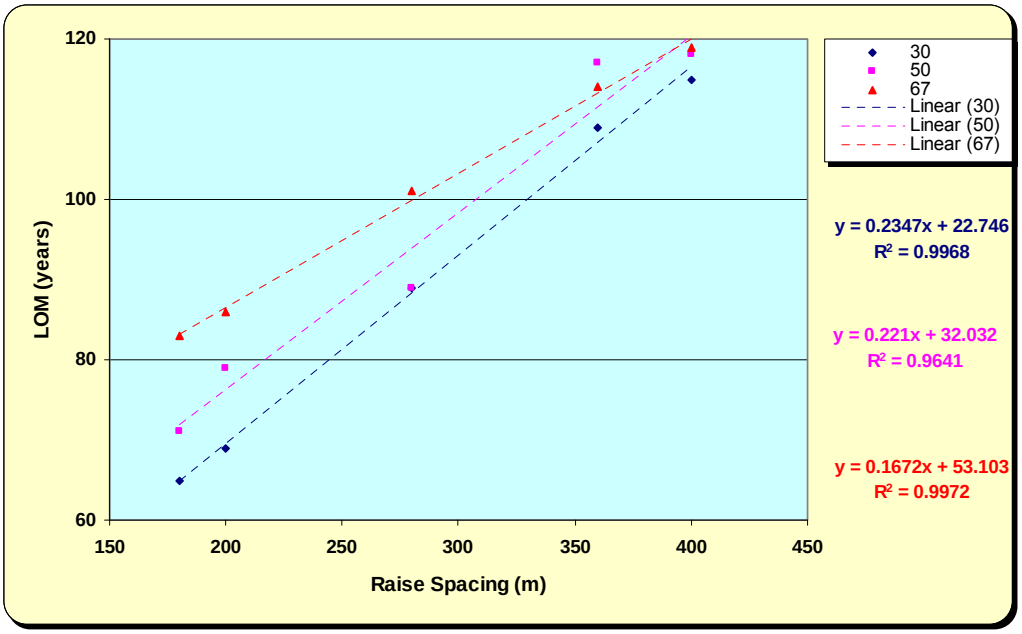


Figure 6.25: Variation of LOM with increasing raise spacing

6.4.11 Build-up period

The build-up period displayed a general logarithmic trend with level and raise spacing as indicated in Figure 6.26 and Figure 6.27.

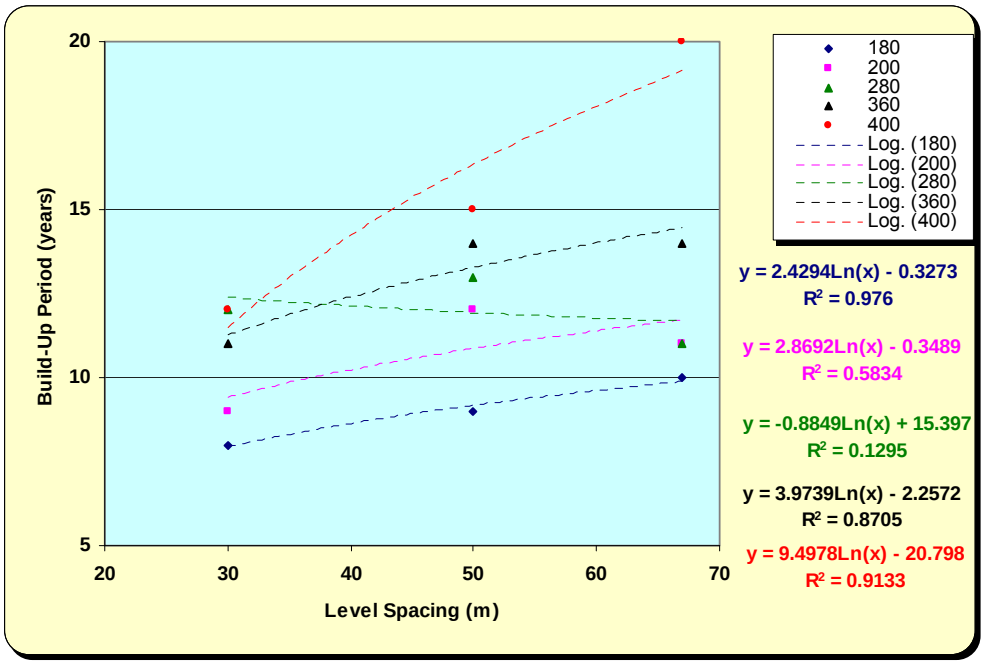


Figure 6.26: Variation of build-up period with level spacing

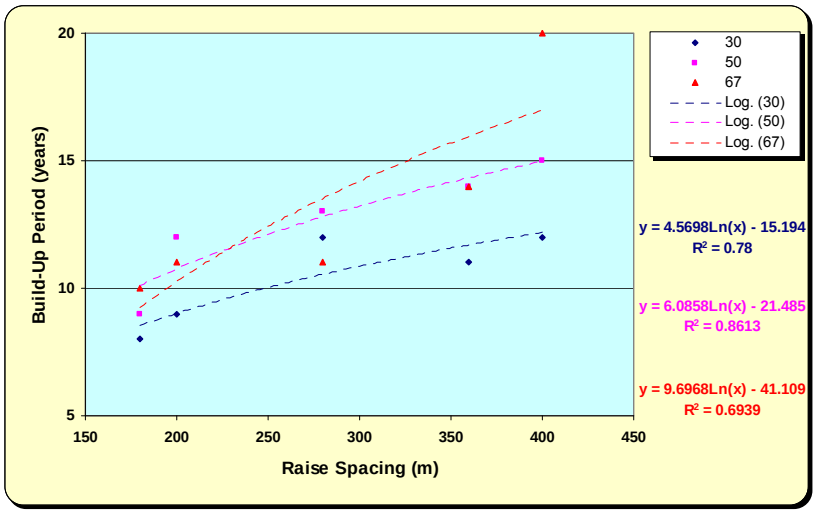


Figure 6.27: Variation of build-up period with increasing raise spacing

6.5 Interpolation and cross-interpolation of optimisation criteria results

In order to analyse the behaviour of the optimisation criteria with respect to level and raise spacing in a 3-D space, the Microsoft Excel® 3-D surface contour functionality was used. However, this tool requires that the level and raise spacing axes be separately scaled in equal intervals. In order to do this an assumption was then made that a vertical level spacing of 67m could, for practical purposes, be approximated to 70m vertical level spacing implying that the optimisation criteria results for 67m vertical level spacing would be assumed to be for 70m vertical level spacing. Therefore, the range of vertical level spacing analysed was between 30m-70m taken in increments of 10m. The range of raise spacing analysed was between 180m-400m taken in increments of 20m, for the reason stated in Section 6.2.3. The optimisation criteria were therefore interpolated and cross-interpolated over these specified ranges of spacing, using the relationships established in Section 6.4. These calculations are contained in the Microsoft Excel® file *Final_Criteria_Summary* contained on the USB memory stick, so only the interpolation and cross-interpolation for PV of development cost per centare is presented here to illustrate how the process was done. Table 6.9 shows a summary of the results.

Table 6.9: PV of development cost per centare mined (ZAR mil)

		Raise spacing (m)											
		180	200	220	240	260	280	300	320	340	360	380	400
Level spacing (m)	30	91.19	88.93	87.88	86.58	85.39	84.50	83.32	82.40	81.55	80.77	80.01	79.21
	40	82.95	80.35	79.60	78.40	77.31	76.33	75.40	74.56	73.77	73.10	72.35	71.72
	50	77.40	73.85	73.54	72.34	71.26	69.98	69.37	68.53	67.75	67.12	66.34	65.88
	60	72.37	69.95	69.40	68.38	67.45	66.50	65.82	65.10	64.43	63.87	63.22	62.72
	70	69.47	67.72	67.02	66.10	65.26	64.49	63.79	63.13	62.53	62.01	61.43	60.91

In Table 6.9 the PV values in black were obtained from the 15 layouts. The PV values in red were interpolated from the relationships established in Section 6.4. The PV values in blue were cross-interpolated from the values in red through establishing new relationships for 40m and 60m vertical level spacing based on the values in red as shown in figure. Cross-interpolation could be done because this study was done at a pre-feasibility level of accuracy in estimates. Ideally, layouts should have been designed and scheduled for all the red and blue entries of PV values, but since it took about 8 weeks taken to completely design and schedule each layout; this would have meant spending close on to 480 weeks (i.e. about 9 years) to complete the study.

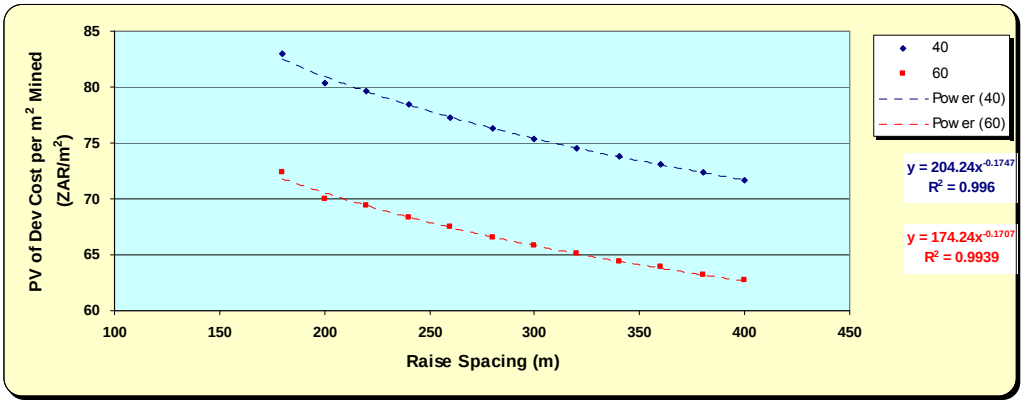


Figure 6.28: Cross-interpolation of PV values to estiamte PV values for layouts that were not designed

6.6 Summary

As was postulated in Section 2.2.6, this chapter has demonstrated that level and raise optimisation criteria have logarithmic, quadratic, exponential, uniform, linear or power function relationships with increasing level and raise spacing. This chapter has analysed the behaviour of these optimisation criteria with respect to changes in level and raise spacing and confirmed some of the earlier findings by Lawrence (1984). If NPV was the sole optimisation criterion then for OB1 the optimal range level spacing should be between 40m-50m and optimal range of raise spacing should be between 200m-250m. However, since it has already been shown that the optimisation of level and raise spacing is a multi-criteria optimisation problem, the optimisation criteria identified in this study were subsequently integrated in an optimisation process that is discussed in the next chapter to derive a more holistic optimal range of level and raise spacing for conventional breast mining layouts.

7 OPTIMISATION BY ANALYTIC HIERARCHY PROCESS (AHP)

7.1 Introduction

As discussed in Chapter 2, optimisation of level and raise spacing must be solved using the MCDA approach. The MCDA approach selected for this research is the AHP methodology and reasons for its choice were discussed in Chapter 2. The present chapter initially discusses a survey undertaken to establish the weighting attached to each of the decision criteria used in selecting the optimal inter-level and raise spacing. The survey results are then analysed in Microsoft Excel® to normalise them and estimate the consistency of the decision makers that were surveyed. The weights are then applied to the scores that each layout obtained for each optimisation criterion Chapter 6, to establish the aggregate AHP priority score for each layout. The aggregate AHP priority scores were then used to rank the layouts based on their performance against the decision criteria. Lastly sensitivity analyses were done to check the effect of inconsistency in the decision makers' judgement on the stability of the optimal level and raise spacing established. Sample size is not important in AHP because it is not a statistical survey.

7.2 Survey of relative weighting of decision criteria

The first step in the AHP was to determine the cardinal weights attached to each of the optimisation criteria. A structured questionnaire survey was undertaken to solicit technical perceptions from mine planning and project planning practitioners and experts in the local South African platinum industry. A sample questionnaire and responses are included in Appendix 10.2. The responses from the survey were then used to estimate the cardinal weights for each of the criteria.

7.3 MS Excel procedure for the AHP

The matrix framework for the AHP described in Chapter 2 can be solved using Microsoft Excel®. Searcy (2004) described a step-by-step process for using Microsoft Excel® to solve an AHP decision problem. Searcy (2004) identified 5 distinct steps but for this study 6 steps were identified since the weights obtained from the survey were group responses and needed to be firstly aggregated before being normalised and that the criteria were a mixture of maximisation and minimisation criteria. Inverses of efficiency scores of layout alternatives against minimisation criteria had to be calculated as proposed by Peters and Zelewski (2008) before the final normalisation process. These steps are discussed below. These matrices resulting from the steps are contained in the Microsoft Excel® file *AHP_Survey_Analysis* included on the USB memory stick.

7.3.1 Enter pair-wise weighting of criteria from each respondent

The responses from each individual company that participated in the survey were entered into a matrix format by applying the reciprocity axiom.

7.3.2 Aggregate the pair-wise responses

In this step the individual responses matrices were aggregated into a single composite matrix, the industry matrix, by taking the geometric mean of all corresponding cell entries across all the individual matrices. The rationale for using the geometric mean is explained in Chapter 2.

7.3.3 Normalise the pair-wise comparisons

This step has two sub-steps. Firstly, the sum of each column is calculated. Secondly, each entry in the matrix is divided by its column sum. The resulting matrix is a normalised pair-wise weight matrix.

7.3.4 Calculate the average aggregate weight of each criterion

The average of each row in the normalised matrix is used as the aggregate weight for each criterion (or its relative importance weight). From the Microsoft Excel® file *AHP_Survey_Analysis* it indicates that NPV is the most important criterion with a relative weight of 0.13 followed by grade with a relative weight of 0.11, and so on until we get to the least important criterion replacement ratio rated at 0.04.

7.3.5 Estimate the consistency of judgements

As mentioned in Chapter 2, a perfectly consistent judgement has CI = number of criteria under comparison. In the survey, 12 criteria were considered and therefore $\lambda_{max} \geq 12$. A check on the CI column show that all the CI values were greater than 12, therefore the calculations were correct. The RI used was for a matrix size of 12 for which the RI = 1.48 (Table 2.2). The CR were then evaluated and all were below 0.1 (Anglo MTS CR = 0.04; Anglo LTSP CR = 0.02; Impala Platinum Mining Projects CR = 0.10; Industry Aggregate Average = 0.01). The judgements can be considered to be reliable to proceed with the AHP since the all inconsistencies are below the 10% threshold.

7.3.6 Apply weights to layout scores to obtain AHP priority scores

The weights were finally aggregated together with the layout efficiency scores against each criterion to get the overall AHP priority score as described in Chapter 2.

7.4 Optimal range of level and raise spacing

The AHP priority scores were plotted onto 3-D contour space and the results are of this exercise are illustrated by Figure 7.1.

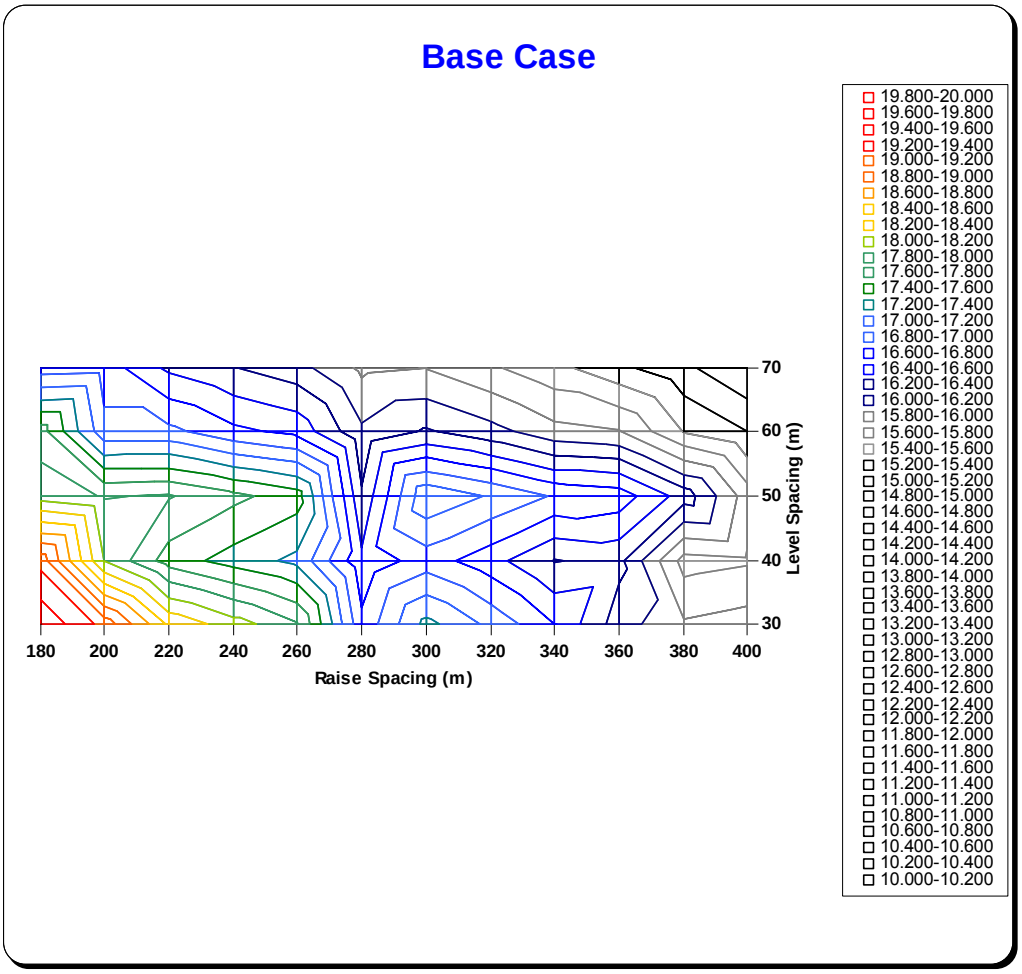


Figure 7.1: A 3-D Excel surface contour plot of the AHP priority score for the base case

The highest AHP priority scores occur in the bottom left corner of Figure 7.1 indicating that for OB1, the optimal range of vertical level spacing is between 30m-50m while the optimal range of raise spacing is between 180m-220m. Figure 7.1 also indicates that two layouts are

associated with local maxima. These layouts are for 30m vertical level spacing (≈180m backlength at 9.6º dip) by 180m raise spacing and 50m vertical level spacing (≈300m backlength at 9.6º dip) by 300m raise spacing. This observation means that local optima are obtained for layouts in which the average backlength is nearly equal to the raise spacing, that is, the stope shape is almost a square shape. A similar finding was established using the Lagrange multiplier in Chapter 5. However, the local optimum at the wider spacing is lower than the local optimum at the smaller spacing. This finding implies that conventional breast mining layouts should be planned at lower spacing but with stope shapes that have nearly square configurations defined by backlength almost equal to raise spacing.

7.5 Sensitivity analysis

The human brain can easily configure an optimal decision such as deriving maximum benefit or minimum loss when faced with a 2-dimensional problem expressed as a quadratic function in an *x*-*y* Cartesian plane, or when the decision problem is 3-dimesnional expressed as a surface in 3-D *x*-*y*-*z* space. When optimisation decisions involve decision criteria that exceed 3-dimenions, humans have to rely on abstract thinking or attempt to simplify the problem back to 2-D or 3-D for easier configuration. Saaty and Ozdemir (2003), Yavuz and Pillay (2007a), Yavuz and Pillay (2007b) and Yavuz (2007), argued that the AHP produces reliable results when the number of criteria do not exceed the magic number seven plus two (=9) because of the general limitations on human performance on abstract thinking. Therefore, when faced with criteria that exceed 9, it is advisable to cluster criteria and perform the AHP analysis on clustered criteria. Where this is not possible, a sensitivity analysis must be done to check the stability of the solution obtained. In this survey a total of 12 criteria were identified and these exceed the recommended maximum number of 9. It was not possible to cluster the criteria, therefore sensitivity analyses were done to establish the stability of the optimal solution derived. The sensitivity analyses are also necessary for two other reasons. Firstly, the process of assigning weights of importance to optimisation or decision criteria is partly subjective depending on an individual's knowledge and experience or a company's policies and experiences. Secondly, as discussed earlier in Chapter 2, human judgements tend have some degree of inconsistency, which the AHP methodology is able measure, as was done in this study. This implies that sensitivity analyses will aid in validating the solution against the inconsistencies inherent in the decision-making and optimisation process.

Sensitivity analyses were done for the following cases:

- ❖ All criteria have equal weighting of 1, implying a case of indifference to the importance of each criterion.

- ❖ The importance of NPV should have been 10% more than what respondents thought it was.
- ❖ The importance of NPV should have been 10% less than what respondents thought it should have been.

The results of the sensitivity analysis confirm the same trend towards smaller spacing and square stope configurations, although the range of spacing changes slightly in each case as illustrated by Figure 7.2, Figure 7.3 and Figure 7.4.

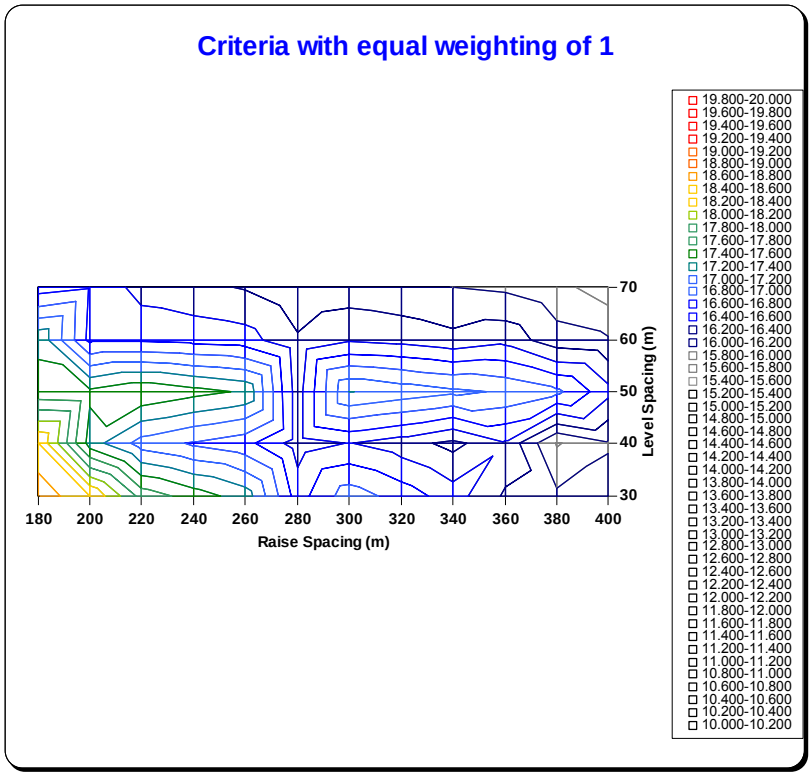


Figure 7.2: A 3-D Excel surface plot of the AHP priority score when criteria carry equal weighting

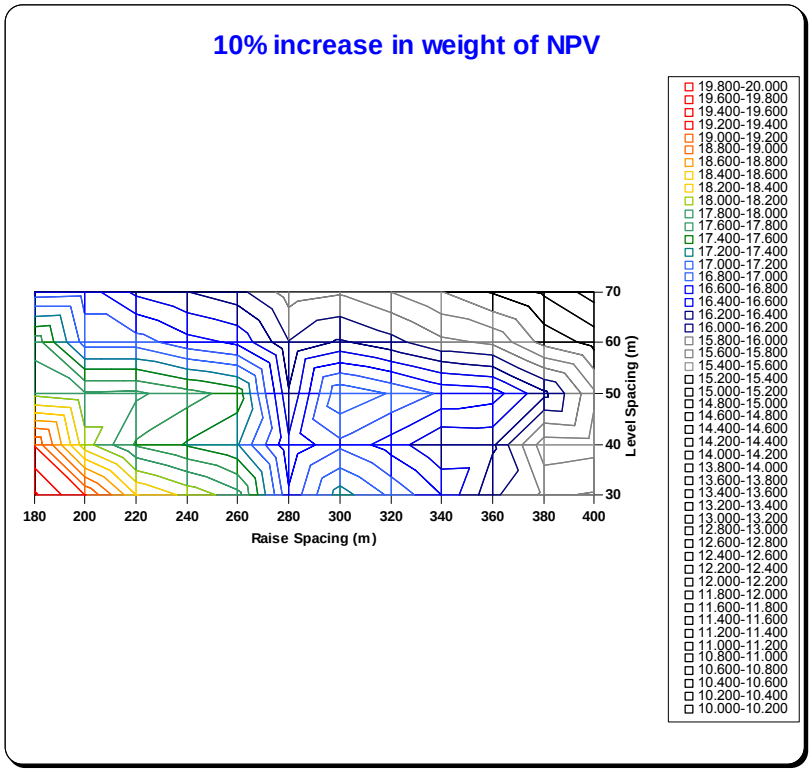


Figure 7.3: A 3-D Excel surface plot of the AHP priority score when importance of NPV increases by 10%

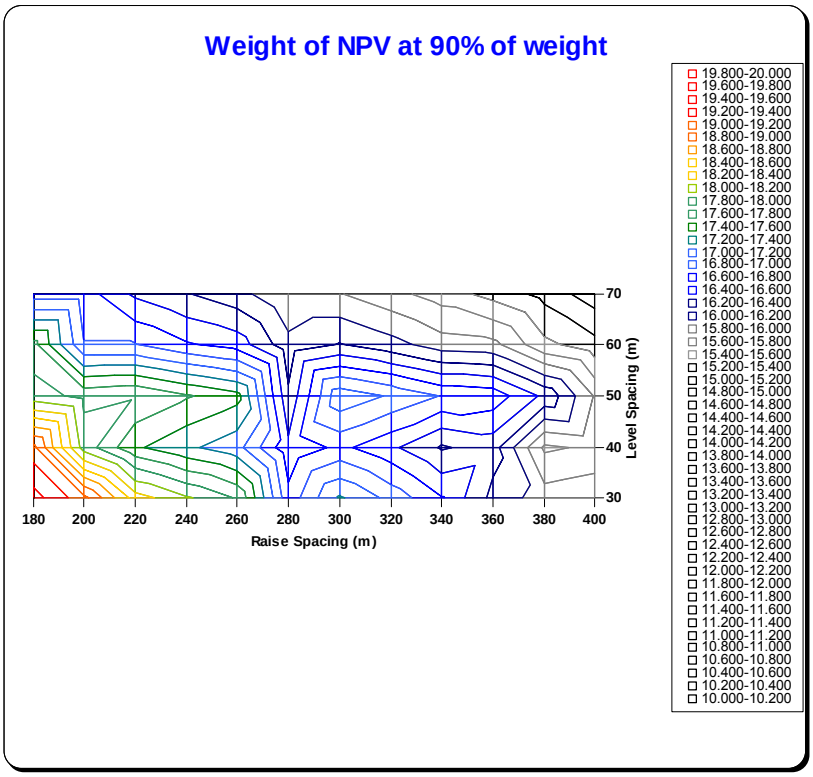


Figure 7.4: A 3-D Excel surface plot of the AHP priority score when importance of NPV decreases by 10%

7.6 Summary

Previous studies on multi-criteria selection have used the AHP as a selection tool. This study used the AHP as aid to optimisation under multi-criteria conditions. The AHP identified that for OB1 the optimal range for vertical level spacing was 30m-50m and the optimal range for raise spacing was 180m-220m. Sensitivity analysis performed indicated that the optimal range is a fairly stable solution and can therefore be accepted with confidence. However, this finding is in contrast to traditionally held perceptions that optimisation is achieved by increasing level and raise spacing, instead the industry should be considering reducing the spacing of level and raises. In addition, local maxima were identified for layout geometries that are approximately square in shape, whereby the backlength is equal to raise spacing. This finding suggests that mines should seriously consider revising their layout geometries from rectangular-shaped stopes to square-shaped stopes. The next chapter is the final chapter which discusses the observations, conclusions and recommendations arsing from the findings made in this chapter in relation to the rest of the thesis.

8 OBSERVATIONS, CONCLUSIONS AND RECOMMENDATIONS

8.1 Introduction

Optimising level and raise spacing in inclined narrow reef mining has been a subject of controversy for decades. This is noted in one of the feedback comments from industry on this research study in that, *"level spacing and raise line spacing has been a controversial topic in the mining industry for decades. No two mining engineers will agree on this issue as there has been no way to scientifically calculate the best option. The only way available to mining engineers previously has been to laboriously model these variables manually, with no conclusive decisions"* (Impala Platinum Review Team, 2009). This chapter notes the observations made, conclusions drawn and recommendations made for further work from the research study.

8.2 Observations

A few key observations from the research study include the following:

- Optimisation of level and raise spacing in planning conventional mining layouts is NOT a mono-criterion optimisation problem based solely on minimising development cost per centare mined (ZAR/m^2), but is a multi-criteria optimisation problem. Solving the optimisation problem as a mono-criterion problem leads to a divergent solution where the cost per centare just keeps on getting smaller and smaller.
- The development cost per centare mined in ZAR/m^2 or the cost per tonne mined in ZAR/t decreases asymptotically, following a power function, with increasing level and raise spacing. The underlying driver for a power relationship derives from the analogy of repeatedly dividing a line into equal parts.
- Other optimisation criteria have logarithmic, linear, or quadratic relationships with increasing level and raise spacing. In particular NPV has a quadratic relationship with increasing raise spacing. If NPV were the sole optimisation criterion then as seen from Chapter 6, a raise spacing of around 240m-250m would be optimal. This finding is consistent with Lawrence's (1984) optimal economic raise spacing in scattered mining layouts.
- Operating flexibility is one of the key criteria that must be considered in planning level and raise spacing. It is possible to measure the operating flexibility through a Flexibility Index (FI) that was derived in this research study.
- Wider spacing of raises and levels has the economic attraction of lower ZAR/m^2 mined. However, this attraction is heavily off-set by reduced productivity, operating flexibility and other factors which fall rapidly at wider raise spacing.

- ❖ The change in the unit value of a PGM resource when mined with conventional breast mining is insignificant when raise spacing increments are made in increments that are less than 20m. This implies that when evaluating conventional mining layouts, the minimum raise spacing changes that should be made should be at least 20m.
- ❖ Although the AHP has traditionally been used as a selection tool in decision making, this study has shown that is can be used as an aid to optimisation.

8.3 Research contributions

Once level and raise spacing are selected for a conventional breast mining layout, it becomes the basis upon which future medium to long term mine plans are developed. Therefore, level and raise spacing has a strategic impacts and should be treated as part of the strategic mine planning process. This viewpoint is supported from the feedback comments from industry because, *"The writer agrees with the research statement that level and raise spacing form the basis of the life of mine planning, hence has significant impact on tactical and strategic issues"* (Impala Platinum Review Team, 2009).

This research study has demonstrated that the optimisation of level and raise spacing in conventional breast mining layouts is not a mono-criterion optimisation problem based on minimising excavation and haulage costs only, but a multi-criteria optimisation problem. Technical operating flexibility is one of the criteria that must be considered in the optimisation process and a methodology was developed in this research study to measure this criterion. A multi-criteria decision analysis (MCDA) methodology called the Analytic Hierarchy Process (AHP) was adapted and successfully used to solve the problem as a multi-criteria optimisation problem.

The results show that for decades, the narrow tabular reef mining industry has been looking in the wrong direction of advocating longer backlengths (i.e. wider level spacing) and wider raise spacing instead of using smaller spacing that affords concentrated mining and higher productivities. One of the feedback comments from industry supporting this viewpoint noted that, *"when conventional mining started in narrow reef mining, it started as 'concentrated breast mining' but as an industry we have over the years lost the plot by changing it to 'scattered' mining which does not afford us high productivities hence these findings make sense that we should be moving back towards smaller level and raise spacing"* (Rogers, 2009). The drive for concentrated mining has previously been highlighted by Brassell (1964), Bullock (2001), and Vieira, Diering and Durrheim (2001). However, care must be taken that concentrated mining achieved through smaller level and raise spacing, is not a panacea for higher productivities because as noted by Brassell (1964:461), the concentration of mining

activities to improve productivity *"is no gimmick that can be introduced overnight with the introduction of new machines and techniques, but rather is the outcome of study, careful planning and the training of personnel, all of which takes much time and money to achieve."* Section 3.2 also noted that adequate face availability on its own did not guarantee operating flexibility unless it is created under concentrated mining conditions.

Another contribution coming from this research study is that mines should seriously consider nearly square geometries for stope outlines because these are associated with overall local maxima as demonstrated by the Lagrange multiplier method in Chapter 5 and the 3-D contour plots of the AHP priority scores in Chapter 7.

Another contribution by this research study is that a precise optimal level and raise spacing is rather an exaggerated level of accuracy since there is a degree of subjectivity by practitioners in assigning the weights of the importance of each criteria considered in deriving optimal spacing, hence an range of optimal spacing was derived. Therefore, the derived range of optimal level and raise spacing should be sued as a guide only but the methodology can be used for any orebody to derive a range that is specific to the orebody. This recommendation concurs with observations noted in Section 2.2.4 of Chapter 2.

Lastly, the optimal vertical level spacing range of 30m-50m and optimal raise spacing range of 180m-220m derived OB1, which is a typical UG2 reef deposit, may be extended to cover the rest of the Bushveld Complex since OB1 represents typical UG2 reef mining conditions. However, since each deposit has geological and geo-technical conditions that are specific to the mine, a more realistic range can be derived using mine specific data with the methodology developed in this research study. This is because the methodology developed in this research study *"takes cognisance of the uniqueness of ore bodies by not providing a 'one size fits all' solution"* (Impala Platinum Review Committee, 2009).

8.4 Research limitations

This research assumed that the scores of each layout against each optimisation criterion were deterministic yet in reality the operational performance results are stochastic. For example the assumed average panel face advance rate of 15m/month that was used as an input to the scheduling process in Chapter 5 is a deterministic figure that comes from some specific statistical distribution in the typical of range 10m-17m achieved by industry. The reason for using a deterministic approach is that the available current mine design and scheduling software does not cater for stochastic input parameters. When stochastic mine design and scheduling software becomes available, this exercise could be repeated with input data drawn from the relevant statistical distributions.

This research assumed that mining was on one reef horizon, the UG2. However, for most mines on the Bushveld Complex, mining actually occurs on both the Merensky and UG2 reefs. Singh *et al* (2005) have previously modelled the rock engineering aspects of mining the UG2 concomitantly with the Merensky. Mining of UG2 in this case might be equivalent to mining in deep to ultra-deep environments and there is the opportunity to use existing infrastructure utilised to mine the Merensky (Singh *et al*, 2005). Therefore an opportunity exists for this exercise to be repeated under conditions when both reef horizons are being mined.

Lastly, this research assumed that some of the mining sub-systems such as the tramming and hoisting systems are already optimised. Therefore, separate studies such as simulation studies, can be done to ensure that such sub-systems are optimised to match the planned production rates. For example the work done by Bye (2003) for the PPRust open pit mine on optimising the blasting and fragmentation process from mine to mill can be done as a separate study for the underground narrow reef platinum mines to ensure that such sub-systems are optimised.

8.5 Recommendations for future research work

Following on from discussions arising from the presentations made to industry on the research findings, the issues listed below constitute possible future areas of research:

- It was expedient to focus the research on tabular, regular deposits because conclusions from this case can then be used as a basis for investigating level and raise spacing for irregular deposits such as the hypothetical mine discussed by Eaton (1934). Thus, a similar exercise can be done for deposits that are neither regular nor tabular.
- In the case of a new mine investigations can be done to evaluate planning the first few levels at short backlengths and narrow raise spacing for a rapid production build-up and thereafter plan the rest of the mine on longer backlengths once the mine has ramped up to full production. The evaluation should note that longer backlengths at narrow raise spacing have the potential to increase the half-level output but will require more expensive haulages to meet higher ventilation and tramming requirements.
- The production profile of some UG2 projects tend to have long tails which are unfavourable economically and there is need to investigate the underlying drivers of long tails and how the profile can be optimised to reduce the tailing-off period. One

of the ways noted in this study is to ensure that each level has a replacement level to enable smoothening out production and shorten the production tail.

- A study can be undertaken to check if there is a correlation between scraper-related incidences and accidents versus the scraping distances on operations to confirm whether an increase in raise spacing has the potential to increase scraper-related accidents due to a 'stretched-out' line of sight.
- The methodology could also be used with modifications to compare combinations of vertical shafts and declines for accessing deeper reef extensions of the Bushveld Complex.
- Optimisation criteria exhibited power function, linear, quadratic and logarithmic and exponential relationships with increasing level and raise spacing. Although providing explanations for these underlying relationships was not part of the central theme of this research, it warrants further investigation because the results of such an investigation would provide further insights into the problem of optimising level and raise spacing.

9 REFERENCES

Ackerman, G R and Jameson, A W. (2001), Chapter 16, Underground mining at Impala Platinum, Ltd., in the Northwest Province of South Africa, in *Underground Mining Methods: Engineering Fundamentals and International Case Studies*, Hustrulid, A and Bullock, R L. (eds), Society for Mining, Metallurgy, and Exploration (SME), Colorado, USA, pp157-163.

Ataei, M. (2005), Multicriteria selection for an alumina-cement plant location in East Azerbaijan province of Iran, in *The Journal of The South African Institute of Mining and Metallurgy*, Vol. 105, No. 8, August 2005, pp507-513.

Alford, C. (1995), Optimisation in underground mine design, in *Proceedings APCOM XXV*, Brisbane, Australia, 9th -14th July 1995, pp213-218.

Almeida, de A T, Alencar, L H and Miranda, de C M G. (2005), Mining methods selection based on multicriteria models, in *Proceedings Application of Computers and Operations Research in the Mineral Industry (APCOM 2005)*, Dessureault, Ganguli, Kecojevic and Dwyer (eds), Tucson, USA, 30 March-1 April, 2005, pp19-24.

Anglo Platinum, Boschfontein Shaft, Geology Department (2004). Geology electronic file on Company Internal Geological Database.

Anglo Platinum Mine Technical Services, MTS (2005), *Group Guideline Mine Technical Services: Conventional Breast Layout Design Guideline, Version 1.0, October 2005*, Company Internal Document.

Anglo Platinum Projects Division (2007), Unki Mines Feasibility Study, Company Internal Report, 25th July 2007.

Anglo Platinum Review Committee (2009), Critique of PhD research presentation by Cuthbert Musingwini to Anglo Platinum on Monday 13th July at 14:00 in Room 1406, 55 Marshall Street, a feedback report included as part of the appendix of this thesis.

Ballington, I, Smith, G L, Lane, G R and Hudson, J. (2005), The systemic interdependency of closure decisions at shaft level, in *Proceedings of the First International Seminar on Strategic versus Tactical Approaches in Mining*, Sandton, 19-21 September 2005, pp153-164.

Beath, C B, Cousins, C A and Westwood, R J. (1961), The exploitation of the platiniferous ores of the Bushveld Igneous Complex with particular reference to the Rustenburg Platinum Mines, in *Transactions of the Seventh Commonwealth Mining and Metallurgical Congress: Papers and Discussions Volume 1*, South African Institute of Mining and Metallurgy, Johannesburg, pp217-243.

Belton, V and Stewart, T J. (2002), *Multiple Criteria Decision Analysis: An Integrated Approach*, Kluwer, Dordrecht.

Bitarafan, M R and Ataei, M. (2004), Mining method selection by multiple criteria decision making tools, in *The Journal of The South African Institute of Mining and Metallurgy*, Vol. 104, No. 9, October 2004, pp493-498.

Brassell, T J. (1964), Stoping improvements, in *Papers and Discussions 1962-1963*, Vol 1, Association of Mine Managers of South Africa, Transvaal and Orange Free State Chamber of Mines, Johannesburg, pp460-495.

Brazil, M, Lee, D, Leuven, van M, Rubinstein, J H, Thomas, D A and Wormald, N C. (2003), Optimising declines in underground mines, in *Transactions of the Institution of Mining and Metallurgy: Mining Technology*, Vol. 112, ppA164-A170.

Brazil, M, Lee, D, Rubinstein, J H, Thomas, D A, Weng, J F and Wormald, N C. (2004), Optimisation in the design of underground mine access, in *Proceedings of the International Symposium on Orebody Modelling and Strategic Mine Planning: Uncertainty and Risk Management*, Perth, Australia, 22nd – 24th November 2004, pp3-6.

Brazil, M, Thomas, D A, Weng, J F, Rubinstein, J H and Lee, D H. (2005), Cost optimisation for underground mining networks, in *Optimization and Engineering*, Vol 6, pp241-256.

Buchanan, D L. (1987), Platinum-group element exploration: targets and guidelines, in *Transactions of the Institution of Mining and Metallurgy, Section A: Mining Industry*, Vol 96, July 1987, pp A108-A115.

Bullock, R L. (2001), Chapter 2: General planning of the non-coal underground mine, in *Underground Mining Methods: Engineering Fundamentals and International Case Studies*, Hustrulid, A. and Bullock, R L. (eds), Society for Mining, Metallurgy, and Exploration (SME), Colorado, USA, pp15-28.

Bye, A. (2003), The development and application of a 3D geotechnical model for mining optimisation: Sandsloot open pit platinum mine South Africa, *PhD Thesis*, University of Natal, Durban.

Carter, P R. (1999), Mining practice for intermediate depth narrow tabular ore bodies with particular reference to the Free State and Klerksdorp goldfields, in *Proceedings School: Narrow Tabular Orebody Mining*, 21st – 22nd October 1999, The South African Institute of Mining and Metallurgy, Randburg.

Carter, P G, Lee, D H and Baarsma, H. (2004), Optimisation methods for the selection of an underground mining method, in *Proceedings of the International Symposium on Orebody Modelling and Strategic Mine Planning: Uncertainty and Risk Management*, Perth, Australia, 22nd – 24th November 2004, pp7-12.

Cawthorn, R G. (1999), The platinum and palladium resources of the Bushveld Complex, in *South African Journal of Science*, Vol. 95, November/December 1999, pp481-489.

Cawthorn, R G. (2006), The centenary of the discovery of platinum in the Bushveld Complex (10 November, 1906), in *Proceedings of the Second International Platinum*

Conference: 'Platinum Surges Ahead', Sun City, South Africa, 8th – 12th October, 2006, The Southern African Institute of Mining and Metallurgy, pp1-3.

Cawthorn, R G. (2007), The centenary of the discovery of platinum in the Bushveld Complex (10 November, 1906), in *The Journal of the Southern African Institute of Mining and Metallurgy*, Vol. 107, No.1, pp1-3.

Chamber of Mines (2005), *Facts and Figures 2005*, an annual publication of the Chamber of Mines of South Africa, available from INTERNET http://www.bullion.org.za. Accessed 28th May 2008.

Chamber of Mines (2006), *Facts and Figures 2006*, an annual publication of the Chamber of Mines of South Africa, available from INTERNET http://www.bullion.org.za. Accessed 18th June 2009.

Chamber of Mines (2007), *Facts and Figures 2007*, an annual publication of the Chamber of Mines of South Africa, available from INTERNET http://www.bullion.org.za. Accessed 18th June 2009.

Chen, Y. (2006), Multiple Criteria Decision Analysis: Classification Problems and Solutions, *PhD Thesis*, University of Waterloo, Canada. Available on INTERNET http://etd.uwaterloo.ca/etd/y3chen2006.pdf. Accessed 4th February 2008.

Chunnett, G. (2006), Anglo Platinum: Taking Advantage of the Current Metal Prices and Shortage of Platinum, in *Proceedings of the Second International Platinum Conference 'Platinum Surges Ahead'*, Sun City, South Africa, 8th – 12th October, 2006, The Southern African Institute of Mining and Metallurgy, pp5-10.

Coetzee, V. (2006), Common-sense improvements to electric smelting at Impala Platinum, in *South African Pyrometallurgy Conference 2006*, The South African Institute of Mining and Metallurgy, Johannesburg, 5th – 8th March 2006, pp43-62.

Collender, F D. (1987), Platinum – one of the world's most strategic metals, in *Transactions of the Institution of Mining and Metallurgy, Section A: Mining Industry*, Vol 96, July 1987, pp A104-A108.

Costa, M. and Vuuren, J J. (2004), A win-win opportunity in stope support at Modikwa Platinum mine, in *Proceedings of Colloquium: Mechanised Mining and Support in the Bushveld Complex*, May 2004, Rustenburg.

Costa, J. (2007), Calculating Geometric Means, Buzzards Bay National Estuary Program, Last updated 24th September 2007, available on INTERNET http://www.buzzardsbay.org/geomean.htm. Accessed 19th February 2008.

Coyle, G. (2004), The Analytic Hierarchy Process (AHP), in Coyle, G, *Practical Strategy: Structured Tools and Techniques*, Open Access Material, Glasgow: Pearson Education Ltd, 2004, ISBN 0-273-68220-2. Available on INTERNET http://www.booksites.net/download/coyle/student_files/AHP_Technique.pdf. Accessed 7th January 2008.

Cox, A M, Alwang, J and Johnson, T G. (2000), Local preferences for economic development outcomes: Analytical Hierarchy Procedure, in *Growth and Change*, Vol. 31 Issue 3, pp341-366.

Cruise, J. (2005), Dr. John Cruise of John Cruise Mining Consultants, *Personal communication*, Johannesburg.

Dapena, J P and Fidalgo, S. (2003), A real options approach to tender offers and acquisitions processes, available on INTERNET http://www.realoptions.org/papers2003/DapenaDoctrabapproach.pdf. Accessed 2nd November 2005.

Darwish, M A and Butt, N A. (1989), Forecasting in the mineral industry, in *International Mining*, December 1989, pp12-17.

Davis, G A. (1998), One project, two discount rates, in *Mining Engineering*, April 1998, pp70-74.

Deeplaul, V. and Bryson, M. (2004), Mintek, a national resource of minerals processing expertise for platinum ores, in *Proceedings of the First International Platinum Conference: 'Platinum Adding Value'*, The South African Institute of Mining and Metallurgy, pp9-14.

Deloitte Touche Tohmatsu (2009), International Tax: South Africa Highlights, available on INTERNET http://www.deloitte.com/dtt/cda/doc/content/dtt_tax_highlight_2009_southafrica.pdf. Accessed 18th June 2009.

Department of Minerals and Energy (2001), *Minerals Act and Regulations No. 50 of 1991*, 1st Edition 2001, Lex Patria, Durban.

Department of Minerals and Energy, DME (2004a), Guideline document for the evaluation of the quantum of closure-related financial provision provided by a mine, Section A: background details and supporting information, Rev 1.5, 23 April 2004. A DME document available on INTERNET http://www.dme.gov.za/pdfs/minerals/fin_prov_guide_a.pdf. Accessed 28th February 2008.

Department of Minerals and Energy, DME (2004b), Section B: Working manual for the assessment of the quantum, A DME document available on INTERNET http://www.dme.gov.za/pdfs/minerals/fin_prov_guide_b.pdf. Accessed 28th February 2008.

Department of Minerals and Energy, DME (2005), TABLE 48: LABOUR STATISTICS – ALL MINES, A DME document available on INTERNET http://www.dme.gov.za/publications/pdf/stats/minerals/tables/tbl48.pdf. Accessed 20th July 2008.

Department of Minerals and Energy, DME (2006a), An overview of current platinum-group metal exploration projects and new mine developments in South Africa, Report

R51/2006, A DME report published in January 2006 and available on INTERNET http://www.dme.gov.za/pdfs/minerals/R51%202006.pdf. Accessed 28th February 2008.

Department of Minerals and Energy, DME (2006b), *South Africa's Mineral Industry (SAMI) 2005/2006*, Department of Minerals and Energy, Pretoria, ISBN 1-919927-35-2.

Department of Minerals and Energy, DME (2007), *South Africa's Mineral Industry (SAMI) 2005/2006*, Department of Minerals and Energy, Pretoria, ISBN 978-1-919927-72-5.

Dessureault, S and Scoble, M J. (2000), Capital investment appraisal for the integration of new technology into mining systems, in *Transactions of the Institution of Mining and Metallurgy (Section A: Mining Technology)*, Vol. 109, January-April 2000, ppA30-A40.

Diering D (1997), Ultra-deep level mining – future requirements, in *Proceedings Colloquium on Technology for Rapidly Accessing the Ore Body*, 30 July 1997, The South African Institute of Mining and Metallurgy: Randburg.

Doumpos, M and Zopounidis, C. (2002), *Multicriteria Decision Aid Classification Methods*, Kluwer, Dordrecht.

Eaton, L. (1934), *Practical Mine Development and Equipment*, 1st Edition, McGraw-Hill Book Compnay, Inc., New York, 405p.

Egerton, F M G. (2004), Presidential address: The mechanisation of UG2 mining in the Bushveld Complex, in *The Journal of The South African Institute of Mining and Metallurgy*, Vol. 104 No. 8, September 2004, pp439-455.

Elevli, B and Demirci, A. (2004), Multicriteria choice of ore transport system for an underground mine: application of PROMETHEE methods, in *The Journal of The South African Institute of Mining and Metallurgy*, June 2004, pp251-256.

Elevli, B, Demirci, A and Dayi, O. (2002), Underground haulage selection: shaft or ramp for a small-scale underground mine, in *The Journal of The South African Institute of Mining and Metallurgy*, Vol. 102 No. 5, September 2004, pp255-260.

Fleming, D. (2002), Survey of development parameters for some Witwatersrand gold mines" *Internal Report constituting part of the MINN358 Course*, School of Mining Engineering, University of Witwatersrand, Johannesburg.

Geldermann, J. and Rentz, O. (2005), Multi-criteria analysis for technique assessment: Case study from industrial coating, in *Journal of Industrial Ecology* 9(3):127-142. Available on INTERNET http://mitpress.mit.edu/journals/jiec/supplements/859jg3esupp.pdf. Accessed 19th February 2008.

Gordon, T J. (1994), The Delphi Method, available on INTERNET http://www.futurovenezuela.org/_curso/5-delphi.pdf. Accessed 15th February 2008.

Hajdasiński, N M. (1995), Time value of money and optimal location of mining facilities, in *Proceedings APCOM XXV Conference*, Brisbane, 9th – 14th July 1995, pp231 – 239.

Harker, P T and Vargas, L G. (1987), The theory of ratio scale estimation: Saaty's Analytic Hierarchy Process, in *Management Science*, Vol. 33, No 11, pp1383-1403.

INTERNET http://links.jstor.org/sici?sici=0025-1909%28198711%2933%3A11%3C1383%3ATTORSE%3E2.0.CO%3B2-W. Accessed 16th October 2007.

Hatch Associates (2004), Consultant takes up underground challenge, INTERNET http://www.miningnews.net/storyview.asp?storyid=24725. Accessed 10th October 2005.

Hay, M P and Rule, C M. (2003), SUPASIM: a floatation plant design and analysis methodology, in *Minerals Engineering*, Vol. 16, Elsevier, pp1103-1109.

Hay, M P and Schroeder, G. (2005), Use of the SUPASIM floatation model in optimising Impala's UG2 circuit, in *Minerals Engineering*, Vol. 18, Elsevier, pp772-784.

Hay, M P. (2009), *E-mail communication* through G.Njowa at Venmyn.

Hey, C. (2003), In-house benchmarking of ore ton build-ups, Company Internal Memorandum, Anglo Platinum, September, 2003.

Impala Platinum Mining Projects (2003), Basic Geology for Production Officials, Company Internal Report, revised January 2003.

Impala Platinum (2007), Extracts - Mining Standards, Procedures, and Code of Practices, A Pocket Reference Book, Mat.Cat.No. 773 715 921, revised 1st August 2007, Company Internal Document.

Impala Platinum (2008), Capital UG2 Phase 2: Impala 11#, Company Internal Document, 28th February 2008.

Impala Review Team (2009), Feedback on PhD research presentation by Cuthbert Musingwini to Impala Platinum on Friday 10th July in the 16# Boardroom at 11:00am, a feedback report included as part of the appendix of this thesis.

Institute of Directors in Southern Africa, IOD (2002), Executive Summary of the King Report 2002, A Report by the IOD, Johannesburg, South Africa, March 2002.

Jager, A J and Ryder, J A. (eds) (1999), *A Handbook on Rock Engineering Practice for Tabular Hard Rock Mines*, The Safety in Mines Research Advisory Committee (SIMRAC), Johannesburg, South Africa.

Jagger, L. (1999), Shallow platinum mining in South Africa, in *Proceedings School on Narrow Tabular Orebody Mining*, 21-22 October 1999, Mintek Randburg, South African Institue of Mining and Metallurgy, 17p.

Jagger, L A. (2006), The implementation of Nos. 16 and 20 shaft projects, in *Proceedings of the Second International Platinum Conference: 'Platinum Surges Ahead'*, 8th – 12th October 2006, Sun City, South Africa, The Southern African Institute of Mining and Metallurgy, pp267-276.

Jawed, M. (1995), Systems approach to production planning of underground coal mines under multiple objective environment, in *Proceedings APCOM XXV Conference*, Brisbane, 9-14 July 1995, pp551 – 553.

Jiyana, T (2009), Change management processes and ergonomics necessary for the implementation of in-stope roof bolting and safety nets at Turffontein shaft, a final year project report, School of Mining Engineering, University of Witwatersrand, June 2009.

Jonhson, A (2007), Mineral Resources Manager, Anglogold Ashanti. *Personal communication* at University of Witwatersrand, 2007.

Johnson, R C and McCarthy, R. (2001), Essential elements and risk in Bankable Feasibility Studies for mining transactions, paper presented at Parsons Behle and Latimer, Salt Lake City, Utah, March 2001, Available on INTERNET http://www.pdac.ca/pdac/publications/papers/2001/pdf/Johnson(T-17).pdf. Accessed 28th February 2009.

Kajner, L and Sparks, G. (1992) Quantifying the value of flexibility when conducting stochastic mine investment analysis, in *CIM Bulletin*, Vol. 85, No. 964, pp68-71.

Karadogan, A., Kahriman, A. and Ozer, U. (2008) "Application of fuzzy set theory in the selection of underground mining method", in *The Journal of the Southern African Institute of Mining and Metallurgy*, Vol. 108, No. 2, February 2008, pp73-79.

Kardi, T. (2006), Analytic Hierarchy Process (AHP) Tutorial, Available on INTERNET http://people.revoledu.com/kardi/tutorial/AHP/Paired-Comparison.htm. Accessed 15th February 2008.

Kazakidis, V N, Mayer, Z and Scoble, M J. (2004), Decision making using the analytic hierarchy process in mining engineering, in *Transactions of the Institution of Mining and Metallurgy: Mining Technology*, Vol. 113, March 2004, ppA30-A42.

Kazakidis, V and Scoble, M. (2003), Planning for flexibility in underground mine production systems*, Mining Engineering*, Amer. Inst. Min. Metall. Pet. Eng., SME, Vol. 55, No 8, August 2003, pp33-39.

Keeney, R L. (1992), *Value Focused Thinking: A Path to Creative Decision Making*, Harvard University Press, Cambridge, Massachusetts.

Kirk R L (1997), Use of the exploration shaft to reduce development time at Palabora, in *Proceedings Colloquium on Technology for Rapidly Accessing the Ore Body*, 30th July 1997, The South African Institute of Mining and Metallurgy, Randburg.

Lanham, A. (2004), Northam makes the breakthrough, in *Mining Weekly*, Vol 10 No. 43, 5th – 11th November, 2004, pp17-20.

Lanham, A. (2005), Good progress on R4.5bn shaft projects, in *Mining Weekly*, Vol. 11, No. 48, 9th – 15th December, 2005, pp20-27.

Lanham, A. (2006). Nprtham clears a major hurdle, in *Modern Mining*, Vol. 2, No.6, June 2006, pp18-22.

Lawrence, D R. (1984), The optimisation of raise-connection spacing in gold mines, in *Journal of the South African Institute of Mining and Metallurgy*, Vol. 84, No. 1, January 1984, pp10-20.

Lewis, R S. (1941), *Elements of Mining*, 3rd Edition revised by Clark, G.B. (1964), John Wiley and Sons, New York.

Linstone, H A, Turoff, M and Helmer, O. (eds.) (2002), *The Delphi Method: Techniques and Applications*, INTERNET http://www.is.njit.edu/pubs/delphibook/delphibook.pdf. Accessed 15th February 2008.

Lionnet, M D and Lomberg, K G. (2006), The application of facies classification in evaluating the Merensky Reef at Bafokeng Rasimone platinum mine, South Africa, in *Proceedings of the Second International Platinum Conference: "Platinum Surges Ahead'*, 8th – 12th October 2006, Sun City, South Africa, pp29-34.

Liqun, Z, Shihui, L, Lianfu, Z and Lianming, J. (1995), The analysis and practice of multi-objective decision-making technique for selecting a mining plan, in *Proceedings APCOM XXV Conference*, Brisbane, 9-14 July 1995, pp255 - 259.

Lizotte, Y and Elbrond, J. (1985), Optimal layout of underground mining levels, in *CIM Bulletin*, Vol. 78, No. 873, January 1985, pp41-48.

Macfarlane, A. (1997), Development methodology for deep level rapid reef access, in *Proceedings Colloquium on Technology for Rapidly Accessing the Ore Body*, 30 July 1997 (The South African Institute of Mining and Metallurgy: Randburg).

Macfarlane, A S. (2005), Establishing a new metric for mineral resource management, in *Proceedings of the First International Seminar on Strategic versus Tactical Approaches in Mining*, Johannesburg, 19-21 September 2005, pp179-203.

Mahlangu, T. (2007), Senior Lecturer: Hydrometallurgy, Department of Materials Science and Metallurgical Engineering, University of Pretoria, *Personal communication*.

McCarthy, P L. (2002), Production scheduling, AMC Reference Library. INTERNET http://www.amcconsultants.com.au/Members/PDF_Papers/MP/Production_Scheduling.pdf. Accessed 20th February 2004.

McCarthy P L and Livingstone R (1993), Shaft or decline? An economic comparison, in *Proceedings Open Pit to Underground: Making the Transition, AIG Bulletin 14*, 1993, pp45-56.

McIntosh Engineering (2000), *Hard Rock Miner's Handbook,* INTERNET. http://www.mcintoshengineering.com/Hard%20Rock%20Handbook/hardrock.htm. Accessed 28th March 2004.

Mineral and Petroleum Resources Development Act (MPRDA), No. 28 of 2002, Government Gazette, Vol. 448, Cape Town, 10 October, 2002.

Mitchell, C. (2009), Comment made during presentation to Anglo Platinum on 13th July 2009 at Anglo Platinum Head Office, 55 Marshall Street.

Mohloki, L. (2007), Projects Manager, Impala Platinum Projects, *Personal communication* during a technical site visit to Impala Platinum Projects, 4th – 14th December 2007.

Moxham, K A. (2004), A hard rock narrow reef mining machine – ARM 1100, in *Proceedings of the First International Platinum Conference: 'Platinum Adding Value'*, The South African Institute of Mining and Metallurgy, 2004, pp207-214.

Musingwini, C, Minnitt, R C A, Phuti, D and Makwara, F. (2003), Just-in-time development model for a sub-level caving underground mine in Zimbabwe, in *The Journal of The South African Institute of Mining and Metallurgy*, Vol 103, No.4, April 2003, pp187-191.

Musingwini, C. (2004), Some thoughts on Strategic Development Planning for 'No-Surprises-Mining', in *Proceedings of Orebody Modelling and Strategic Mine Planning*, Perth, 22nd – 24th November 2004, pp59-62.

Musingwini C, Minnitt R C A and Woodhall M (2006), Technical operating flexibility in the analysis of mine layouts and schedules, in *Proceedings of the 2nd International Platinum Conference – "Platinum Surges Ahead"*, Sun City, South Africa, 8th-12th October 2006, The Southern African Institute of Mining and Metallurgy, pp159-164. An electronic copy of the paper is available on INTERNET http://www.platinum.org.za/Pt2006/Papers/159-164_Musingwini.pdf.

Musingwini C, Minnitt R C A and Woodhall M (2007), Technical operating flexibility in the analysis of mine layouts and schedules, in *Journal of the Southern African Institute of Mining and Metallurgy*, Vol. 107, No. 2 pp129-136. A copy of the paper is available on INTERNET http://www.saimm.co.za/Publications/downloads/v107n02p129.pdf.

Musingwini, C and Minnitt, R C A. (2008), Ranking the efficiency of selected platinum mining methods using the analytic hierarchy process (AHP), in *Proceedings of the Third International Platinum Conference 'Platinum in Transformation'*, The Southern African Institute of Mining and Metallurgy, 2008, pp319-326.

Musingwini, C, Montaz, A and Dikgale, T. (2009), A linear programming and stochastic analysis of mining replacement rate for typical Bushveld Complex platinum reef conventional mining under variable geological losses, a paper accepted to the *Eighteenth International Symposium on Mine Planning & Equipment Selection (MPES 2009) Eleventh International Symposium on Environmental Issues and Waste Management in Energy and Mineral Production (SWEMP 2009)*, November 16th – 19th, 2009, Banff, Alberta, Canada.

Nilsson, D. (1998), Chapter 40: Planning economics of sublevel caving, in *Techniques in Underground Mining: Selections from Underground Mining Methods Handbook*, Gertsch, R.E. and Bullock, R.L. (eds), Society for Mining Metallurgy, and Exploration (SME), Colorado, pp725-740.

Njowa, G. (2006), Historical perspective, current industry practices and the latest developments in public reporting of mineral assets and properties for the extractive industry in South Africa, *MSc Dissertation*, University of Witwatersrand.

Nkosi, M D, Kruger, Y and Tyobeka, P. (2006), Six Sigma Project Closure Report: Project Proj-174, Ideal Ore Reserve Replacement Ratio – Immediate Focus Area: EXCESS SPD (Stope Preparation Drive) Development and the influence on the Ideal Replacement Ratio, January 2006. Company Internal Report.

Northam (2006), *Northam Annual Report 2006*, Northam Mine.

Northam (2007), *Northam Annual Report 2007*, Northam Mine.

Northam (2009), Presentation to University of Witwatersrand, School of Mining Engineering 4th Year Mining Engineering students at Northam mine on 25th June 2009.

Peters, M. and Zelewski, S. (2008), Pitfalls in the application of analytic hierarchy process to performance measurement, in *Management Decision*, Vol. 46 No. 7, pp1039-1051. Full text available on subscription to Emerald on INTERNET http://www.emeraldinsight.com/0025-1747.htm. Accessed 18th April 2009.

Pickering, R G B. (2007a), Sandvik Mining and Construction and Narrow Reef Mechanisation, presentation to University of Witwatersrand School of Mining Engineering, 10th May 2007.

Pickering, R G B. (2007b), Presidential address: Has the South African narrow reef mining industry learnt how to change?, in *The Journal of The Southern African Institute of Mining and Metallurgy*, vol 107 No.9 September 2007, pp557-565.

Pickering, R, Smit, A, and Moxham, K. (2006), Mining by rock cutting in narrow reefs, in *Proceedings of the Second International Platinum Conference: 'Platinum Surging Ahead'*, The South African Institute of Mining and Metallurgy, pp221-230.

Pincock (2004), Feasibility Study does not mean Feasible, *Pincock Perspectives*, Issue No. 57, September 2004, Available on INTERNET http://www.pincock.com/perspectives/Issue57-Feasibility-Study.pdf. Accessed on the 28th February 2009.

Pincock (2008), The processing of platinum group metals (PGM) – Part 1, in *Pincock Perspectives*, Issue No.89, March 2008.

Ragoonanthun, N. (2003), Winning Paper: Optimisation of a raiseline at Mponeng Mine, in *The Journal of The South African Institute of Mining and Metallurgy*, Vol. 103, No. 2, March 2003, pp79-85.

Ralph Morris and Associates (1994), Mimosa Platinum Project - Zimbabwe, a report prepared for Zimasco, November 1994.

Research Channel Africa (2009), Platinum Mining in South Africa: May 2009, a report available on subscription at INTERNET http://www.researchchannel.co.za. Accessed 18th June 2009.

Rogers, M. (2005), Head: Mine Technical Services, Anglo Platinum, *Personal communication*, at Anglo Platinum Head Office, 55 Marshall Street.

Rogers, M. (2009), Comment made during presentation to Anglo Platinum on 13th July 2009 at Anglo Platinum Head Office, 55 Marshall Street.

Ruffini, A. (2005), South African gold is not a sunset industry, in *Engineering & Mining Journal (E&MJ)*, Vol. 206, No 8, October 2005, p80.

Rupprecht, S M (2006), Selecting an appropriate decline development advance rate, in *The Journal of The Southern African Institute of Mining and Metallurgy*, Vol. 106 No.12, December 2006, pp845-848.

Saaty, T L. (1980), *The Analytic Hierarchy Process*, McGraw Hill International, New York, 1980.

Saaty, T L. (2008), *Decision Making for Leaders: The Analytic Hierarchy Process for Decisions in a Complex World*, RWS Publications, Pittsburg, USA.

Saaty, T L and Ozdemir, M S. (2003), Why the magic number seven plus or minus two, in *Mathematical and Computer Modelling*, Vol. 38, pp233-244.

Samis, M and Poulin, R. (1998), Valuing management flexibility: A basis to compare the standard DCF and MAP valuation frameworks, in *CIM Bulletin*, Vol. 91, No. 1019, pp69-74.

Schoor, van M and Vogt, D. (2004), Geophysical techniques and their application to mechanized mining, in *Proceedings Colloquium: Mechanized Mining and Support in the Bushveld Complex*, The South African Institute of Mining and Metallurgy, Rustenburg, 12th May 2004.

Searcy, D L. (2004), Aligning the balanced scorecard and a firm's strategy using the Analytic Hierarchy Process, in *Management Accounting Quarterly*, Summer 2004, Vol. 5, No. 4, pp1-10.

Singh, N, Urcan, H, Naidoo, K, Ryder, J, Watson, B P, Milev, A M and Roberts, M K C. (2005), The influence of pillars on the Merensky Reef horizon on stoping operations on the underlying UG2 Reef horizon, in *The Journal of The South African Institute of Mining and Metallurgy*, Vol. 105 No.6 July 2005.

Singh, N. and Vogt, D. (2006), Strategic and tactical research needs to ensuring a sustained contribution from mining to South Africa's economy, in *Proceedings of the 2nd International Seminar on Strategic versus Tactical Approaches in Mining*, Perth, 8th – 10th March 2006, ppS12.1-S12.16.

Smith, G L and Vermeulen, A. (2006), Strategic vs tactical approaches to mine planning-evolution and application at Anglo Platinum, in *Proceedings of the 2nd International Seminar on Strategic versus Tactical Approaches in Mining*, Perth, 8th – 10th March 2006, ppS9.1-S9.11.

Smith, M L and O'Rourke, A. (2005), The connection between production schedule and cut-off optimization in underground mines, in *Proceedings Application of Computers and Operations Research in the Mineral Industry (APCOM 2005)*, Tucson, USA, 30 March-1 April, 2005, pp643-654.

Snowden (2009), Fair market value of exploration properties, in *Snowden Newsletter*, Issue 55, July 2009, pp1-2.

South African Mineral Resource Committee (2007), South African Code for the reporting of exploration results, mineral resources and mineral reserves (The SAMREC Code) April 2007.

Stacey, T R. (2006), Design – A strategic issue, in *Proceedings of the 2nd International Seminar on Strategic versus Tactical Approaches in Mining*, 8th-10th March, 2006, Perth, Australia, ppS4.1-S4.13.

Stacey, T R, Terbrugge, P J, and Wesseloo, J. (2007), Risk as a rock engineering design criterion, in *Proceedings of the Challenges in Deep Level Mining*, Potvin, Hadjigeorgiou and Stacey (eds), Australian Centre for Geomechanics, 2007, pp17-23.

Steffen, O K and Rigby, N. (2005), The value of strategic mine planning, in *Proceedings of the First International Seminar on Strategic versus Tactical Approaches in Mining*, Johannesburg, 19-21 September 2005, pp97-120.

Stilwell, L C and Minnitt, R C A. (2006), Is platinum paying its rent?, in *Proceedings of the Second International Platinum Conference: 'Platinum Surges Ahead'*, 8th -12th October 2006, Sun City, South Africa, The Southern African Institute of Mining and Metallurgy, pp285-293.

Storrar, C D. (1977), *South African Mine Valuation*, Chamber of Mines of South Africa, Johannesburg.

Swanepoel, E. (2002), Mining in the Eastern Complex, in *Papers and Discussions 1997-2001*, Association of Mine Managers of South Africa and Chamber of Mines of South Africa, pp375-403.

Swart, A H and Handley, M F. (2005), The design of stable stope spans for shallow mining operations, in *The Journal of The South African Institute of Mining and Metallurgy*, Vol 105, No. 4, April 2005, pp275-286.

Taylor, H K. (1977), Mine Valuation and Feasibility Studies, Mineral Industry Costs, in Hoskins, J R and Green W R. (eds), *Mineral Industry Costs*, Northwest Mining Association, Spokane, WA, pp1-17.

Taylor, H K. (1986), Rates of working of mines – a simple rule of thumb, a technical note in *Transactions of the Institution of Mining and Metallurgy (Section A: Mining Industry)*, October 1986, ppA203-A204.

The New Collins Dictionary and Thesaurus (1989), William Collins Sons & Co Ltd, Glasgow, 1173pp.

Uysal, Ö and Demirci, A. (2006), Shortwall stoping versus sub-level longwall caving-retreat in Eli coal Fields, in *The Journal of the Southern African Institute of Mining and Metallurgy*, Vol. 106, No. 6, June 2006, pp425-432.

van den Berg, A A. (2007), Estimating the operating cost for a conventional platinum operation using activity based costing, a project report submitted to the School of Mining Engineering, University of Witwatersrand, August 2007.

Vermeulen, A. (2006), Senior Divisional Mining Engineer, Anglo Platinum Mine Technical Services, *Personal communication* during visit to Anglo Platinum Offices on 55 Marshall Street.

Vermeulen, A. (2009), Senior Divisional Mining Engineer, Anglo Platinum Mine Technical Services (MTS). *E-mail communication* on feedback on presentation of research findings to Anglo Platinum by C. Musingwini on 13th July 2009, e-mail dated 27th July, 2009.

Vieira, F M C C, Diering, D H and Durrheim, R J. (2001), Methods to mine the ultra-deep tabular gold-bearing reefs of the Witwatersrand Basin, South Africa, in *Underground Mining Methods: Engineering Fundamentals and International Case Studies*, Hustrulid W A and Bullock R L (eds.), Society for Mining Metallurgy and Exploration (SME), pp 691-704.

Vieira, F M C C. (2003), Utility-based framework for optimal mine layout selection, subject to multiple-attribute decision criteria, in *Proceedings of the 31st International Symposium on Application of Computers and Operations Research in the Minerals Industries*, Cape Town, 14-16 May 2003, pp133-149.

Vieira, F M C C. (2004), Rock engineering-based evaluation of mining layouts applicable to ultra-deep, gold bearing, tabular deposits, *PhD Thesis*, University of the Witwatersrand, Johannesburg.

Vieira, F M C C. (2005), An integrated, multi-disciplinary evaluation of ultra-deep layouts, in *Proceedings Application of Computers and Operations Research in the Mineral Industry (APCOM 2005)*, Tucson, USA, 30th March-1st April, 2005, pp655-665.

Walraven, F, Armstrong, R A and Kruger, FJ. (1990), A chronostratigraphic framework for the north-central Kaapvaal Craton, the Bushveld Complex and Vredefort structure, in *Technophysics*, Vol. 171, pp23-48.

Watson, B P. (2004), A rock mass rating system for evaluating stope stability on the Bushveld platinum mines, in *The Journal of The South African Institute of Mining and Metallurgy*, Vol 104, No.4, May 2004.

Watson, B P, Mosomane, S M, Kuijpers, J S, and Roberts, D P. (2008a), Investigations into the residual strength of a 2.5m wide Bushveld Merensky Reef crush pillar, in *The Journal of The Southern African Institute of Mining and Metallurgy*, Vol 108, No.8, August 2008, pp473-480.

Watson, B P, Ryder, J A, Kataka, M O, Kuijpers, J S, and Leteane, F P. (2008b), Merensky pillar strength formulae based on back-analysis of pillar failures at Impala Platinum, in *The Journal of The Southern African Institute of Mining and Metallurgy*, Vol 108, No.8, August 2008, pp449-461.

Woodhall, M. (2002), Managing the mining life cycle: mineral resource management in deep level gold mining, *MSc Dissertation*, University of the Witwatersrand, Johannesburg.

Woodhall, M. (2005), Planning? What planning?, in *Proceedings of the First International Seminar on Strategic versus Tactical Approaches in Mining*, Johannesburg, 19-21 September 2005, pp87-96.

Wu, H, Yuan, J, Zhang, Y, and Song, S. (2007), The evaluation of the core competition of the Wugang Mining Cooperation using the Analytic Hierarchy Process, in *The International Journal of Mineral Resources Engineering*, Vol. 12, No. 2, April-June 2007, pp119-126.

Yavuz, M. (2007), Analytical hierarchy process for selection of roadheaders, by O. Acaroglu, H. Ergin and S. Eskikaya in the Journal of SAIMM, Vol. 106, No. 8, pp569-575, a comment in *The Journal of the Southern African Institute of Mining and Metallurgy*, Vol 107, No. 2, pp138.

Yavuz, M. and Pillay, S. (2007a), Mining method selection by multiple criteria decision making tool by M.R. Bitarafan and M. Atei in the *Journal of SAIMM*, Vol. 104, No.9, pp493-498, a comment in *The Journal of the Southern African Institute of Mining and Metallurgy*, Vol 107, No. 2, pp137.

Yavuz, M and Pillay, S. (2007b), EQS: a computer software using fuzzy logic for equipment selection in mining engineering by A. Başçetin, O. Öztaş and A.I. Kanli in the Journal of SAIMM, Vol 106, No.1, pp63-70, a comment in *The Journal of the Southern African Institute of Mining and Metallurgy*, Vol 107, No. 2, pp138.

York, G. (1999), The role and design of pillar systems in tabular mines, in *Proceedings of School on Narrow Tabular Orebody Mining*, Randburg, 21st – 22nd October 1999.

Young, G J. (1923), *Elements of Mining*, 2nd Edition, McGraw-Hill Book Company, New York, 671p.

Zambó, J. (1968), *Optimum Location of Mining Facilities*, Akadémiai Kiadó, Budapest, 145p.

Zindi, L. (2007), Director - Greenfield Projects, Impala Platinum, *Personal Communication* during a technical site visit to Impala Platinum Mining Projects, 4th – 14th December 2007.

Zindi, L. (2008), No. 17 Shaft project – overcoming 'fourth generation' technical challenges, in *Proceedings of the Third International Platinum Conference: 'Platinum in Transformation'*, 6th - 9th October 2008, Sun City, South Africa, The Southern African institute of Mining and Metallurgy, pp375-385.

10 APPENDICES

10.1 Summary of paper abstracts and e-mail communications on the research

Parts of Chapters 3 and 4 were compiled and published as the paper: Musingwini C, Minnitt R C A and Woodhall M. (2006), Technical operating flexibility in the analysis of mine layouts and schedules, in *Proceedings of the 2nd International Platinum Conference – "Platinum Surges Ahead"*, Sun City, South Africa, 8th-12th October 2006, Southern African Institute of Mining and Metallurgy, pp159-164. An electronic copy of the paper is available on the internet at URL: http://www.platinum.org.za/Pt2006/Papers/159-164_Musingwini.pdf. The following is an extract of the paper abstract:

"Abstract

An often overlooked factor in the analysis of mine layouts and schedules is technical operating flexibility, mainly due to its nebulous nature. By glossing over technical operating flexibility the resultant mine layouts and schedules tend to be sub-optimal. The need to incorporate technical operating flexibility into the analysis and comparison of mine layouts and schedules is increasing in importance. This paper illustrates the nature of technical operating flexibility, reviews previous work on valuing of operating flexibility and proposes how technical operating flexibility can be quantified for tabular reef mines by using a platinum reef deposit as a case study. Once technical operating flexibility is quantified it becomes possible to explore its incorporation into the analysis of mine layouts and schedules and subsequent optimisation processes. The work described in this paper is part of a current PhD study at the University of the Witwatersrand.

Keywords: *Mine plans, layouts, schedules, technical operating flexibility, ore availability."*

After the paper was published in the *Proceedings of the Second International Platinum Conference: 'Platinum Surges Ahead',* a request was made from Canada, through the Southern African Institute of Mining and Metallurgy (SAIMM), to include part of the paper in an up-coming book on Real Options. The following is an extract of the e-mail detailing the request.

From: Nazli Mamdoo
To: Richard Minnitt; Cuthbert Musingwini;
CC:
Subject: FW: Requesting permission to use figure
Date: Thursday, February 14, 2008 9:42:52 AM
Attachments:

Dear Prof Minnitt

With regard to your paper "**Technical Operating Flexibility in the Analysis of Mine Layouts and Schedules**" that was published in the Platinum 2006 proceedings, please see the e-mail below regarding figure 6

Thank you

Yours sincerely

NAZLI MAMDOO
JOURNAL AND PUBLICATIONS COORDINATOR

-----Original Message-----
From: Charlotte Mosher [mailto:cmosher@laurentian.ca]
Sent: 12 February 2008 06:32 PM
To: nazli@saimm.co.za
Subject: Requesting permission to use figure

I am requesting your permission to reproduce the figure listed below from the following publication in my future contribution in:

Real Options in Engineering Design. Operations (edited by H.Back-Nembhard). CRC Press

---

Musingwini, C. Minnitt, R.C.A., Woodhall M., (2006), "Technical Operating Flexibility in the Analysis of Mine Layouts and Schedules", Int. Platinum Conf., Platinum Surges Ahead, The South African Inst. of Mining and Metallurgy, South Africa, pp. 159-164.

Figure 6

Thank you,

Charlotte Mosher/for/
Vassilios Kazakidis (vkazakidis@laurentian.ca)
Laurentian University
Ramsey Lake Road
Sudbury, ON P3E 2C6

The SAIMM subsequently invited a revised version of the same paper, subject to recommended changes by referees, for publication in the SAIMM journal. The revised version was published as the paper: Musingwini C, Minnitt, R C A and Woodhall, M. (2007), Technical operating flexibility in the analysis of mine layouts and schedules, in *Journal of the Southern African Institute of Mining and Metallurgy*, Vol. 107, No. 2 pp129-136. A copy of the paper is available from the SAIMM website at URL: http://www.saimm.co.za/Publications/downloads/v107n02p129.pdf. The following is an extract of the paper abstract:

"Abstract

An often overlooked factor in the analysis of mine layouts and schedules is technical operating flexibility (or tactical flexibility), mainly due to its nebulous nature. By glossing over technical operating flexibility the resultant mine layouts and schedules may be sub-optimal. The need to incorporate technical operating flexibility into the analysis and comparison of mine layouts and schedules is increasing in importance. The nature of technical operating flexibility is illustrated, previous work on valuing of operating flexibility reviewed and a proposal made on how technical operating flexibility can be quantified for tabular reef mines by using a platinum reef deposit as a case study. Once technical operating flexibility has been quantified it becomes possible to explore its incorporation into the analysis of mine layouts and schedules and subsequent optimisation processes. This paper is a revised version of a paper presented in the Proceedings of the Second International Platinum Conference "Platinum Surges Ahead" in 2006. The work described in this paper is part of a current PhD study at the University of the Witwatersrand.

Keywords: *Mine plans, layouts, schedules, technical operating flexibility/tactical flexibility, ore availability."*

After the revised paper was published there was an expression of interest from AngloGold Ashanti to apply the concept operating flexibility and proposed to use a revised form of the Flexibility Index (FI) to suit their mining layout which is Sequential Grid Mining (SGM) at Great Noligwa Mine. They want to link flexibility with safety because some of the accidents occur because 'there was inadequate operational flexibility to move workers to safer panels'. The e-mail extract below outlines the expression of interest:

From: Richard Minnitt [mailto:Richard.Minnitt@wits.ac.za]
Sent: 29 October 2007 10:40 AM
To: Johnson, Andre
Cc: Cuthbert Musingwini
Subject: RE: Mining Flexibility

Hi Andre

Nice to hear from you and yes I am well.

I would be very interested in meeting with you to discuss the work you are doing. In fact one of our staff members, Cuthbert Musingwini, is doing a PhD study related to this topic and I

am sure he would be interested in discussing it with you as well. I have copied him on this reply.

I'm not sure if you know that Mike Woodhall, and ex-AngloGold Ashanti man, who now works for GMSI, also did a Masters Research topic related to flexibility in mining operations.

I am free on Thursday or Friday this week if that suits you. We should also find out from Cuthbert what his availability is like.

Kind regards
Dick.

From: Johnson, Andre [mailto:AJohnson@AngloGoldAshanti.com]
Sent: 29 October 2007 09:44 AM
To: Richard Minnitt
Subject: Mining Flexibility

Dick,

All is fine with me, hoping to hear likewise.

I am currently involved with a project on developing a system to manage and measure mining flexibility for AngloGold Ashanti.

I understand you have been involved with similar work recently. If you are or have been involved, I am interested to share some ideas with you and exchange some thoughts on this subject.

Please let me know as soon as possible, for me to make an appointment

Regards,

Andre Johnson
Project Manager - Mining & MRM
AngloGold Ashanti Limited - AUR
Office +27 11 637 6655
Fax +27 11 637 6593
Mobile +27 82 827 8919
E-mail ajohnson@anglogoldashanti.com

Parts of Chapters 2 and 7 were compiled into the paper: Musingwini C and Minnitt, R C A. (2008), Ranking the efficiency of selected platinum mining methods using the analytic hierarchy process (AHP), in *Proceedings of the Third International Platinum Conference 'Platinum in Transformation'*, The Southern African Institute of Mining and Metallurgy, pp319-326. The following is an abstract of the paper:

"Abstract

The South African platinum mining industry is using a number of different mining methods, including variations of the same method, to extract the narrow reef tabular platinum deposits of the Bushveld Complex. These mining methods fall into three broad categories, namely conventional, mechanised and hybrid mining. A question sometimes asked in the industry is whether mechanized mining methods are more efficient than conventional mining methods. An objective answer requires the methods to be evaluated against multiple criteria simultaneously, whereby each criterion has a relative degree of importance in the overall decision. The most efficient method is the one that scores highest on each criterion. However, some of the criteria can be conflicting, such as by increasing dilution the shaft head grade decreases. The analytic hierarchy process (AHP) methodology was selected for the study because it is used to solve problems of this nature.

A survey was carried out to determine the relative importance of each efficiency criterion by drawing on the knowledge and experience of mine technical services and project management practitioners in the industry. Efficiency data for four different mining methods drawn from the Egerton (2004) study were used as a case study. The conventional mining method ranked as the most efficient mining method from the four methods considered. The exercise indicates potential for the AHP to be used in the South African platinum mining industry as a tool for selecting optimal layout designs, conduct regular evaluation of the performance of production shafts, or objectively evaluate line managers for promotion. The work described in this paper is part of the methodology used in a current PhD research study at the University of the Witwatersrand.

Keywords: *Analytic hierarchy process (AHP); multiple criteria decision analysis (MCDA); decision-making; conventional mining, hybrid mining, mechanized mining."*

The paper Musingwini C, Montaz A and Dikgale, T. (2009), A linear programming and stochastic analysis of mining replacement rate for typical Bushveld Complex platinum reef conventional mining under variable geological losses, was accepted for presentation at the *Eighteenth International Symposium on Mine Planning & Equipment Selection (MPES 2009) Eleventh International Symposium on Environmental Issues and Waste Management in Energy and Mineral Production (SWEMP 2009)*, November 16-19, 2009, Banff, Alberta, Canada. This paper is partly based on Chapters 3 and 5 and has the following abstract:

"Abstract

The Merensky and UG2 reefs of the Bushveld Complex in South Africa are the largest source of known platinum reserves in the world. Conventional, hybrid or mechanised mining methods are used to extract the platinum reefs. Conventional breast mining is the most prevalent mining method and is practiced in either of two variants namely the cross-cut or laybye access.

In conventional breast mining, development precedes stoping to demarcate the stopes. Mining replacement rate is the rate at which development generates new stopes to replace depleting ones thus, sustaining production. Financial wisdom demands deferring development as far as possible into the future because it is a cost. However, from an operational perspective, deferring development sacrifices operational flexibility. Inadequate flexibility leads to failure in meeting planned production targets or operating in inadequately prepared working areas that compromise safety. This problem is further compounded by geological losses in the form of potholes, dykes and faults whose exact location, extent and nature are never known with certainty prior to mining. Therefore it is imperative to carefully balance development and stoping rates by adopting an appropriate mining replacement rate. Existing operations use mining replacement rates based on empirical approaches. This paper presents a linear programming and stochastic analytical approach to explore mining replacement rate within the range 10% to 60% for geological losses typical of UG2 reefs.

This paper reports work on a current honours research project in the School of Computational and Applied Mathematics at the University of Witwatersrand (Wits) being pursued as a follow-up to a problem arising from part of a PhD research study in the Wits School of Mining Engineering which was presented to the 2009 Mathematics in Industry Study Group (MISG) in South Africa.

Keywords: *UG2 reef; conventional breast mining; cross-cut access; laybye access; mining replacement rate; pothole; dyke; fault; linear programming."*

10.2 Copies of AHP survey letter, questionnaire and responses

University
of the Witwatersrand,
Johannesburg

School of Mining Engineering

Private Bag 3, Wits 2050, South Africa • Fax: +27 11 339-8295 • Telephone: +27 11 717-7403

Date: .....

Dear Sir/Madam

Re: Survey on Decision Matrix for Tabular Reef Mines

There is a current research project being done in the School to establish a decision matrix for evaluating mine plans on tabular reef mines. By way of this letter you are being requested to participate in the survey. Please try to answer the questions as honestly as possible. Your contribution will be greatly valued.

Thank you.

C. Musingwini (Lecturer)

Mining Company: …….. **Date: ……..**

Please select the type of mining experience you have by ticking in the box next to it:

Tabular Gold		Tabular Platinum		Open pit		Underground		Other (specify)	

Please select a category that best describes your work by ticking in the box next to it:

Mine Planning		Mineral Resources Management		Geology		Production		Rock Engineering	

In comparing mine plans for narrow tabular reefs how much weight would you allocate to each of the following criteria? Please rank each criterion on a scale of 0-10. 0 means that the criterion is not considered at all and 10 that the criterion is very important. Also indicate whether the value of the criterion should be maximised or minimised. Under "Other Criteria" please include and give weight to any other criteria which you think are important but have not been listed. ***Please note that there is no right or wrong answer as all responses will be normalised.***

Criteria	**Maximise or Minimise criterion**	**Weight or Importance of criterion (on a scale of 0-10)**
Net Present Value [NPV] (R mil)		
Operating costs (R/t)		
Replacement ratio (m^2/m)		
Build-up or ramp-up period to full production (years)		
Extraction ratio (%)		
Productivity (m^2/employee/month or tonnes/employee/month)		
Ventilation costs (R/t mined)		
Life of Mine [LOM] (years)		
Life of raiseline (months)		
Operating flexibility (% additional capacity or production panels above the minimum required to meet target production rate)		
Rock engineering stability of excavations		
Dilution (%)		
Production rate (tonnes/month or m^2/month)		
Capital cost (R mil)		
Shaft head grade (g/t)		
Payback period (years)		
Other Criteria		

Based on your experience indicate your preference for each criterion over others in choosing the best mine plan, again on a scale of 0-10. For example, you may regard NPV to be 5 times more important than LOM in making your decision. To help you in the preference ranking consider for example, two mine plans, Plan A and Plan B, developed for the same deposit and have different NPV and LOM as given below.

Plan	**NPV (R mil)**	**LOM (years)**
A	300	10
B	500	15

Which plan is preferable, Plan A or Plan B? Give reasons.

.....

.....

.....

Now please complete the preference ranking table below, row by row, by ranking each criterion in a row against all other criteria in columns. Assume that the criteria will be either maximised or minimised as per your selection in the first table. The ranking is to show how many times a criterion in a row is more important than each of the criteria in columns. Your preference ranking can be fractional. For example if you consider productivity to be three times as important as extraction ratio in choosing the best mine plan, then you would enter a ranking of 3 in the cell marked by *. And if you consider production rate to be half as important as head grade then enter a ranking of ½ in the cell marked by **. If you think two criteria are equally important, then you would enter a ranking of 1. Some of the criteria may be interdependent on each other. For example you might find that by increasing dilution, the shaft head grade decreases. In such a case you can enter a ranking of 1. Please do not write anything in the grayed-out diagonal areas.

	NPV	Operating Costs	Replacement ratio	Build-up or ramp-up period	Extraction ratio	Productivity	Ventilation costs	LOM	Life of raiseline	Operating flexibility	Rock engineering stability	Dilution	Production rate	Capital Cost	Shaft head grade	Payback period	**Other Criteria**		
NPV																			
Operating Costs																			
Replacement ratio																			
Build-up or ramp-up period																			
Extraction ratio																			
Productivity					*														
Ventilation costs																			
LOM																			
Life of raiseline																			
Operating flexibility																			
Rock engineering stability																			
Dilution																			
Production rate															**				
Capital Cost																			
Shaft head grade																			
Payback period																			
Other Criteria																			

Please indicate any other comments here:

.....

.....

.....

.....

Thank you for your time.

+27 113735103

University
of the Witwatersrand,
Johannesburg

School of Mining Engineering

Private Bag 3, Wits 2050, South Africa • Fax: +27 11 339-8295 • Telephone: +27 11 717 7403

Date: 1st March 2007

Dear Sir/Madam

Re: Survey on Decision Matrix for Tabular Reef Mines

There is a current research project being done in the School to establish a decision matrix for evaluating mine plans on tabular reef mines. By way of this letter you are being requested to participate in the survey. Please try to answer the questions as honestly as possible. Your contribution will be greatly valued.

Thank you.

C. Musingwini (Lecturer)

Cuthbert.

Apology for delay. You can call me on (011) 373-6590 if my responses need clarification.

Peter Aylward
% Anglo Platinum, 55 Marshall St.

Mining Company: ANGLOPLATINUM **Date:** 1st March 2007

Please select the type of mining experience you have by ticking in the box next to it:

Tabular Gold		Tabular Platinum	✓	Open pit		Underground	✓	Other (specify)

Please select a category that best describes your work by ticking in the box next to it:

Mine Planning	✓	Mineral Resources Management	✓	Geology		Production		Rock Engineering	

In comparing mine plans for narrow tabular reefs how much weight would you allocate to each of the following criteria? Please rank each criterion on a scale of 0-10. 0 means that the criterion is not considered at all and 10 that the criterion is very important. Also indicate whether the value of the criterion should be maximised or minimised. Under "Other Criteria" please include and give weight to any other criteria which you think are important but have not been listed. ***Please note that there is no right or wrong answer as all responses will be normalised.***

— = neutral

Criteria	Maximise or Minimise criterion	Weight or Importance of criterion (on a scale of 0-10)
Net Present Value [NPV] (R mil)	—	8
Operating costs (R/t)	Max	8
Replacement ratio (m^2/m)	—	5
Build-up or ramp-up period to full production (years)	Max	9
Extraction ratio (%)	—	6
Productivity (m^2/employee/month or tonnes/employee/month)	max	9
Ventilation costs (R/t mined)	—	5
Life of Mine [LOM] (years)	—	8
Life of raiseline (months)	—	6
Operating flexibility (% additional capacity or production panels above the minimum required to meet target production rate)	max	8
Rock engineering stability of excavations	max	8
Dilution (%)	max	9
Production rate (tonnes/month or m^2/month)	—	8
Capital cost (R mil)	max	10
Shaft head grade (g/t)	max	6
Payback period (years)	—	5
Other Criteria		

Based on your experience indicate your preference for each criterion over others in choosing the best mine plan, again on a scale of 0-10. For example, you may regard NPV to be 5 times more important than LOM in making your decision. To help you in the preference ranking consider for example, two mine plans, Plan A and Plan B, developed for the same deposit and have different NPV and LOM as given below.

Plan	NPV (R mil)	LOM (years)
A	300	10
B	500	15

Which plan is preferable. Plan A or Plan B? Give reasons.
Plan B is preferable due to higher value (at same discount rate) and longer life is preferable to be able to amortise the initial capital over longer period. Also longer life implies more flexibility.

Now please complete the preference ranking table below, row by row, by ranking each criterion in a row against all other criteria in columns. Assume that the criteria will be either maximised or minimised as per your selection in the first table. The ranking is to show how many times a criterion in a row is more important than each of the criteria in columns. Your preference ranking can be fractional. For example if you consider productivity to be three times as important as extraction ratio in choosing the best mine plan, then you would enter a ranking of 3 in the cell marked by *. And if you consider production rate to be half as important as head grade then enter a ranking of ½ in the cell marked by **. If you think two criteria are equally important, then you would enter a ranking of 1. Some of the criteria may be interdependent on each other. For example you might find that by increasing dilution, the shaft head grade decreases. In such a case you can enter a ranking of 1. Please do not write anything in the grayed-out diagonal areas.

	NPV	Operating Costs	Replacement ratio	Build-up or ramp-up period	Extraction ratio	Productivity	Ventilation costs	LOM	Life of raiseline	Operating flexibility	Rock engineering stability	Dilution	Production rate	Capital Cost	Shaft head grade	Payback period	Other Criteria		
NPV		2	2	1	1	1	3	1	2	1	1	1	1	1	1	2			
Operating Costs	0.5		2	1	1	1	2	1	2	1	1	0.5	1	1	0,5	2			
Replacement ratio	0,5	0,5		1	1	1	2	1	2	1	1	0.5	1	0.5	0.5	2			
Build-up or ramp-up period	1	1	1		1	1	2	1	2	1	1	1	1	2	1	2			
Extraction ratio	1	1	1	1		1	2	1	2	1	1	1	1	1	1	2			
Productivity	1	1	1	1	1		2	1	2	1	1	1	1	1	1	2			
Ventilation costs	0,3	0,5	0,5	0,5	1	0,5		0,5	1	0,5	0,5	0,5	0,3	0,3	0,3	1			
LOM	1	1	1	1	1	1	2		2	1	1	0,5	1	1	0,5	2			
Life of raiseline	0,5	0,5	0,5	0,5	0,5	0,5	1	0,5		0,5	1	0,5	0,5	0,3	0,3	2			
Operating flexibility	1	1	1	1	1	1	2	1	2		1	0,5	1	1	1	2			
Rock engineering stability	1	1	1	1	1	1	2	1	1	1		1	1	1	1	2			
Dilution	1	2	2	1	1	1	2	2	2	2	1		1	1	1	2			
Production rate	1	1	1	1	1	1	3	1	2	1	1	1		1	*	2			
Capital Cost	1	1	2	0,5	1	1	1	1	3	1	1	1	1		1	2			
Shaft head grade	1	2	2	1	1	1	3	2	3	1	1	1	1	1		3			
Payback period	0,5	0,5	0,5	0,5	0,5	0,5	1	0,5	0,5	0,5	0,5	0,5	0,5	0,5	0,3				
Other Criteria																			

Please indicate any other comments here:
Would have been clearer to list definitions of the above measures, in case meanings differed in minds of respondents.

Thank you for your time.

Fax to C. Musingwini (011) 339 8295 +27 11 3735183

From: Mark Button, Snr Mining Engineer.

Mining Company: Anglo Platinum **Date:** 12-01-2007

(2 pages)

Please select the type of mining experience you have by ticking in the box next to it:

Tabular Gold	✓	Tabular Platinum	✓	Open pit	✓	Underground	✓	Other (specify)	Alluvial

Please select a category that best describes your work by ticking in the box next to it:

Mine Planning	✓	Mineral Resources Management		Geology		Production		Rock Engineering	

In comparing mine plans for narrow tabular reefs how much weight would you allocate to each of the following criteria? Please rank each criterion on a scale of 0-10. 0 means that the criterion is not considered at all and 10 that the criterion is very important. Also indicate whether the value of the criterion should be maximised or minimised. Under "Other Criteria" please include and give weight to any other criteria which you think are important but have not been listed. ***Please note that there is no right or wrong answer as all responses will be normalised.***

Criteria	Maximise or Minimise criterion	Weight or Importance of criterion (on a scale of 0-10)
Net Present Value [NPV] (R mil)	MAX	10
Operating costs (R/t)	MIN	8
Replacement ratio (m^2/m)	MAX	4
Build-up or ramp-up period to full production (years)	MIN	7
Extraction ratio (%)	MAX	6
Productivity (m^2/employee/month or tonnes/employee/month)	MAX	8
Ventilation costs (R/t mined)	MIN	3
Life of Mine [LOM] (years)	MAX	6
Life of raiseline (months)	MAX	8
Operating flexibility (% additional capacity or production panels above the minimum required to meet target production rate)	MAX	9
Rock engineering stability of excavations	MAX	10
Dilution (%)	MIN	8
Production rate (tonnes/month or m^2/month)	MAX	7
Capital cost (R mil)	MIN	5
Shaft head grade (g/t)	MAX	9
Payback period (years)	MIN	3
Other Criteria		
MECHANISATION	MAX	5

Based on your experience indicate your preference for each criterion over others in choosing the best mine plan, again on a scale of 0-10. For example, you may regard NPV to be 5 times more important than LOM in making your decision. To help you in the preference ranking consider for example, two mine plans, Plan A and Plan B, developed for the same deposit and have different NPV and LOM as given below.

Plan	NPV (R mil)	LOM (years)
A	300	10
B	500	15

1

+27 11 3735183

Which plan is preferable, Plan A or Plan B? Give reasons.

Plan B clearly as it has i) Higher NPV and ii) Longer LoM. However, if Plan A had the higher NPV, this would be prefered.

Now please complete the preference ranking table below, row by row, by ranking each criterion in a row against all other criteria in columns. Assume that the criteria will be either maximised or minimised as per your selection in the first table. The ranking is to show how many times a criterion in a row is more important than each of the criteria in columns. Your preference ranking can be fractional. For example if you consider productivity to be three times as important as extraction ratio in choosing the best mine plan, then you would enter a ranking of 3 in the cell marked by *. And if you consider production rate to be half as important as head grade then enter a ranking of ½ in the cell marked by **. If you think two criteria are equally important, then you would enter a ranking of 1. Some of the criteria may be interdependent on each other. For example you might find that by increasing dilution, the shaft head grade decreases. In such a case you can enter a ranking of 1. Please do not write anything in the grayed-out diagonal areas.

	NPV	Operating Costs	Replacement ratio	Build-up or ramp-up period	Extraction ratio	Productivity	Ventilation costs	LOM	Life of raiseline	Operating flexibility	Rock engineering stability	Dilution	Production rate	Capital Cost	Shaft head grade	Payback period	Other Criteria	Mechanism
NPV		2	5	1	5	3	5	3	5	4	2	2	3	5	2	2		4
Operating Costs			4	2	4	2	2	4	4	1	2	2	3	2	1	2		2
Replacement ratio				1/4	1/3	1/4	1	1/2	1	1/4	1/4	1/3	1/2	1/4	1/4	1/4		1
Build-up or ramp-up period					3	4	5	4	3	1	1	1	3	2	1	1		3
Extraction ratio						1/3	1	1	1/2	1/4	1	1/3	1/2	1	1/4	1/4		1
Productivity					*		3	3	2	1	1	1/2	2	3	1/3	1		3
Ventilation costs								1	1/3	1/5	1/4	1/5	1/3	1/2	1/5	1/4		1
LOM									1/2	1/4	1/4	1/4	1	1/2	1/5	1/4		1/3
Life of raiseline										1/3	1/4	1/5	1	1/4	1/5	1/4		1/3
Operating flexibility		Inverse of figures									1	1	2	3	1	1		3
Rock engineering stability		on RHS of matrix.										1	3	4	1	2		3
Dilution													3	3	1	2		4
Production rate														1	1/3	1/2		1
Capital Cost															1/4	1/2		2
Shaft head grade																3		4
Payback period																		4
Other Criteria																		
Mechanism																		

Please indicate any other comments here:

Gosh! I hope this all makes sense & adds up!

Thank you for your time.

IMPALA PLATINUM

FACSIMILE

Strictly Private & Confidential

To:	Dr CUTHBERT		
Company:	WITS UNIVERSITY		
Fax Number:	011 339 8295	Date:	08 NOV 2006
Sender:	LYDIA		
Number of Pages:	4 + COVER		
Subject:	SURVEY ON DEC MATRIX FOR TAB REEF MINES		

Confidentiality Note: This telefax contains information which is the property of the sender and intended for the use of the addressee named above . If the recipient of this telefax is not the intended recipient you are hereby advised that any copying, distribution or dissemination of this fax or use of the information contained herein is strictly prohibited. If you have received this telefax in error, please telephone the sender at the number indicated below to enable us to arrange for the retrieval of the transmission.

Good morning!

Kindly receive a fax from Mr Luke Zindi.

Thanks

Lyd.

Impala Platinum Limited
Reg. No. 1952/071942/06
Rustenburg Operations • P.O. Box 5683 • Rustenburg • 0300
Tel: +27 14 569 7568 • Fax: +27 14 569 7562 • lydia.kgokong@implats.co.za • www.implats.co.za

Directors: KC Rumble (Chief Executive Officer) • S Bessit; DH Brown; P Finney* ; CE Markus • LJ Paton
Secretary: R Mahadevey
*British

08 Nov 06 10:04 Impala Mining Projects 014 - 569 7562 p.1
014 569 7562

Mining Company: IMPALA PLATINUM Date: 30/10/2006

Please select the type of mining experience you have by ticking in the box next to it:

Tabular Gold	X	Tabular Platinum	X	Open pit	X	Underground	X	Other (specify)	

Please select a category that best describes your work by ticking in the box next to it:

Mine Planning	X	Mineral Resources Management		Geology		Production		Rock Engineering	

In comparing mine plans for narrow tabular reefs how much weight would you allocate to each of the following criteria? Please rank each criterion on a scale of 0-10. 0 means that the criterion is not considered at all and 10 that the criterion is very important. Also indicate whether the value of the criterion should be maximised or minimised. Under "Other Criteria" please include and give weight to any other criteria which you think are important but have not been listed. ***Please note that there is no right or wrong answer as all responses will be normalised.***

Criteria	Maximise or Minimise criterion	Weight or Importance of criterion (on a scale of 0-10)
Net Present Value [NPV] (R mil)	MAXIMISE	10
Operating costs (R/t)	MINIMISE	7
Replacement ratio (m³/m)	MAXIMIZE	8
Build-up or ramp-up period to full production (years)	MINIMIZE ~~Maximise~~	9
Extraction ratio (%)	MAXIMISE	8
Productivity (m³/employee/month or tonnes/employee/month)	MAXIMISE	9
Ventilation costs (R/t mined)	MINIMISE	9
Life of Mine [LOM] (years)	N/A ~~MAXIMISE~~	N/A
Life of raiseline (months)	MAXIMISE	9
Operating flexibility (% additional capacity or production panels above the minimum required to meet target production rate)	MAXIMISE	8
Rock engineering stability of excavations	MAXIMISE	10
Dilution (%)	MINIMISE	9
Production rate (tonnes/month or m²/month)	MAXIMISE	10
Capital cost (R mil)	MINIMISE	10
Shaft head grade (g/t)	MAXIMISE	8
Payback period (years)	MINIMISE	10
Other Criteria		

Based on your experience indicate your preference for each criterion over others in choosing the best mine plan, again on a scale of 0-10. For example, you may regard NPV to be 5 times more important than LOM in making your decision. To help you in the preference ranking consider for example, two mine plans, Plan A and Plan B, developed for the same deposit and have different NPV and LOM as given below.

Plan	NPV (R mil)	LOM (years)
A	300	10
B	500	15

08 Nov 06 10:05 Impala Mining Projects 014 - 569 7562 p.4
014 569 7562

Which plan is preferable, Plan A or Plan B? Give reasons.
INFORMATION LIMITED TO DECIDE WE NEED TO KNOW THE NPV
AT YEAR 10 OF PLAN B, IF THE RISK OF THE ASSUMPTION IS Revised
AFTER YEAR 10 THEN A IS FAVOURABLE. WE ALSO NEED TO CONSIDER
THE Reinvestment RATE TO SEE IF PROJECT B COMPLETES IN YEAR 10 AND

Now please complete the preference ranking table below, row by row, by ranking each criterion in a row against all other criteria in columns. Assume that the criteria will be either maximised or minimised as per your selection in the first table. The ranking is to show how many times a criterion in a row is more important than each of the criteria in columns. Your preference ranking can be fractional. For example if you consider productivity to be three times as important as extraction ratio in choosing the best mine plan, then you would enter a ranking of 3 in the cell marked by *. And if you consider production rate to be half as important as head grade then enter a ranking of ½ in the cell marked by **. If you think two criteria are equally important, then you would enter a ranking of 1. Some of the criteria may be interdependent on each other. For example you might find that by increasing dilution, the shaft head grade decreases. In such a case you can enter a ranking of 1. Please do not write anything in the grayed-out diagonal areas.

20 investment RATE IS FAVOURABLE THEN "A" CAN BE CHOSEN. PROJECT "B" IS ATTRACTIVE IN That THE Revenue ARE MAXIMISED

	NPV	Operating Costs	Replacement ratio	Build-up or ramp-up period	Extraction ratio	Productivity	Ventilation costs	LOM	Life of raiseline	Operating flexibility	Rock engineering stability	Dilution	Production rate	Capital Cost	Shaft head grade	Payback period	Other Criteria	
NPV		2	1	3	2	2	1	1	1	2	2	1	1	1	1	2		
Operating Costs	1		2	½	2	1	½	½	½	2	1	1	½	½	⅓	¼		
Replacement ratio	½	½		⅓	½	½	½	⅓	½	1	⅕	½	⅓	⅕	⅓	¼		
Build-up or ramp-up period	1	3	2		3	1	1	3	1	1	⅕	1	1	⅙	½	⅓		
Extraction ratio	½	2	2	⅓		½	⅓	⅕	½	1	⅕	⅓	⅕	¼	1	⅕		
Productivity	½	2	1	1	3		1	⅓	1	2	⅕	1	½	⅓	2	⅕		
Ventilation costs	1	2	2	1	2	1		⅓	1	1	⅕	1	½	¼	1	⅓		
LOM	1	4	3	3	5	3	5		3	2	⅕	½	1	1	2	1		
Life of raiseline	1	3	2	1	2	1	1	⅓		2	¼	1	⅓	½	1	1		
Operating flexibility	½	2	3	1	1	½	1	½	⅓		¼	3	½	1	1	1		
Rock engineering stability	½	5	6	4	3	2	2	5	5	4		2	3	1	1	1		
Dilution	½	2	2	1	2	1	1	2	1	⅓	½		½	⅓	2	⅓		
Production rate	1	1	1	2	4	1	1	1	2	2	⅓	2		1	1**	1		
Capital Cost	1	3	2	3	3	2	1	1	2	1	1	3	½		1	1		
Shaft head grade	1	2	1	2	3	1	2	½	1	1	1	½	1	1		1		
Payback period	2	1	2	1	1	1	1	1	1	1	1	3	2	1	1			
Other Criteria																		

Please indicate any other comments here:
NPV is an outcome so it should not be used as a primary parameter. LOM is also an outcome of available resource/ reserve and production rate.

Thank you for your time.

NPV could be compared against Pay back period.

08 Nov 06 10:05 Impala Mining Projects 014 - 569 7562 p.5
014 569 7562

10.3 Industry feedback comments

IMPALA PLATINUM

14 July 2009

University of Witwatersrand
School of Mining Engineering
Private Bag 3
WITWATERSRAND
2050

FEEDBACK ON PhD RESEARCH PRESENTATION BY CUTHBERT MUSINGWINI TO IMPALA PLATINUM ON FRIDAY 10th JULY IN THE 16# BOARDROOM AT 11:00am

This note serves to confirm that Cuthbert Musingwini made a presentation to Impala Platinum mining personnel on the 10th of July 2009. Seven personnel drawn from operations, technical services and senior management who attended the presentation to provide a critique on the research work were:

Luke Zindi	:	Project Director: Greenfields
Hans Baasden	:	Group Mining Engineer
Tinus Gericke	:	General Manager: Technical Services
Abbey Mkhuma	:	Mine Planner: Projects
Roux van Aarde	:	Impala Project Surveyor
Willie Knoetze	:	RSV Surveyor and
Niel de Bruin	:	Manager Geology: Projects

The following summarises comments from these practitioners on both the theoretical and practical aspects of the work.

Title of Presentation

"Techno-economic Optimisation of Level and Raise Spacing Range in Planning a Bushveld Complex Platinum Reef Conventional Breast Mining Layout"

Research Question

Is there an optimal range of level and raise spacing for a given Bushveld Complex platinum reef mine using conventional breast mining considering that current operations using conventional breast mining are planned on different combinations of level and raise spacing?

Impala Platinum Limited
Reg. No. 1952/071942/06
Rustenburg Operations • P.O. Box 5683 • Rustenburg • 0300
Tel: +27 14 569 000 • www.implats.co.za

Directors: DH Brown (Chief Executive Officer) • JC Andrews • S Bessit • D Earp • PD Finney* • B Molotlegi
LJ Paton • M Pooe • KP Seopela • J Theron
Secretary: A Parboosing
*British

Relevance of Research Study to the Platinum Mining Industry

Hans Baasden's Comments:

Level spacing and raise line spacing has been a controversial topic in the mining industry for decades. No two mining engineers will agree on this issue as there has been no way to scientifically calculate the best option. The only way available to mining engineers previously has been to laboriously model these variables manually, with no conclusive decisions.

The methodology demonstrated by Cuthbert has been able to quantify the issues and provide a model for optimising these two variables.

He is to be congratulated with his work and I strongly recommend that he publishes his findings as a paper for the Association of Mine Managers (AMMSA).

Tinus Gericke's Comments:

Tinus commented that reducing level intervals will result in considerable increase in capital expenditure and the development phase will be extended resulting in delays in production build up. This will result in negative impact to net present value contrary to the research findings.

He further commented that managing or smoothing of the tail production by decreasing level spacing to improve net present value is flawed because the tail production can not be accurately planned because it occurs at the end of the life of the project.

Niel de Bruin's Comments:

Niel would like to know if the model would be able to handle a project where the extraction rate is changed and the geology becomes more complex resulting in reduced extraction rate.

Roux van Aarde's Comments:

Roux commented that he believes that the research is relevant to platinum mining industry. He however points out that variability of the ore body could also prevent equidistant spacing of raise lines. He concurs with the research findings that smaller spacing can assist with quicker build up due to increased attack points. He however cautions that smaller mining blocks could negatively impact on productivity due to high frequency in re-establishing in complex geological setting. He proposes an approach of starting with reduced raise spacing during build up and then increasing the spacing at full production.

Abbey Mkhuma's Comments:

Abbey commented that the reason why there has been no satisfactory optimization approach regarding level and raise spacing was due to the fact that the decision has always been treated as if it was a mono criteria decision problem. He believes that the research findings are relevant to the narrow reef platinum ore bodies similar to Impala Rustenburg lease area.

Luke Zindi's Comments:

Management at Impala Rustenburg lease area adopted the conventional mining approach customized for depth for the future mining projects. It is reassuring to see the research findings confirming that conventional mining approach is going to be the predominant mining approach in the platinum industry. The research has also confirmed that the Impala lay bye access has more advantages than the cross cut access.

The writer agrees with the research statement that level and raise spacing form the basis of the life of mine planning, hence has significant impact on tactical and strategic issues.

The compromise on safety due to increase in line of sight has some merits based on the number of scraper related incidence experienced in the industry.

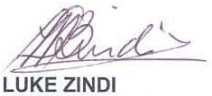
The results of the research clearly indicate an increase in value corresponding to a decrease in level and raise line spacing. It is proposed that further investigations to explore more data towards decreased level and raise line spacing be conducted. Similar investigation could be conducted on up-dip or down-dip mining configuration.

The research has clearly demonstrated that the problem can be resolved by applying a multi-criteria decision-making process. The methodology applied in the research is therefore appropriate. The research can be further enhanced by customizing it to handle mining constraints.

Conclusion:

In principle the Impala review team agrees that the methodology and findings are applicable to the narrow reef platinum setting. The methodology takes cognisance of the uniqueness of ore bodies by not providing "one size fits all" solution. Future engagement in using the methodology to develop a protocol which Impala can use for future feasibility studies will be investigated. The researcher has to take cognisance of the fact that the review team is basing its review on very limited information and await the completed research paper.

Yours sincerely



LUKE ZINDI
On behalf of the Impala Review Team

ANGLO PLATINUM

Anglo Platinum Limited

C Musingwini,
Lecturer,
Wits University.

14th July 2009

Dear Cuthbert

CRITIQUE OF PhD RESEARCH PRESENTATION BY CUTHBERT MUSINGWINI TO ANGLO PLATINUM ON MONDAY 13th JULY AT 14:00 IN ROOM 1406, 55 MARSHALL STREET

Preamble

This critique is on the presentation made to Anglo Platinum personnel by Cuthbert Musingwini on the 13th of July 2009 as part of the closure process for his PhD thesis. The presentation was titled "Techno-economic Optimisation of Level and Raise Spacing Range in Planning a Bushveld Complex Platinum Reef Conventional Breast Mining Layout". Cuthbert's research question was, "Is there an optimal range of level and raise spacing for a given Bushveld Complex platinum reef mine using conventional breast mining considering that current operations using conventional breast mining are planned on different combinations of level and raise spacing?". The presentation was attended by Mike Rogers (Head, Mine Technical Services), Clive Mitchell (Senior Mining Engineer), Mark Farren (Head of Mining), Ken Hanekom (Senior mining engineer), Udo Sachse (Senior Mining Engineer, and Phillip Tobias (General Manager). A summary of the key points discussed is outlined below.

Validity and Practical Application of the Research Work and Findings

This project was motivated well.
The stakeholders were consulted well.
The different analysis techniques were well sourced.
It is a very relevant study.
All the relevant issues were examined: back length; raise spacing; replacement ratios.
15 layouts were designed and scheduled.
Its conclusions were extremely important to our mines:

1. Back lengths should be 250m to 350m long.
2. Raises around 200m apart.
3. There is no significant value in reducing the raise spacing below 200m. In fact the NPV reduces if this is done.

In summary the research work is extremely valid and the application of the research work and findings is very relevant to our current mine optimisation requirements.

On behalf of the Anglo Platinum committee,

Clive H.D. Mitchell.

Clive HD Mitchell
Senior Mining Engineer.
Anglo Platinum Limited
55 Marshall Street Johannesburg 2001 P O Box 62179 Marshalltown 2107
Tel +27 11 373 6111 Fax +27 11 373 5111 Telegrams Anglo Platinum Johannesburg
Website www.angloplatinum.com

Incorporated in the Republic of South Africa Registration No. 1946/022452/06

Directors TMF Phaswana (Non-executive Chairman) TA Wixley (Deputy Chairman) NF Nicolau (Chief Executive Officer) CB Carroll** KD Dlamini RMW Dunne* BA Khumalo RJ King* R Médori*** MV Moosa B Nqwababa SEN Sebotsa
Alternates PG Whitcutt *(British) **(American) ***(French)
Company Secretary JD Meyer

A member of the Anglo American plc group

10.4 AMMSA programme for the 06th August 2009

AGENDA

ASSOCIATION OF MINE MANAGERS CENTRAL DISTRICT MEETING THURSDAY, 06 AUGUST 2009
ELAND PLATINUM MINE – XSTRATA ALLOYS

VENUE : MAGALIES PARK

TIME : 10h00 for 10h30

1. WELCOME
2. APOLOGIES
3. NEW MEMBERS
4. MINUTES OF PREVIOUS AMM MEETING
5. MINUTES OF PREVIOUS COUNCIL MEETING
6. MATTERS FROM PREVIOUS MEETINGS
7. PRESENTATIONS:
   - ► WATER RESOURCES MANAGEMENT IN THE MINING SECTOR
     Eland Platinum Mine as a case study

     By Dr Fanie Botha
   - ► ELAND PLATINUM MINE RESIDUE DISPOSAL FACILITY (MRDF)
     The integrated disposal of platinum tailings and waste rock into a single final engineered landform

     By Andrew Savvas
   - ► Techno-Economic Optimization of Level and Raise Spacing
     Range in planning a Bushveld Complex Platinum Reef Conventional Breast Mining Layout.

     By Cuthbert Musingwini (Lecturer at Wits)
8. CORRESPONDENCE
9. PAPERS ON OFFER
10. GENERAL
11. DATE AND VENUE OF NEXT MEETING
12. INDUSTRIAL RELATIONS MATTERS
13. UNUSUAL INCIDENTS OR INJURIES
14. CLOSURE
15. LUNCH (13:00)
16. SOCIAL ACTIVITY – GOLF (14:00)